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SEPTEMBER, 1916

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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Our extensive familiarity with the metallurgy of these metals makes us more than mere manufacturers. It equips us to advise with engineers in the determination of the very best apparatus and method of installation to meet the necessities of any particular problem.

Every engineer whose interests touch the production of copper and lead should have a copy of our smelter catalogue No. 12-F, which illustrates many furnaces and much other equipment used in copper and lead smelting. May we send it to you?

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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 117 SEPTEMBER 1916

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MANUSCRIPT CLOSING DATE FOR THE NEW YORK MEETING, 1917

The 114th (New York) Meeting of the Institute will be held in the third week of February, 1917. The Committee on Papers and Publications has set Dec. 1, 1916, as the closing date for the receipt of manuscripts intended for this meeting.

Manuscripts must be received by the Secretary on or before this date. If they are sent through members of the technical committee they must be sent to those committees enough in advance of the time to be forwarded and in the hands of the Secretary by the time specified.

While Dec. 1, 1916, is the final closing date, it is a great advantage to get papers in as much before that time as possible as it enables them to be submitted in proof form to their authors. Papers received on or before Nov. 1, 1916, will, as far as possible, be published in the January Bulletin or earlier, thus giving good distribution among the members in advance of the meeting.

The Institute's records show that the discussion of papers is very much better when they are distributed somewhat in advance of the meeting instead of in the Bulletin which goes out immediately preceding the meeting.

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CHARLES KIRCHHOFF.

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CHARLES KIRCHHOFF—IN MEMORIAM

Born, March 28, 1853 Died July 23, 1916

On July 25, 1916, the Board of Directors adopted the following minute and directed that it be entered upon the records of the Board and published in the Bulletin of the Institute, and that copies thereof be sent to the family of Mr. Kirchhoff:

Charles Kirchhoff was one of the oldest, most widely known and most highly esteemed members of the American Institute of Mining Engineers. On his return in 1875 from a European course of study, to begin a highly successful professional and business career, he joined the Institute, then just commencing to acquire both national and international recognition. He became at once, and for more than 40 years continued to be, its earnest and active supporter, serving it as Manager, Councilor or Director for terms aggregating 9 years, as Vice-President for 2 years, and twice as President for a year.

During the critical period of the erection of the United Engineering Society's building, and the transfer of the Institute to permanent headquarters therein, Mr. Kirchhoff was its efficient representative in the United Engineering Society, and on the Building Committee.

In 1910, amid the congratulations of colleagues, employees, and innumerable friends, Mr. Kirchhoff retired from active business as Vice-President and General Manager of a great technical publishing company. Yet he sacrificed of his well-earned leisure yet one more year, to serve the Institute as President through 1911.

The remainder of his life was spent in retirement; but, to those who were privileged to know him most intimately, his long, brave, silent, losing fight against a progressive and fatal disease was a not unworthy close to his useful and distinguished career.

Mr. Kirchhoff was one who secured, because he practiced, loyalty in friendship, justice and kindness in personal intercourse, openness and equity in business. Modest, but not timid; prudent, but progressive; gentle, but firm; full of chivalry and of common sense; indefatigably industrious and inexhaustibly patient, he won victories without making enemies, and crowned with a heroic death the record of a life unstained.

This Board extends to the widow and relatives of Mr. Kirchhoff its profound sympathy in their bereavement, assuring them that the entire membership of the Institute unites in this expression of gratitude for the life and labors, and sorrow in the loss, of an honored colleague and dear friend.

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ARIZONA MEETING, SEPTEMBER 18 to 25, 1916

For the first time in its history the Institute will hold a meeting in the State of Arizona.

A few years ago Arizona stood third in the copper-producing districts of the United States. Since that time, with the development of porphyry mines, the output has gone up with great rapidity until it not only is the leading district, but its output at the present time is at the rate of nearly double the Montana output, which now stands second in the list.

In connection with this development, most interesting departures have been made in methods of mining, concentration and smelting. Principles of mining and metallurgy of the greatest interest will be covered by papers which will appear in the *Bulletin* and be presented at Technical Sessions, while ample opportunity will be afforded for those who go to the meeting to see all the points of interest to them. In addition, members will be well rewarded by the marvelous natural features to be seen on the trip, including the petrified forests, the cliff dwellings on the Apache Trail and the Grand Canyon of Arizona.

It is planned to start the special train from New York, but members and guests may join the special at other points en route as indicated by the following schedule:

ITINERARY

Thursday, September 14th

Lv	New York, Grand Central Terminal	5.30 p. m.
	Dinner in dining car a la carte	

New England Members via B. & A. R. R.

Lv	Boston (Regular train)	2.00 p. m.
Lv	Worcester (Regular train)	3.12 p. m.
Lv	Springfield (Regular train)	4.37 p. m.
	Dinner in dining car a la carte	
Ar	Albany New York Central R. R.	7.45 p. m.
Lv	Albany	9.00 p. m.
Lv	Schenectady	9.32 p. m.
Lv	Utica	11.08 p. m.

Friday, September 15th

Lv	Syracuse	12.30 a. m.
Lv	Cleveland	7.25 a. m.
Lv	Toledo	10.05 a. m.
Ar	Chicago	4.00 p. m.
Lv	Chicago, C. R. I. & P.	8.05 p. m.

Saturday, September 16th

Lv	Kansas City, C. R. I. & P.	11.00 a. m.
Lv	Topeka, C. R. I. & P.	1.05 p. m.
Lv	Hutchinson	6.05 p. m.

Sunday, September 17th

Lv	Santa Rosa, E. P. & S. W. R. R.	6.30 a. m.
Ar	El Paso	2.40 p. m.
Lv	El Paso, Southern Pacific	10.20 p. m.

Monday, September 18th

Ar	Santa Rita	4.00 a. m.
	Visit mines of Chino Copper Co. and mill of Empire Zinc Co.	

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Lv Santa Rita, Santa Fe..... 2.00 p. m.
Ar Hurley..... 2.45 p. m.
Visit mill of Chino Copper Co.
Lv Hurley, Sante Fe..... 10.15 p. m.

Tuesday, September 19th

Ar Douglas..... 7.00 a. m.
Visit reduction works of Copper Queen Consolidated Mining Co., and of Calumet & Arizona Mining Co.
Technical Session on "Smelting."
Technical Session on "Leaching."
Lv Douglas, E. P. & S. W..... 11.00 p. m.

Wednesday, September 20th

Ar Bisbee, Arizona..... 1.00 a. m.
Visit to mines of district.
Technical Session, subject "Mining and Geology"
Lv Bisbee, E. P. & S. W..... 10.00 p. m.

Thursday, September 21st

Ar Globe..... 9.00 a. m.
Visit to mines and reduction works of Old Dominion Copper Co.
Technical Session, subject "Concentration and Flotation."
Technical Session, subject "Fine Grinding."

Friday, September 22nd (Globe)

Visit reduction works of International Smelting & Refining Co., and mills of The Inspiration Consolidated Copper Co., and the Miami Copper Co.
Technical Session, subject "Mining and Smelting."
(The Ray Consolidated Copper Co. will welcome members and guests who desire to visit the mine at Ray. Special arrangements must be made by omitting other features on the program and with the understanding that those who go to Ray must reach Phoenix in time to take the train from there.)

Saturday, September 23rd (Globe)

Lv Globe by automobile over Apache Trail, including visit to Roosevelt Dam.
Ar Phoenix about..... 5.30 p. m.
Lv Phoenix, Sante Fe..... 8.00 p. m.

THE APACHE TRAIL

History records that the ancient trail leading to the ruined cliff dwellings in the Salt River Valley was discovered in 1540 by Francisco Vasquez de Coronado, the Spanish explorer, in his search for the treasures of the "seven cities" of Cibola, the abode of the Zuni Indians—the "Fathers of the Pueblos." Later the trail was used by the Apache Indians and this wild and rugged region was one of their last retreats—the scene of desperate encounters with our troopers in the campaign of 1872. Today the modern motor car follows the same historic and romantic trail through scenery of unusual variety and beauty over the highway between Globe and Phoenix, a distance of 120 miles.

The building of the great Roosevelt Dam, which required the construction of the roadway through the wildest and most rugged sections of the route, has made possible this unique mountain tour, and the consequent transformation of the Salt River and Tonto Creek basins into a great lake region has added a charming aspect to views already grandly impressive.

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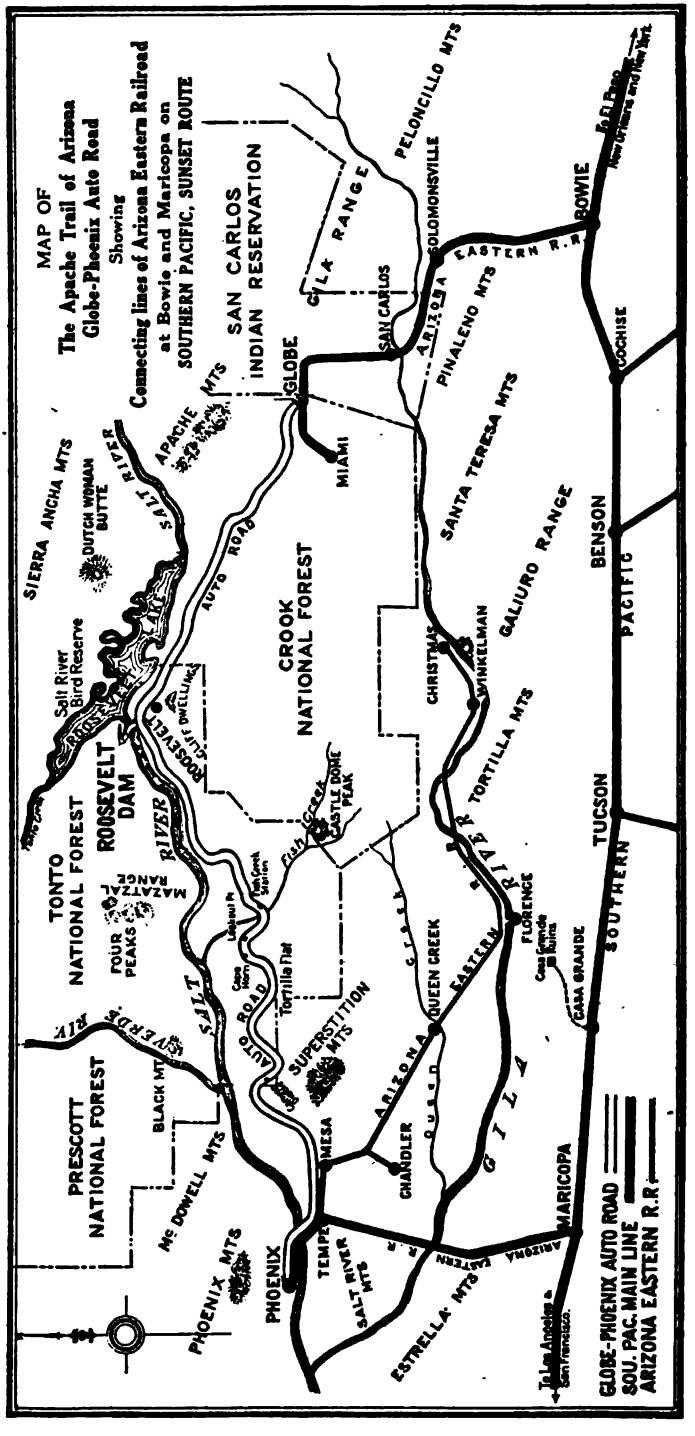

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CLIFF DWELLINGS.

ROOSEVELT DAM AND SOUTH SPILLWAY.

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CLIFF DWELLINGS

At a point four and one-half miles east of Roosevelt, a wagon trail leads up a small canyon to a point where a foot trail may be taken up the steep mountain side to a group of ancient cliff dwellings which are well worth visiting. Another and more extensive group of ruins can be reached two miles farther by trail.

ROOSEVELT DAM

This great dam has a maximum height of 280 ft. When the reservoir is full the water backs up fifteen miles in the Tonto and fifteen miles in Salt River. A hydro-electric plant located below the dam furnished current for use during the construction period and now provides power for irrigation pumping on the Salt River Project and for sale to private consumers.

The Salt River Valley is a land of perennial summer, of oranges, of cotton and of dates. Alfalfa is a staple, and ostrich farming is among the successful enterprises. Irrigation has hastened the valley's progress and confirmed its business stability. Phoenix is its metropolis, an energetic and attractive city, and the capital of the State.

Sunday, September 24th

Ar Grand Canyon..... 8.00 a. m.
Lv Grand Canyon, Sante Fe..... 7.40 p. m.

GRAND CANYON OF ARIZONA

The series of tremendous chasms which form the channel of the Colorado River in its course through northern Arizona, reach their culmination in a chaotic gorge, 217 miles long, from 9 to 13 miles wide, and more than 6,600 ft. below the level of the plateau.

Standing upon the brink of the plateau at the point of the canyon's greatest width and depth, the beholder is confronted by a scene whose majesty and beauty are indescribable. It is one of the few widely advertised spots which one need not fear approaching with anticipations too exalted. It has never been adequately described, and never will be.

The poem which follows this itinerary was written by Dr. Raymond in July, 1889, while he was camping with a party of friends near the Grand Canyon. The picture facing the poem, while it fails to show the succession of pillars and "temples" that may be seen from the brink of the Canyon, gives to some extent an idea of the great height and majesty of the walls. The poem and the picture were printed in the souvenir menu of the dinner given to Dr. Raymond by his friends in commemoration of his seventieth birthday.

Monday, September 25th

Lv Albuquerque, Santa Fe..... 7.00 p. m.

Tuesday, September 26th

Lv La Junta, Sante Fe..... 7.10 a. m.
Lv Kansas City, Sante Fe..... 11.00 p. m.

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Wednesday, September 27th

Ar Chicago	11.15 a. m.
Lv Chicago, New York Central	12.40 p. m.
Lv Cleveland, New York Central	7.35 p. m.

Thursday, September 28th

Ar Albany	6.27 a. m.
Ar Boston	11.55 p. m.
Ar Grand Central Terminal	9.40 a. m.

Fares quoted below include railroad and sleeping car fares for the entire trip. Fares from New York and Albany also include excess fares applying to Chicago and return owing to extra fast schedule of train.

For the exclusive use of a compartment by one person, one and one-half railroad tickets are required and for a drawing room, two tickets are required.

The arrangements as announced, also railroad and sleeping car fares shown below, are conditional upon one hundred fares New York to El Paso, one hundred twenty-five fares El Paso to Grand Canyon and one hundred fares Grand Canyon to New York.

Starting point	Lower 1 Person	Upper 1 Person	Compartment 2 Persons (Each)	Drawing Room 2 Persons (Each)
New York	$228.50	$217.50	$233.50	$261.00
Albany	216.60	205.60	221.60	249.10
Schenectady	208.96	197.96	213.96	241.46
Utica	205.84	194.84	210.84	238.34
Syracuse	203.84	192.84	208.84	236.34
Rochester	200.60	189.60	205.60	233.10
Buffalo	198.40	187.40	203.40	230.90
Cleveland	188.80	177.80	193.80	221.30
Toledo	186.30	175.30	191.30	218.80
Chicago	176.80	165.80	181.80	209.30
Kansas City	160.30	149.30	165.30	192.80

APPLICATIONS

Those desirous of joining the Tour should promptly communicate with Mr. W. V. Lifsey, Assistant General Passenger Agent, New York Central Lines, 1216 Broadway, New York City. Reservations will be made in the order in which applications are received.

Members of the Arizona Section desiring accommodations on the Special Train should communicate with Mr. Arthur Notman, Box 400, Bisbee, Ariz.

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THE GRAND CAÑON OF THE COLORADO.

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THE GRAND CAÑON

A thought of God, on earth expressed!
The silence of His perfect rest;
The patience of eternal power;
The ceaseless change from hour to hour;
Forms in alternate gloom and flame
That bide yet evermore the same,
And do but wear such fitful guise,
Reflected in our human eyes,
Which compass only in their range
The things that change or seem to change;
The blended hues of heavenly birth
Beyond the tenderest tints of earth,
That fill and flood her spaces wide
With surges of celestial tide;
The beauty of that awful brink
Where meaner thoughts in rapture sink,
And souls see clear, though eyes grow dim,
While space and time are lost in Him!

Methinks I could not faint or flee
In any conflict yet to be,
Whatever pathway must be trod,
Might I but keep this thought of God!

ROSSITER W. RAYMOND.

July, 1889.

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DR. HENRY STURGIS DRINKER

When the suggestion was made by Major General Leonard Wood that the Institute assist in the formation of a National Reserve Corps of Engineers, Dr. Drinker's name occurred first to the Board of Directors as the man who would be most capable of carrying out the proposed plan. Dr. Drinker has long been associated with work in connection with military training and is at present the Chairman of the Governing Committee on Military Training Camps Association which is having such important results at Plattsburg and other places in this country.

As Chairman of the Institute's Committee on National Reserve Corps of Engineers, Dr. Drinker was a member of the Joint Conference Committee of the National Engineering Societies on this subject. He was an earnest worker in the movement which has finally resulted in definite legislation by Congress whereby a Reserve Corps of Engineers is authorized in the new Army Bill which became effective July 1, 1916. Notice of the results of the work was sent to all the members of the National Engineering Societies early in July, and the War Department of the United States has recently issued a circular which has been sent to all the members of the National Engineering Societies, as well as to many other engineers, outlining the line whereby engineers may enroll themselves in the Officers' Reserve Corps. Dr. Drinker has, therefore, fulfilled the mission which he undertook when assuming the Chairmanship of the Institute's Committee. He now asks the permission of the Board of Directors to withdraw as Chairman and has requested that he be replaced in this position by Mr. Arthur S. Dwight, in the belief that the Chairmanship of the Institute's Committee and membership on the Joint Committee should be vested in a man located in New York City, for the better carrying out of the important work now before the Joint Committee. We are happy to say that Dr. Drinker has consented to remain a member of the Institute's Committee, but the completion of the first phase of his work, as Chairman, should not be allowed to pass without an expression of appreciation on behalf of the Institute for his services.

NOMINATIONS FOR OFFICERS

The suggestions of the members of the Institute are very much desired by the Committee on Nominations prior to deciding upon its nominations to fill the places of those officers who retire early in 1917. The Committee must make its final report to the Board of Directors early in October. The members are, therefore, asked to send in promptly their suggestions for Directors as well as for Vice-Presidents and President. The Committee hopes to hear from a large number of the members.

The officers to be elected at the annual meeting in February, 1917, are:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York

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City and district, which is provided for in the Constitution. (N.B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, L. D. Ricketts (not eligible for re-election).

Past President, Benjamin B. Thayer.

Vice-Presidents, H. C. Hoover, District 6; Joseph W. Richards, District 2.

Directors, Reginald W. Brock, District 11; Albert R. Ledoux, District 0; D. C. Jackling, District 7; Charles W. Merrill, District 6; Henry L. Smyth, District 1.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, 1208 Hollingsworth Bldg., Los Angeles, Cal.

SEELEY W. MUDD,
Chairman, Committee on Nominations.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JULY 25, 1916

A special meeting of the Board of Directors was held on July 25, 1916, pursuant to notice duly given according to the By-Laws.

This special meeting was for the purpose of giving authority for the signing of a contract with the American Society of Civil Engineers, as mentioned in another place in this Bulletin. The interest in the matter is indicated by the attendance of 11 members of the Board, some of whom came from a long distance.

An invitation was received from the St. Louis Section of the Institute and from members in and near St. Louis to hold the 115th Meeting of the Institute in and near St. Louis in late September or early October, 1917. This invitation was accepted.

Announcement was made that the Mexican Institute of Mining and Metallurgy had been dissolved and reorganized as the Mexican Section of the Institute.

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The sum of $250 was appropriated to the Nevada Section.
The usual election of members took place.
Resolutions were passed regarding the death of Charles Kirchhoff.

AMERICAN SOCIETY OF CIVIL ENGINEERS

The American Society of Civil Engineers, by a vote of its membership of six to one, has favored the acceptance of the invitation of the three present Founder Societies, namely, the American Society of Mechanical Engineers, American Institute of Electrical Engineers and American Institute of Mining Engineers, to become a fourth Founder Society and share with the rest the occupancy of the Engineering Societies' Building.

Fortified by this vote, the Board of Direction of the American Society of Civil Engineers has caused a contract to be entered into between that Society and the present Founder Societies and the United Engineering Society.

Work has already begun on adding three stories to the Engineering Societies' Building at a cost not to exceed $250,000. This cost will be borne by the American Society of Civil Engineers, thereby making its contribution approximately equal to that of the other Founder Societies.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period July 10, 1916 to August 1, 1916:

W. A. Barnes, San Francisco, Cal.
W. L. Bell, Davenport, Fla.
James P. Clarke, Jr., Little Rock, Ark.
R. A. Carhart, Mitchell, S. D.
G. M. Colvocoresses, Humboldt, Ariz.
W. E. Defty, Phoenix, Ariz.
Carl F. Dietz, Worcester, Mass.
W. J. Mead, Madison, Wis.
Charles Mentzel, San Pedro, N. M.
H. H. Miller, New York, N. Y.
W. E. Pomeroy, El Paso, Tex.
M. K. Rodgers, Los Angeles, Cal.
S. C. Rodgers, Guanajuato, Mex.

John M. Baker and Hamilton W. Baker have resigned as general manager and mine superintendent respectively of the Baker Mines Co., of Oregon. After Aug. 1, their address will be 43 Emerson St., Denver, Colo.

Noel Cunningham has opened an office at 200 Fifth Avenue, New York, where he will continue his work as consulting metallurgical engineer.

George E. Farish, of New York, has moved his western office from Denver, Colo., to the Nevada Bank Building, 14 Montgomery St., San Francisco, Cal.

Captain J. P. Hodgson, formerly mine superintendent of the Copper Queen, is now acting in a consulting capacity in the mining department of Phelps, Dodge & Co.

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William F. Jahn, formerly mill superintendent for the New York and Honduras Rosario Mining Co. at San Juancito Honduras, has accepted a position as mill superintendent for the Tough-Oakes Gold Mines, Limited at Kirkland Lake, Ontario, Canada.

Barnard MacDonald has moved his office from Los Angeles to Mills Bldg., El Paso, Tex.

H. K. Najarian has accepted a position with Bradley Bruff & Lakarthe of San Francisco, and is employed on designing work at the new smelter of the Missouri Cobalt Co. at Fredericktown, Mo.

Fred S. Porter has accepted a position with the Canadian Klondike Mining Co., Dawson, Yukon Territory.

R. W. Schultz, formerly with the Mond Nickel Co., has become associated with the Minerals Separation, Ltd.

Gerald Sherman is now general superintendent of the mine department of the Copper Queen.

J. Frank Sinclair, general foreman of the lower divisions of the Copper Queen, has been promoted to the position of superintendent of those divisions.

F. W. Traphagen has resigned as professor of metallurgy at the Colorado School of Mines and is now president of the Colorado Metal Mining & Reduction Co.

Felix E. Wormser is now with the Snake River Mining Co., Huntington, Ore.

Pope Yeatman, Mining Engineer, of 120 Broadway, New York, N. Y. will resume independent practice on Sept. 1, 1916.

POSITIONS VACANT

Engineer to take charge of construction of smelter and mill. Experience with blast furnaces, roasters, belt-conveying systems and general machinery, essential. Salary $250 per month up according to experience. No. 129.

Mine Engineer.—Man with experience, ability and initiative wanted to handle an organization, direct sampling, surveying, mapping and record keeping. Experience in laying out development work, haulageways, shaft pockets and ore-handling systems, particularly desired. No. 130.

Recent technical graduate for experimental work with a metal refining company. Laboratory work first to acquaint worker with methods, then research work. Good opportunity for advancement. Beginning salary $100 a month. No. 132.

Mine superintendent. Man of experience and energy wanted in mine being developed to produce 5,000 tons of iron-ore per day from open-cut and underground. About to start underground operations. Expect to follow methods of low-grade copper mines of Southwest. No. 133.

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Two fellowships for metallurgical research have been established by the University of Idaho. Each fellowship carries $500 per annum. No. 134.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, Cornell M. E., aged 31, married. Will soon complete 2 years' engagement in the Russian oil fields. Eight years' practical experience in drilling for and producing oil. Can speak Russian fluently and some French and German. Open for engagement. Interview after August 15, in the United States. No. 299.

Member, technical graduate, who has specialized in the treatment of gold and silver ores, desires position as mill superintendent or foreman; 7 years' practical experience. Highest references as to character and ability to get results. No. 307.

Member, graduate mining engineer. Married. Age, 38. For past 6 years, general superintendent of large silver mine in Mexico. Desires connection with responsible mining company, preferably in the United States. Has had 15 years' experience in the United States and Mexico, including mine engineering, sampling, examinations, milling and mine superintendence. No. 312.

Member, specialist in the mining and reduction of quicksilver ores, open for engagement Sept. 1. 19 years' experience. 14 years as Superintendent and Manager. Aged 39, married. No. 313.

Metallurgist, 4 years' experience in electric furnace operation on steel, ferro-alloys and smelting of ores, both practical and research, desires position as metallurgist or superintendent. No. 314.

Member, aged 44 years, superintendent of coal mines for 13 years and of coke ovens for 4 years, desires to locate in altitude of less than 3,000 ft. No. 315.

Member, technical mining graduate. 5 years' experience in coal and metal mines and construction work. Married. Aged 27. Employed at present. Interview in Salt Lake City or Denver. No. 316.

Member, technical graduate, 15 years' experience in engineering and mining in Western States, desires position as manager or superintendent. Age, 38. Married. Excellent references. No. 317.

Member, American, aged 39, married, desires position as superintendent or mine superintendent. 14 years' experience in opencut and underground work. 12 years' Spanish, Portugese, Cuban and Mexican labor. Fluent Spanish. Good training in first aid and safety first. Specialty of high explosives and steam-shovel work. Good costs. Go anywhere, but tropics preferred. References. Interview—New York or Philadelphia. No. 318.

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LOCAL SECTION NEWS

MONTANA SECTION

J. L. BRUCE, *Chairman*

W. C. SIDERFIN, *Vice-Chairman*

WALTER E. GABY, *Secretary-Treasurer*, 834 W. Granite St., Butte, Mont.

N. B. BRALY W. T. BURNS

A special meeting of the Directors of the Montana Section was held on July 24, 1916, with W. C. Siderfin, Vice-Chairman, presiding.

The meeting was called for the purpose of electing a Secretary-Treasurer to fill the vacancy created by the resignation of M. H. Gidel.

Upon proper motions, the resignation of M. H. Gidel was accepted, and Walter E. Gaby was elected to fill the office for the balance of the fiscal year.

No further business was transacted, and upon motion the meeting adjourned. WALTER E. GABY, *Secretary.*

SOUTHERN CALIFORNIA SECTION

C. COLCOCK JONES, *Chairman*

ALVIN B. CARPENTER, *Vice-Chairman*

F. J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Building, Los Angeles, Cal.

A. B. W. HODGES E. A. MONTGOMERY

R. A. PEREZ W. F. STAUNTON

The Annual Meeting of 1916 was held on June 28, at 7:30 p. m. There were present the Messrs. A. B. Carpenter, A. B. W. Hodges, F. J. H. Merrill, Seeley W. Mudd, R. A. Perez, and A. J. Underwood. The Chairman, Mr. Mudd, presided.

The Secretary read the minutes of the annual meeting of 1915, and of the meeting of September 28, 1915, as well as of the last meeting of the Executive Committee of May 25, 1916. On motion these minutes were ordered approved as read.

The Secretary-Treasurer then read his reports for the past year, and it was voted that these reports be approved and entered in the minute book.

A vote of thanks was given to the Chairman and the Secretary-Treasurer for their work during the year just past.

The amendment to Article III of the By-Laws, announced in the notice of the Annual Meeting, was put to vote and unanimously carried.

The Secretary read the minutes of the meeting of the Nominating Committee on May 31, and announced the result of the annual election as indicated above.

After discussion of plans for attending the Arizona meeting, the meeting adjourned.

F. J. H. MERRILL, *Secretary.*

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LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

COAL, ITS ECONOMICAL AND SMOKELESS COMBUSTION. By J. F. Cosgrove. Philadelphia, 1916.

COKE OVEN ACCIDENTS IN THE UNITED STATES DURING THE CALENDAR YEAR 1915. Technical Paper 151, U. S. Bureau of Mines, Washington, 1916.

MINE SAMPLING AND VALUING. By C. S. Herzig. San Francisco, 1914.

TECHNOLOGY OF MARBLE QUARRYING. Bulletin 106, U. S. Bureau of Mines, Washington, 1916.

Chemistry and Microscopy

HANDBUCH DER PHYSIKALISCH CHEMISCHEN TECHNIK FÜR FORSCHER UND TECHNIKER. By Kurt Arndt. Stuttgart, 1915.

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MICROSCOPICAL DETERMINATION OF THE OPAQUE MINERALS. By Joseph Murdoch. New York, J. Wiley & Sons, 1916. (Gift of publisher.) Price $2.00.

[NOTE.—Treats of the technique of mineral examination by treatment with various reagents under the microscope. Ingeniously arranged tables are provided. W. P. C.]

Mineralogy and Mineral Resources

CHILE. 1915. (Gift of Eduardo Carrasco.)

LIMESTONES AND MARLS OF THE COASTAL PLAIN OF GEORGIA. A report. Bulletin 21, Georgia Geological Survey. Atlanta, 1916.

ZIRCON, MONAZITE AND OTHER MINERALS USED IN THE PRODUCTION OF CHEMICAL COMPOUNDS EMPLOYED IN THE MANUFACTURE OF LIGHTING APPARATUS. Bulletin 25, North Carolina Geological and Economic Survey. Raleigh, 1916.

Power

BALANCING OF ENGINES, STEAM, GAS AND PETROL. By Archibald Sharp. New York, 1907.

POWER TRANSMISSION BY LEATHER BELTING. By R. T. Kent. New York, J. Wiley & Sons, 1916. (Gift of Publisher.) Price, $1.25 net.

[NOTE.—Belting practice has changed greatly in the past fifteen years. The literature is buried in the transactions of engineering societies and in technical journals. This book seeks to gather this into a compilation of the greatest service to the user of belting. W. P. C.]

General

AIRCRAFT IN WARFARE. By F. W. Lanchester. New York, 1916.

THE AUTHENTIC HISTORY OF THE UNITED STATES STEEL CORPORATION. By Arundel Cotter. New York, Moody Magazine & Book Co., 1916. (Gift of publisher.)

[NOTE.—The author admits a prejudice in favor of the corporation. It is an interesting contribution to industrial history. W. P. C.]

BIOGRAPHY OF AUGUSTE DAUBRÉE. June 25, 1814–May 29, 1896. (Gift of Miss F. Daubrée.)

BUSINESS AFTER THE WAR. By G. E. Roberts. New York, 1916. (Gift of National City Bank.)

ENGINEER'S YEAR BOOK, 1916. By H. R. Kempe. London, 1916.

THE LEGACY OF THE EXPOSITION. INTERPRETATION OF THE INTELLECTUAL AND MORAL HERITAGE LEFT TO MANKIND BY THE WORLD CELEBRATION AT SAN FRANCISCO, 1915. San Francisco, 1916. (Gift of Panama Pacific International Exposition Co.)

Company Report

TEMISKAMING AND NORTHERN ONTARIO RAILWAY, MINING INDUSTRY IN THAT PART OF NORTHERN ONTARIO SERVED BY. 1915. Toronto, 1916. (Gift of Temiskaming and Northern Ontario Railway.)

Trade Catalogs

THE ALLEN CONE CO., El Paso, Texas. Description of Allen Cone for classifying, tube milling, etc.

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Bulletin E-41. Duntley electric tools for street and interurban railways. 1916.

CHICAGO STEEL TAPE CO., Chicago, Ill. Price list. 1916.

INGERSOLL RAND CO., New York, N. Y. Form 9024. Steam condensing plants. June, 1916.

NATIONAL TRANSIT PUMP AND MACHINE CO., Oil City, Pa. Bulletin 502. Horizontal oil engines.

WIEDEKE, GUSTAV & CO., Dayton, O. Catalog No. 48. Ideal tube expander and cutter. 1915.

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MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period July 10, 1916, to July 31, 1916:

BOYD, ROBERT RUSSELL.....Supt., Arizona Commercial Min. Co., Globe, Ariz.
BRADLEY, HENRY FRANKLIN, Chem., American Smelt. & Ref. Co., Murray, Utah.
BROWN, THOMAS ELLIS, JR.....55 Hill St., Morristown, N. J
CARY, WEBSTER P., Expert Operator, General Engineering Co.,
Salt Lake City, Utah.
CLARK, ALONZO WEBSTER.....2222 Scottwood Ave., Toledo, Ohio.
CORP, CLIFFORD.....Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
CROMBIE, W. RANSOM, Min. Engr.....Detroit Copper Min. Co., Morenci, Ariz.
DELANEY, EUGENE A., Chief Engr., Berwind-White Coal Mining Co., Windber, Pa.
ELBELT, WILLIAM HENRY, Dept. Foreman., Chino Copper Co., Hurley, N. Mex.
FOSTER, L. JULIO, Mgr., Minas i Fundicion "La Junta," San Bernardo, Chile,
South America.
GASKILL, JAMES PHILLIPS, Genl. Mgr., Ajo Cons. Copper Co., Ajo, Ariz.,
via Gila, Ariz.
HAYS, FRED N., Asst. Steam Engr., Carnegie Steel Co., Farrell & Sharon Works,
Sharon, Pa.
HOOD, KARL KEDZIE, Min. Engr.....American Zinc Co., Mascot, Tenn.
JOHNSON, ERNEST C., Prest. & Treas., Johnson Engineering Works,
First National Bank Bldg., Chicago, Ill.
KNIFFIN, LLOYD MALCOLM, Asst. Supt., United States Metals Ref. Co.,
East Chicago, Ind.
KNOWLES, BENJAMIN WALLIS, Min. Engr., Hedley Gold Mining Co.,
Hedley, B. C., Canada.
LABBE, ARMAND LOUIS.....Chem., American Smelt. & Ref. Co., Murray, Utah.
LANG, SIDNEY A., Min. Engr., Braden Copper Co., Rancagua, Chile, South America.
LEES, JOHN W.....Genl. Supt., Inland Steel Co., Indiana Harbor, Ind.
LEOPOLD, F. N.....40 North Dearborn St., Chicago, Ill.
MCAFEE, DAN SHIELDS, The Dorr Company, 17 Battery Pl., New York, N. Y.
MCCLAIN, HENRY G., Genl. Supt., Liberty Bell Gold Min. Co., Telluride, Colo.
MARSHALL, ALBERT E., Chemical Engr.....Box 75, Sparkill, New York, N. Y.
MILLIGAN, WILLIAM EDGAR, Chem., Braden Copper Co., Rancagua, Chile,
South America.
MOSES, FREDERICK GALLAWAY, Flotation Engr., General Engineering Co.,
159 Pierpont St., Salt Lake City, Utah.
NEWBY, JERRY B., Oil Geol.....Gypsy Oil Co., Box B, Tulsa, Okla.
PALMER, EDGAR, Pres., New Jersey Zinc Co., 55 Wall St., New York, N. Y.
PATTINSON, REGINALD LANCASTER, Producer of Natural Gas and Petroleum,
29 5th St., Chatham, Ont., Canada.
RATHBUN, R. B., Asst. Research Engr., American Smelt. & Ref. Co., Murray, Utah.
RIGBY, WILLIAM ARTHUR, Chief Engr., Witherbee, Sherman & Co., Mineville,
Essex Co., N. Y.
SARGENT, ARTHUR E., Met. Engr.....Terry, So. Dak.
SCHIMERKA, FRANCIS S., Met. Chemist.....Shannon Copper Co., Clifton, Ariz.
STOWELL, GEORGE ELWIN, Instructor in Mining, Oregon Agricultural College,
Mining Dept., Corvallis, Ore.
STREET, LLOYD G., Met. Engr.....Mineral Point Zinc Co., Depue, Ill.
THAYER, WARREN N., Prof. of Geology, Ohio Mechanics Institute, Cincinnati, Ohio.
THOMPSON, DAVID P., Asst. Genl. Supt., Inland Steel Co., Indiana Harbor, Ind.
TITZELL, GEORGE GRAHAM, JR.....110 So. First St., Duquesne, Pa.
VELASCO, LOUIS IBANEZ, Engr.....Lehigh Coal and Navigation Co., Lansford, Pa.
WARD, HOWARD RIDGELY, Min. Engr., American International Corpn.,
120 Broadway, New York, N. Y.
WARDELL, JOHN WILFRID, Mech. and Const. Engr., Spassky Copper Mine, Ltd.,
Spassky Zavod, Karagandy, Akmolinsk Dist., Siberia.
WILLIAMS, MICHAEL THOMAS, Genl. Mgr., Edna May Gold Min. Co.,
N. L., Westonia, Western Australia.

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Junior Members

AMBLER, HARRY A., Student.....4050 Russell Ave., St. Louis, Mo.
ANDERSON, BARCLAY GLADSTONE, Asst. in Geol. Dept., Burro Mountain Copper Co., Tyrone, N. Mex.
BICKNELL, HAROLD LEWIS, Furnace Operator, Snyder Electric Furnace Co., 324 N. Grove Ave., Oak Park, Ill.
DIEFENDERFER, A. J.....34 North Oakland Ave., Sharon, Pa.
FINKELDEY, WILLIAM H., Min. Engr., Compania Minera y de Fomento Watkins, Fomento, Prov. Santa Clara, Cuba.
LEVY, MILTON MORTIMER.....1 Vernon Apts., Salt Lake City, Utah.
MILLER, ROY HARRISON, Milling....Anaconda Copper Min. Co., Anaconda, Mont.
MILYKO, ALEXANDER, Asst. Geol., Producers Oil Co., Box 75, Houston, Tex.
PETZEL, CHARLES L., Student.....Columbia University, New York, N. Y.
WALTER, ALPHONS R., Learner of concentrating plant, Bethlehem Steel Co., So. Bethlehem, Pa.
WORMSER, FELIX E.....P. O. Box 156, Snake River Mines Co., Huntington, Ore.

Total Membership, July 31, 1916.....5,654

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period July 10, 1916 to July 31, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Hans Claudius Anchor, So. Porcupine, Ont., Canada.

Proposed by H. P. De Pencier, J. Ralph Scott, C. E. Rodgers.

Born 1857, Hasle, Bornholm, Denmark. Attended Public School in Denmark up till 1871. Studied under my Father at Coal Mine (Lignite), Island of Bornholm, till 1874. 1875–76, Mining and railroad construction, Comstock, Nev. Learned assaying for the common metals, 1877. 1877–85, Worked in different mines through the State of Nevada. 1885–86, Black Jack, Silver City, Ida. 1887–97, De Lamar, Ida. 1897–98, Twin Devils, Ida. (copper). 1898–1902, Boise Basin, Ida. (gold). 1902–05, Bully Hill, Cal. (copper). 1906–07, Hermoza Copper Co., N. Mex. 1908–10, Traveling, mostly examining prospects. 1910, Sampled and outlined orebody Dome Mines, So. Porcupine.

Present position: Charge of Developments, Dome Extension Mines Co., Ltd.

Edward Otto Forster Brown, London, E. C., England.

Proposed by M. C. Scheble, Edgar Rickard, H. Foster Bain.

Born 1881, Cardiff, Great Britain. 1895–1900, Winchester College. 1900–05, Training mainly at collieries. 1905, Obtained first class Mine Managers' Certificate. 1905–07, Inspecting collieries and mines, S. Africa, China, Japan, United States, Canada. 1908–12, Cons. Engr., 7 Cohring Cos. in Coahuila, Mex., and reporting in coal fields in British Columbia and Pennsylvania.

Present position: Managing and Cons. Engr., for a number of coal mines in Kent, S. Wales, N. Wales and Wolpam Mines in Spain.

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Gilbert Haven Cady, Urbana, Ill.
Proposed by J. N. Ede, Thomas T. Read, William H. Shearman.
Born 1882, Chicago, Ill. 1905, Northwestern Univ., A. B. 1907–08, Graduate student in Geology, Northwestern Univ. 1909–10, Graduate student in Geology, Yale Univ. 1911–12, Graduate student in Geology, Illinois Univ. 1912–13, Graduate student in Geology, Chicago Univ. 1907–16, Member of the staff of the Illinois Geol. Survey with rank of Geologist since 1914. Have been in continuous employment on the survey since 1914. Have especial charge of the coal investigations and have prepared two published reports on Illinois Coal Districts—Longwell District and Jefferson, Franklin and Williamson Counties District—and have other reports in preparation. Have also presented for publication manuscripts for two U. S. Geol. Survey folios concerning West Frankfort-Galatia quadrangles and Hennepen La Salle quadrangles. 1913–14, Instructor in Geology, Northwestern Univ. Am a member of the U. S. Geol. Survey with rank as Asst. Geol..
Present position: Geologist, Illinois Geol. Survey.

H. T. Chen, Yunnan Prov., Yunnan, China.
Proposed by Geo. O. Scarfe, Robert H. Bedford, A. D. Foote.
Born 1878, Hunnan Prov., China. Attended Schools in China, and Japan. Berkeley High School. 1911, Univ. of Cal., College of Mining, B. S. 1911, winter, 1912, spring, was the director of Industrial Dept. of Hunnan Prov. 1912, summer, engaged to be Asst. Engr., Yunnan Tin Trading Co. 1913, spring, was promoted to be Engr. in Chief of Yunnan Tin Trading Co. The works are entirely modern, containing 52 washing tables and 6 reverberatory furnaces.
Present position: Engr. in Chief, Yunnan Tin Co.

Charles L. Constant, New York, N. Y.
Proposed by H. W. Hardinge, Ronald Clark, R. B. T. Kiliani.
Born 1856, Cincinnati, O. 1877, Grad. from School of Mines, Columbia, as Civ. and Min. Engr.

James Belcher Etherington, Winthrop, Mass.
Proposed by D. A. Lyon, Oliver C. Ralston, C. A. Wright.
Born 1851, Nova Scotia. 1865, Mixed Schools in Nova Scotia. 1872, Miss Boynton's Private School, Lynn, Mass. 1873, Bryant and Stratton Commercial School, Boston. 1874–79, Bookkeeper for Oscar F. Howe, Woodenware, Boston, Mass. 1881–92, Dealer, Mfgr. & Exporter, Manufactures of Wood, firm of J. B. Etherington & Co., Boston. 1892–98, Mfgr. of hardwood lumber, Bradford, Pa., firm of Bradford Hardwood Lumber Co. Since 1901, have been active principally in magnetic separation of ferruginous ores. Knowledge of the subject gained chiefly through study of Encyclopedia Brittanica and through association with Dr. Campbell, Dr. S. P. Sharples, Boston, and various others in the zinc business in the United States, Canada and Belgium.
Present position: Zinc Concentrating Co., Boston, Mass.

Julio F. Fernandez, Manzanillo, Cuba.
Proposed by Louis F. Chibas, E. Chibas, Thomas T. Read.
Born 1882, Havana. 1906–10, Asst. Engr., Public Works Dept. 1910–12, Chief Engr. of Manzanillo Water and Light Co. I made the project of Manzanillo Water Work. Have been the Concessionary. Studied in Spain, Zaragozo University. Afterward in Cuba by correspondence with American School of Correspondence of Chicago, Ill. 1912–14, I made the Manzanillo project of city pavement, 20 miles of concrete, and at the same time the studies of Manzanillo and Wignero R. R., 52 miles. I am the Promoter and the Director. 1914–16, Promoter and studies of Manzanillo Electrical Traction Co. In this moment I am exploiting two copper mines in Manzanillo and am Promoter of the Manzanillo Mines Copper Co.
Present position: Director and Promoter of all foregoing work.

Rollins Sanders Foster, Butte, Mont.
Proposed by C. W. Goodale, F. A. Linforth, B. B. Thayer.
Born 1885, Montana. 1900–03, Montana State School of Mines. 1904, Univ. of Cal. 1905–07, Miner, sampler and timekeeper, Gagnon Mine for C. G. Adami. 1907–11, Asst. Surveyor and Shift boss, North Butte Min. Co. 1911–13, Prospector, Belgian Congo, Société Forestière et Minière du Congo. 1914–15, Miner and Shift boss, Anaconda Copper Min. Co. 1915–16, Asst. Safety Engr. and Safety Engr., Anaconda Copper Min. Co.
Present position: Safety Engr., Anaconda Copper Min. Co.

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G. V. Foutz, Minas de Ponupo, La Maya, Oriente, Cuba.
Proposed by Charles F. Rand, E. Aguilera, Pedro Aguilera.
Born 1893, Mason-Dixon, Pa. 1911, Waynesboro, Pa., High School. 1915, Pennsylvania State College, B. S., Industrial Chemistry. 1915, Aetna Explosives Co., Emporium, Pa., Carnegie Steel Co., Duquesne, Pa., Aguilera & Co., Santiago de Cuba.
Present position: Chemist.

Walter Irving Garms, Hayden, Ariz.
Proposed by Louis S. Cates, D. D. Moffat, William T. MacDonald.
Born 1886, San Franciso, Cal. 1900–05, Technical course in Cal. School of Mech. Arts, San Francisco. 1905–10, Univ. of Cal., B. S. in College of Mining. 1907, Mining, South Eureka Min. & Mill. Co., Sutter Creek, Cal. 1910, Sampling and Geology, Ray Cons. Copper Co., Ray, Ariz. 1911–16, Milling and construction, Ray Cons. Copper Co., Hayden, Ariz.
Present position: General Mill Foreman, Ray Cons. Copper Co.

George William Hain, Lansford, Pa.
Proposed by Edwin Ludlow, W. G. Whildin, J. B. Warriner.
Born 1886, Tremont, Pa. 1909, Lehigh Univ., E. M. 1909–12, Min. Engr., Juragua Iron Co., Santiago de Cuba. 1912–15, Mgr., West Virginia Engrg. Co., Fairmont, W. Va.
Present position: 1915 to date, Asst. to Dist. Supt., Lehigh Coal and Navigation Co.

Daniel R. Hull, Kenosha, Wis.
Proposed by Milo W. Krejci, John H. Klepinger, E. S. Bardwell.
Born 1888, Southington, Conn. 1908–11, Sheffield Scientific School, Yale Univ. 1911–16, Met. Chemist, Kenosha Branch, American Brass Co.
Present position: American Brass Co.

Arthur D. Knowlton, Salt Lake City, Utah.
Proposed by David Lemmon, L. O. Howard, J. H. Winwood.
Born 1881, Tooele Co., Utah. 1903, Univ. of Utah, B. S. in Min. Engrg. 1903–05, with State Engineer, Utah. 1905–07, with City Engineer, Salt Lake City. 1907–16, Min. Engrg., U. S. Mineral Surveyor, Utah and Nevada. 1916, Cons. Engr., Boston Development Co.
Present position: Engr., Boston Development Co.

Ray P. McGrath, San Francisco, Cal.
Proposed by Frederick K. Copeland, Stillman Batchellor, Edward Jüssen.
Born 1883, Stratford, N. H. 1906, Dartmouth College. 1906, worked for Sullivan Machinery Co. Have been with this company since that time acting as a sales engineer, testing, installing and selling mining machinery.
Present position: Dist. Mgr., Sullivan Mach. Co.

Will E. McKee, Warren, Ariz.
Proposed by John C. Greenway, Ira B. Joralemon, W. B. Gohring.
Born Teppreanor Co., Ind., 1866. 1890, Univ. of Ill., B. S. in Mech. Engrg. 1890–91, Draughtsman, Link Belt Mach. Co. 1891–92, A. L. Ide & Son, Springfield, Ill. 1892–93, Construction, Pabst Brewing Co. 1894, Engr., Sanger Bros., Dallas, Tex. 1895–96, Cons. Engr., E. F. Osborn, Chicago. 1897–1905, Master Mechanic, Cleveland Cliffs Iron Co., Ishpeming, Mich.
Present position: 1905 to date, Supt. Machinery, Calumet & Arizona Mining Co.

John Henry Miles, Trinity Center, Cal.
Proposed by Benjamin F. Tibby, Harry L. Mead, Frank E. Johnson.
Born 1882, Calumet, Mich. Butte Grammar School. Four years, Butte Business College. 1905–07, Supt. construction, Boston Machine Shop Co., Oroville, Cal. 1907–11, Master mechanic, Yuba Cons. Gold Fields Dredging Co., Hammonton, Cal. 1911–13, Supt. of Dredge Folsom Division, Natomas Cons. of Cal. 1913–16, Genl. Supt., Boston and Idaho Gold Dredging Co., Idaho City, Ida.
Present position: Genl. Supt., Alta Bert Gold Dredging Co.

Hennann Wilhelm Paul, Yokohama, Japan.
Proposed by Walter A. Schmidt, L. D. Ricketts, Thomas T. Read.
Born 1878, Weddingen, Germany. 1904, Engr., Freiberg in Saxony. 1905–06, Borhumer Verein für Bergben und Gusstahlfabrikation, Borhum in Westphalia, as Chemist. 1906–09, technical expert for their Mining, Metallurgy and Ore Business,

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Otto Reimers & Co., Yokohama, Japan. 1910, Metallgesellschaft, Frankfort and Telge and Schroeter, Shanghai, China, for their Mining and Ore Business in China. 1911–12, Otto Reimers & Co., Yokohama, Japan.

Present position: Min. and Met. Engr. Representation of the Metallgesellschaft and the Metallbank, Metallurgische gesellschaft, Frankfort, for Japan, and of the International Precipitation Co., Los Angeles, Cal., for Japan.

Francis Stuyvesant Peabody, Chicago, Ill.

Proposed by Charles H. MacDowell, Robert W. Hunt, Carl Scholz.

Born 1859, Chicago, Ill. 1881, Grad., Sheffield Scientific School, Ph. B. Mining coal 20 years.

Present position: Pres., Peabody Coal Co.

H. J. D. Penhale, Oruro, Bolivia, S. Amer.

Proposed by M. G. F. Söhnlein, P. F. Bliek, H. L. Venables.

Born 1893, Illogan, Cornwall, England. Cornwall County Schools. School of Metaliferous Mining, Redruth, Cornwall. 1911, Asst. Surveyor, South Crofty Mines, Cornwall. 1912, Engr. to the Devon United Mines, Ltd. 1913, Mgr. (Mine) Antequera Mines, Bolivia. 1913–14, Surveyor, Compania Minera, Nane, Bolivia. 1914–15, Asst. Mine Mgr. and Surveyor, Cia. de Minas de Cobre de Gatico.

Present position: Engr., House of Penny y Duncan.

Kiva Pommerantz, Santiago, Chile, S. Amer.

Proposed by Louis A. Wright, Mark R. Lamb, F. H. Barclay.

Born 1884, Kuhurlui, Besarabia, Russia. 1901, Grad., B. S., in Komrat, Russia. 1901–05, Mining Academy, Freiberg, Saxony, "Diplom-Ingenieur". 1905–08, Surveyor and later on Superintendent of one of the zinc mines belonging to the "Minengesellschaft Fr. Speidel, Tharsos" Turkey. 1908–11, Supt., Manganese and Copper Mines of Hopa (Asia Minor). 1911–14, Technical Mgr., "Société Hongroise d'Etudes de l'Industrie Minière et Metallurgique," Budapest.

Present position: 1914 to date, Cons. Engr., Ore Trading Co., Ltd.

Harold Carmichael Robson, Akmolinsk, Siberia.

Proposed by Walter G. Perkins, C. G. Hall, Herbert C. Woolmer.

Born 1890, Penzance, Cornwall, England. 1897–1900, The Abbey School, Penzance, Cornwall, England. 1900–05, St. Mary's School, Penzance. 1905–09, School of Mines, Penzance, receiving school diplomas. 1909–10, Practical work, Dolcoath Mine, Camborne, Cornwall, England, Min. & Mill. 1910–11, Asst. to W. Roberts, Siberian Syndicate, London Wall, E. C., England, who reported on various concessions in the Altai District, Siberia. 1911–14, Junior Asst. Engr., Spassky Copper Mine, Ltd.

Present position: 1914 to date, Asst. Engr. in Smelter, Spassky Copper Mine, Ltd.

Leon Fair Russ, Tampico, Mexico.

Proposed by C. W. Hamilton, E. DeGolyer, Lewis C. Chapman.

Born 1886, Coushetti, La. 1909, Univ. of Texas, B. A. 1909–10, special work geology and chemistry, Univ. of Texas. 1907–10, Geologic field work, Field Asst., U. S. Geol. Survey. 1911, Geol., Cia. Mex. de Pet. "El Aguila," S. A.

Present position: Geol., Cia. Mex. de Pet. "El Aguila," S. A.

Jesse D. Smith, Galena, Ill.

Proposed by Arthur Thacher, H. A. Wheeler, J. F. Thompson.

Born 1860, New Madrid, Mo. Few years in Public Schools. 1897–1905, Central Lead Co., Flat River, Mo. 1905, Federal Lead Co., Flat River, Mo. 1906, St. Louis Smelting & Refining Co., Flat River, Mo. 1907–1916, Mineral Point Zinc Co., Mineral Point, Wis.

Present position: Supt., Black Jack Mine.

Thomas B. Sturges, Pittsburgh, Pa.

Proposed by Richard T. Dana, Clarence B. Sturges, A. H. Carnahan.

Born 1884, Wilkes-Barre, Pa. 1901, C. P. Course, Wilkes-Barre, Pa., High School. Other geological and engineering knowledge gained by private study and field work. 1902, Rodman and inspector, Webster Coal & Coke Co., Cresson, Pa. 1903–06, Min. Engr., Cons. & Fernwood Collieries, Hillside Coal & Iron Co., Dunmore, Pa. 1906–16, Genl. Mgr., Pennsylvania Drilling Co. Prospecting coal and other mineral land in United States & Canada, Cuba, Central America, etc.

Present position: Vice-Pres. & Genl. Mgr., Penn. Drilling Co.

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Haruyoshi Tomita, Akitaken, Japan.
Proposed by E. P. Mathewson, H. W. Aldrich, Ichiro Kamimura.
Born 1887, Kumamoto, Japan. 1906–09, The 5th Higher School, Kumamoto. 1909–12, Tokyo Imperial Univ.
Present position: 1912 to date; Met. Engr., Osarusawa Mine, Mitsubishi Co.

Frederick H. Vahrenkamp, Salt Lake City, Utah.
Proposed by David Lemmon, G. B. Wilson, L. O. Howard.
Born 1870, Concordia, Mo. 1888, High School. 1892–96, Private studies in engineering and geology. 1898–1904, Business associate of Prof. Mitchell of Mitchell & Vahrenkamp, Los Angeles, Cal. 1905–06, Prest., Anvil Springs Gold Mining Co. 1906–08, Prest., Bonnie Clare Mining Co., Goldfield, Nev. 1908–09, E. M. Humboldt Exploration Co., New York, N. Y. 1909–12, Genl. Mgr., Coates Motor Car Co., New York, N. Y. 1913–14, Prest., Onyx Travertine Co., Salt Lake City, Utah.
Present position: Prest., Boston Development Co., Boston Nevada Copper Co.

William Wilke, Buffalo, N. Y.
Proposed by C. W. Stimpson, R. F. Barker, W. D. Leonard.
Born 1855, Darmstadt, Germany. Public and High School in Darmstadt, Germany. 1872–77, Polytechnicum Darmstadt, M. E. 1881–86, Asst. Engr., Midvale Steel Co., Philadelphia, Pa. 1886–93, Designing and erecting plant for the manufacture of mineral acids and other chemical products, developing new processes; Chem. Engr., Harrison Bros. & Co., Chem. Works, Philadelphia, Pa. 1893–16, Own firm of Wm. Wilke, Buffalo, N. Y., designing and erecting chemical works; specialty, sulphuric acid, vitric, hydrochloric, acetic, etc. Built plants for Standard Oil Co., Du Pont de Nemours Co., General Chemical Co., Graselli Chemical Co., Penna. Salt. Mfg. Co., Butterworth-Judson Co., Virginia-Carolina Chemical Co., The Phosphate Mining Co., etc.
Present position: Consulting Engineer, American Smelting & Refining Co.

Associate Members

H. H. Allen, Salt Lake City, Utah.
Proposed by J. B. Ambler, George R. Sheldon, C. T. Van Winkle.
Born 1891, Bessemer, Mich. 1905–09, Ishpeming (Mich.) High School. 1909–11, Michigan Agricultural College. 1912–14, Michigan Agricultural College. 1916, Shops of Sullivan Mach. Co., Claremont, N. H. Michigan Agricultural College, B. S. in Engrg. 1915, H. H. Franklin Mfg. Co., Syracuse, N. Y. 1915, Vienna Cons. Mines & Smelt. Co., Hailey, Ida. 1915, Sullivan Machinery Co., Chicago, Ill.
Present position: Sales Engr., Sullivan Mach. Co.

Walter J. Riley, East Chicago, Ind.
Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.
Born 1875, Chicago, Ill. De La Salle Univ. Pres., 1st State Trust & Savings Bank; 1st Calumet Trust & Savings Bank. O. F. Jordan Co., East Chicago, Ind.
Present position: As above.

Paul Steyens, Warren, Ariz.
Proposed by James C. Greenway, Ira B. Joralemon, W. B. Gohring.
Born 1879, Minneapolis, Minn. 1898, Central High School, Minneapolis, Minn. Last eight and one-half years, Private Secy. to Genl. Mgr., Calumet & Arizona Mining Co. Previously in business for self.
Present position: Private Secy. to Genl. Mgr., Calumet & Arizona Min. Co.

Junior Member

Oscar Allen Knight, Cambridge, Mass.
Proposed by Albert Sauveur, H. M. Boylston, F. C. Langenberg.
Born 1893, Athens, O. 1916, Ohio Univ., A. B.
Present position: Student, Mining School, Harvard Univ.

Change of Status—Junior Member to Member

Fletcher H. Wood, Metcalf, Ariz.
Born 1888, Mt. Vernon, N. Y. 1895–1907, Public Schools, Mt. Vernon, N. Y. 1907–09, Sheffield Scientific School. 1912–15, Colorado School of Mines. 1909–12, Costilla Estates Dev. Co. 1913, summer, Mucker, Portland Gold M. & M. Co. 1914, summer, Mucker and Miner, Portland Gold M. & M. Co. 1915, summer, Timber-helper, Portland Gold M. & M. Co. 1915, fall, Engrg. Dept., Arizona Copper Co. 1915-16, Engrg. Dept., Ray Cons. Copper Co.
Present position: Dev. Boss, Arizona Copper Co.

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CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period July 10, 1916 to July 31, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 115, February to August, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of July 31, 1916.

ALLEN, GLENN L.....Met., Shattuck-Arizona Copper Co., Bisbee, Ariz.
BABB, P. A.....Ave. Cinco de Mayo No. 32, Mexico City, Mexico.
BAILLIE, FRANK S.....Mgr., Baker Mines Co., Cornucopia, Ore.
BAKER, HAMILTON W.....1043 Emerson St., Denver, Colo.
BAKER, JOHN M.....1043 Emerson St., Denver, Colo.
BARKER, L. M.....Hayden, Ariz.
BARNES, KENNETT F.....5151 Washington Ave., St. Louis, Mo.
BARTLETT, WILLIAM SYDNEY, Managing Director, The Copper Mines of Copiapo, Ltd., Copiapo, Chile, South America.
BOWMAN, REGINALD G.....Tooele, Utah.
BRIBER, F. E.....Highland Boy Mine, Bingham, Utah.
BURG, ROBERT S.....American Smelt. & Ref. Co., Durango, Colo.
CAZIN, FRANZ.....Herculaneum, Mo.
CHARLES, L. J.....566 High St., Denver, Colo.
CHATIN, AUGUST H.....3832 Umatilla St., Denver, Colo.
CHEDSEY, WILLIAM R.....1414 Gaylord St., Denver, Colo.
CRAGIN, RODNEY S.....Hill City, S. Dak.
CRANDALL, S. A., Draughtsman, Bunker Hill and Sullivan Smelter, Kellogg, Ida.
DAVIS, LLOYD J.....1010 Boatmen's Bank Bldg., St. Louis, Mo.
DICK-CLELAND, A. F., Dick-Cleland Harper & Co., 49 Jamaica St., Glasgow, Scotland.
DOBSON, CHRISTOPHER G.....Britania Mine, Howe Sound, B. C., Canada.
DOWD, JAMES J.....Box 389, Houghton, Mich.
FAUST, GUY C.....19 West Jackson St., Wilkes-Barre, Pa.
FIELD, FREDERICK M.....Easton-Pacific Co., Virginia City, Mont.
FOHS, F. JULIUS, Cons. Oil Geol.....Suite 307, Gallais Bldg., Tulsa, Okla.
FULLAWAY, RICHARD MERLE.....817 E. 3rd St., Los Angeles, Cal.
GIRAULT, E.....327 Goliad St., San Antonio, Tex.
GRAYBILL, JOHN H., Industrial Engrg. Dept., Westinghouse Electric and Manufacturing Co., E. Pittsburgh, Pa.
HASTINGS, J. H.....820 Sheridan St., Monongahela, Pa.
HILL, JOSEPH H.....Ray Cons. Copper Co., Ray, Ariz.
HLEBNIKOFF, KENNETH I.....Goldfield, Nev.
HOSKIN, ARTHUR J.....P. O. Box 526, Leadville, Colo.
HSU, SHEN-CHIN.....Songlin, Huchow, Chekiang, China.
HU, S. H.....1004 Brush St., Detroit, Mich.
HUDSON, A. W.....Copper Queen Cons. Min. Co., Bisbee, Ariz.
HUGHES, WILSON W.....Monitor Belmont Min. Co., Belmont, Nev.
JOHNSON, C. B.....Federal Lead Co., Federal, Ill.
JOHNSON, J. HARLAN, So. Dakota National Guard, Hospital Corps, Camp Hagman, So. Dak.
JONES, CLEMENS AP-C.....Hold publications.
JOPLING, R. F.....Hold publications.
KEMP, JAMES T.....51 Hyatt St., New Brighton, N. Y.
KERNAN, THOMAS H., School of Mines, Experiment Station, Univ. of Minn., Minneapolis, Minn.
KING, PAUL S.....807 West Ninth St., Wilmington, Del.
KNAPP, GEORGE F.....R. R. 1, Box 67B, West Wrentham, Mass.
LASKY, BERNARD H.....U. S. Smelt. Ref. & Min. Co., Tucson, Ariz.
LAWSHE, VERNER T.....4819 Warrington Ave., Philadelphia, Pa.
L'ENGLE, E. FLEMING.....Frisco Bldg., Joplin, Mo.
LESSIG, NICOLAI, Genl. Mgr., Ridder Mining and Trading Co., Ridder Mine, Tomsk Government, Russia.
LEWIS, JOHN MOORE.....Chief Engr., Houston Coal Co., Elkhorn, W. Va.
LEWIS, ROBERT S.....Box 902, Stanford University, Cal.
LIDDELL, PARKER.....Reno, Nevada.

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LIEBIG, J. O.....161 Sherman Ave., Newark, N. J.
LIHME, C. B.....Lake Forest, Ill.
LINDSAY, J. A. M.....Broken Hill Prop. Co., Broken Hill, N. S. W., Australia.
LIST, E.....Rolla, Mo.
McDONALD, JOHN A.....Cananea Cons. Copper Co., Cananea, Son., Mexico.
McHARDY, R. H.....General Delivery, Bisbee, Ariz.
MacDONALD, BERNARD.....Mills Bldg., El Paso, Tex.
MacDONALD, JESSE J.....Box 421, Clarkdale, Ariz.
MACKENZIE, GEORGE S.....Craig Lee, Broughty Ferry, Dundee, Scotland.
MALCOLMSON, J. D.....Thompson, Nev.
MASTERS, J. E.....434 E. 2d Ave., Tucson, Ariz.
MEANS, A. H.....Copper Mountain, Ajo, Ariz.
MILLARD, WILLIAM J.....Box 75, Houston, Tex.
MILLER, W. B.....Sardis, B. C., Canada.
MOSONYI, EMIL, Salvador Deep Well Boring Co., P. O. Box 158, San Salvador, Republic El Salvador, Central America.
NEELD, H. C.....Hold publications.
PATTINSON, R. L.....Chatham, Ontario, Canada.
PELTON, HAROLD G.....Union Oil Co., Avila, San Luis Obispo Co., Cal.
PESCHEL, WILLIAM M.....51a R. F. D. 2, Lewiston, Ida.
PHILLIPS, WILLIAM B.....Austin, Tex.
PIGOTT, CURTIS.....Met., Midvale Smelter, U. S. Smelt. Co., Midvale, Utah.
RAU, H. L.....Hazel Green, Wis.
RAY, FRANK A.....515 Huntington Bank Bldg., Columbus, Ohio.
ROBINSON, E. M.....Psi U House, S. Bethlehem, Pa.
ROESLER, MAX, Geol.....Juragua Iron Co., Box 383, Santiago de Cuba, Cuba.
RORK, FRANK C.....Humboldt, Ariz.
RUEBEL, ERNST H., Custodian's House, State Fair Grounds, Springfield, Ill.
RYDER, THOMAS J.....Apartado 189, Vera Cruz, Ver., Mexico.
SCHMIDT, R. D.....Corrigan McKinney Co., Cleveland, Ohio.
SCHWARZ, CHARLES E., Min. and Met. Engr.....P. O. Box 715, Miami, Okla.
SKOGMARK, JOHN.....Room 1509, 35 Nassau St., New York, N. Y.
SMITH, ELWYN L.....Granby, Mo.
SOPER, ELLIS C., Cons. Engr.....Mariel, Cuba.
SPROAT, A. D.....73 W. 5th St., Chillicothe, Ohio.
STEEL, DONALD.....Douglas, Alaska.
STROUP, THOMAS A., Engr.....Arthur Mine, Utah Copper Co., Garfield, Utah.
THOMPSON, WARREN D.....Kennebec Hotel, Long Beach, Cal.
THROPP, J. E., JR.....Berkshire Iron Works, Sheridan, Lebanon Co., Pa.
TONG, S. K.....404 W. 115th St., New York, N. Y.
TSUBOI, YOSHIO.....Tsuboi Mining Office, 126, Koyasu, Yokohama, Japan.
VIGEON, E. C., Genl. Mgr., Bede Metal and Chemical Co., Ltd., Hebburn, near Newcastle-on-Tyne, England.
WALTERS, R. E.....Asst. Mgr., Snow Storm Mines Cons., Troy, Mont.
WATSON, HUGH C.....Yak Mine, Leadville, Colo.
WATT, ARTHUR P., Met.....St. Louis Smelt. & Ref. Co., St. Francois, Mo.
WELLS, J. S. C.....Care R. de C. Greene, 211 Broad St., Elizabeth, N. J.
WORTH, JOHN G., Min. Engr....836 Real Estate Trust Bldg., Philadelphia, Pa.
WRIGHT, LOUIS A.....Casilla 125 D. Santiago, Chile, South America.
YOUNG, FRED W.....Mercer County Light, Heat & Power Co., Greenville, Pa.

MEMBERS' ADDRESSES WANTED

Name.	Last Address of Record from which Mail has been Returned.
BARNES, BLAKESLEE	Arrow Engineering Co., Palmyra, Mo.
BELL, D. A. S.	136 McLaren St., Ottawa, Can.
BLOW, J. J.	172 Rodney St., Brooklyn, N. Y.
BOOTH, E. L.	349 W. 145th St., New York, N. Y.
BOYS, H. R.	Aurora Cons. Min. Co., Aurora, Nev.
CAIRNS, JOHN M.	Henry S. King & Co., 65 Cornhill, London, E. C., England.
COLE, ROBERT J.	McKay Apartments, 7th & Pike Sts., Seattle, Wash.
CRARY, CHARLES N.	Kimberly, Nev.
CUSHING, DANIEL	Sandusky Foundry & Machine Co., Sandusky, Ohio.
DALEY, STEPHEN H., JR.	Box 943, Niagara Falls, N. Y.

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DRAPER, CARL H. ..... Apartado 77, Guadalajara, Jal., Mexico.
FOSTER, GEORGE C. ..... 18 Cadogan Blk., Calgary, Alta., Canada.
GOEDICKE, CARL ..... Box 535, San Antonio, Tex.
GORDON-FIREBRACE, W. E. ..... 812 Salisbury House, London, E. C., England.
GUNTHER, C. GODFREY ..... Stratford, Conn.
HANLON, JOHN EDWARD ..... Timmins, Ont., Canada.
HOBART, EDMUND NORRIS ..... Clifton, Ariz.
JONES, THOMAS J., Mine Mgr., Kyshtim Min. Wks., Perm Govt., via Petrograd, Russia.
KELLOGG, R. M. ..... Wellington, Lyon Co., Nev.
KISHMAN, MAURICE W. ..... 101 Masonic Ave., Cripple Creek, Colo.
LENOIR, FRANK H. ..... Douglas, Alaska.
McGEE, JOHN ..... United Greenwater Co., Dale via Amboy, Cal.
MAINWARING, H. M. C., Mine Mgr., Chillagoe, Ltd., Chillagoe, Queensland, Aust.
MILLER, FRANK BARTON ..... Blair, Esmeralda, Nev.
MOHRMAN, E. M. ..... 1293 W. 111th St., Cleveland, Ohio.
MOORE, REDICK R. ..... Zortman, Mont.
O'BRIEN, P. C. K. ..... Hotel Essex, 684 Larkin St., San Francisco, Cal.
PARRISH, S. F. ..... Battle Mountain, Nev.
PATCHELL, F. J. ..... 4516 No. Lincoln St., Chicago, Ill.
PATERSON, A. W. ..... 1814 11th Ave., Spokane, Wash.
PERRY, ROBERT S. ..... Kingscourt Apts., 36th & Chestnut Sts., Phila., Pa.
RALPH, E. W. ..... Boston Ely Mining Co., Kimberly, Nev.
REDFEARN, A. M. ..... Hold publications.
REVELL, G. E. ..... Box 132, Nelson, B. C., Canada.
REYNOLDS, L. ..... Apartado 25, Guanajuato, Mexico.
RHODES, W. B. ..... Golden, Colo.
RODRIQUEZ, J. C. ..... Apartado 76, Saltillo, Coah., Mexico.
ROGERS, B. C., Care Standard Mine, Detroit Copper Min. Co., Box 427, Metcalf, Ariz.
SALES, A. J. ..... Giroux Cons. Mines Co., Kimberly, Nev.
SANDIFER, H. C. ..... Box 15, Bis., Mexico City, Mexico.
SMITH, A. H. ..... Gateway, B. C., Canada.
SPARKS, J. T. ..... 326 Erie St., El Paso, Tex.
STODDART, A. W. ..... 638 Salisbury House, London, E. C., England.
SULLIVAN, W. P. ..... The Henry Walke Co., Norfolk, Va.
TAYLOR, A. W. ..... Korean Exploration Co., Chiksan Mines, Chiksan, Korea.
THOMAS, EDMUND ..... Weaver Mine, Gibson, N. Mexico.
TOD, GRANT H. ..... Care Coast Mfg. & Supply Co., Livermore, Cal.
TONG, S. K. ..... 413 W. 115th St., New York, N. Y.
VAN NESS, W. W. ..... 622 Salisbury House, London, E. C., England.
WAINEWRIGHT, W. B. ..... 61 Grace Church St., London, E. C., England.
WARD, HARRY J. ..... 948 Market St., San Francisco, Cal.
WENTWORTH, I. H. ..... 245 Beldon Ave., Harlandale Addition, San Antonio, Tex.
WRAIGHT, E. C. ..... 63 Wavertree Road, Streatham Hill, London, S. W., England.
YEANDLE, W. H., JR., Care Guilermo Brockmann, 2a Capuchines, No. 55, Mexico City, Mexico.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period July 10, 1916 to July 31, 1916.

Date of Election.	Name.	Date of Decease.
1900	*Deidesheimer, P.	—— —, 1916.
1879	**Hinchman, Charles S.	June 3, 1916.
1875	*Kirchhoff, Charles	July 23, 1916.
1914	*Levy, A. G.	—— —, 1916.
1891	**Louis, David A	—— —, 1913.
1907	*Pocock, Cecil W.	July 19, 1916.
1899	*Louis Rosenfeld	July 20, 1916.

*Member. **Life member.

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EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman.* PERCY E. BARBOUR, *Vice-Chairman.*
A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer.*

Boston

Meets first Monday of each winter month.

W. E. C. EUSTIS, *Chairman.* R. L. AGASSIZ, *Vice-Chairman.*
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.

STANLY A. EASTON, *Chairman.* FREDERIC KEFFER, *Vice-Chairman.*
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
D. C. LIVINGSTON, FRANK A. ROSS.

Puget Sound

Meets second Saturday of each month.

GLENVILLE A. COLLINS, *Chairman.* H. L. MANLEY, *Vice-Chairman.*
AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
I. F. LAUCKS, JOHN N. POTT.

Southern California

C. COLCOCK JONES, *Chairman.* ALVIN B. CARPENTER, *Vice-Chairman.*
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Bldg., Los Angeles, Cal.
A. B. W. HODGES, R. A. PEREZ,
E. A. MONTGOMERY, WILLIAM F. STAUNTON.

Colorado

L. P. HAMMOND, *Chairman.* F. H. BOSTWICK, *Vice-Chairman.*
P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
G. A. KENNEDY. M. S. MacCARTHY.

Montana

J. L. BRUCE, *Chairman.* W. C. SIDERFIN, *Vice-Chairman.*
WALTER E. GABY, *Secretary-Treasurer*, 834 W. Granite St., Butte, Mont.
W. T. BURNS, N. B. BRALY.

San Francisco

Meets second Tuesday of each month.

T. A. RICKARD, *Chairman.* W. H. SHOCKLEY, *Vice-Chairman.*
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
E. A. HERSAM, H. W. YOUNG.

Pennsylvania Anthracite

R. V. NORRIS, *Chairman.*

CHARLES F. HUBER, *Vice-Chairman.* EDWIN LUDLOW, *Vice-Chairman.*
W. J. RICHARDS, *Vice-Chairman.* ARTHUR H. STORRS, *Vice-Chairman.*
PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP,
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

C. J. ADAMI, *Chairman.* HERMAN GARLICH, *Vice-Chairman.*
F. W. DeWOLF, *Vice-Chairman.* M. M. VALERIUS, *Vice-Chairman.*
WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
C. R. FORBES, F. D. RASH,

Chicago

CHARLES H. MacDOWELL, *Chairman.* LUTHER V. RICE, *Vice-Chairman.*
HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

Utah

C. W. WHITLEY, *Chairman.* WALTER FITCH, *Vice-Chairman.*
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
E. R. ZALINSKI, WILLIAM WRAITH.

Arizona

GERALD SHERMAN, *Chairman.*
NORMAN CARMICHAEL, *1st Vice-Chair.* B. BRITTON GOTTSBERGER, *2nd Vice-Chair.*
ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
W. L. CLARK, J. C. GREENWAY.
W. G. McBRIDE. FOREST RUTHERFORD.

Nevada

J. W. HUTCHINSON, *Chairman.* FRANCIS CHURCH LINCOLN, *Vice-Chairman.*
HENRY M. RIVES, *Secretary-Treasurer*, Reno, Nevada.
W. H. BLACKBURN, E. A. JULIAN,
EMMET D. BOYLE, JOHN G. KIRCHEN,
FREDERICK BRADSHAW, C. B. LAKENAN.
TASKER L. ODDIE.

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STANDING COMMITTEES

Executive

L. D. RICKETTS, *Chairman.*

GEORGE D. BARRON, W. L. SAUNDERS,
SIDNEY J. JENNINGS, BENJAMIN B. THAYER.

Membership

KARL EILERS, *Chairman.*

ARTHUR S. DWIGHT, LOUIS D. HUNTOON,
LEWIS W. FRANCIS, ARTHUR L. WALKER.

Finance

GEORGE D. BARRON, *Chairman.*

ALBERT R. LEDOUX, CHARLES F. RAND.

Library

E. GYBBON SPILSBURY, *Chairman.*[3]

KARL EILERS,[1] E. F. ROEBER,[4]
ALEX C. HUMPHREYS,[2] BRADLEY STOUGHTON.

Papers and Publications

BRADLEY STOUGHTON, *Chairman.*

EXECUTIVE COMMITTEE

KARL EILERS, GEORGE C. STONE,
JOSEPH W. RICHARDS, SAMUEL A. TAYLOR.
E. F. ROEBER,

J. L. W. BIRKINBINE,	HEINRICH O. HOFMAN,	ROSSITER W. RAYMOND,
WILLIAM H. BLAUVELT,	WALTER E. HOPPER,	THOMAS T. READ,
H. A. BRASSERT,	HENRY M. HOWE,	ROBERT H. RICHARDS,
DAVID H. BROWNE,	LOUIS D. HUNTOON,	L. D. RICKETTS,
WILLIAM CAMPBELL,	J. E. JOHNSON, JR.,	HEINRICH RIES,
R. M. CATLIN,	WILLIAM KELLY,	RENO H. SALES,
ALLAN J. CLARK,	JAMES F. KEMP,	ALBERT SAUVEUR,
FREDERICK G. COTTRELL,	CHARLES K. LEITH,	HENRY L. SMYTH,
NATHANIEL H. EMMONS,	ANTHONY F. LUCAS,	A. A. STEVENSON,
JOHN W. FINCH,	EDWARD P. MATHEWSON,	RALPH H. SWEETSER,
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Geology of the Warren Mining District

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(Arizona Meeting, September, 1916)

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I. INTRODUCTION

THE main object of this article is to present the results of observations by the Copper Queen Consolidated Mining Co.'s geological department as an addition to the already published reports on the Warren district. Since these observations relate mostly to the ore deposits, little will be found here on the general geology that has not been covered by F. L. Ransome's admirable work on the Bisbee Quadrangle.[1] A summary of this very important part of the study of the camp will be given, however, mainly to bring out the new facts discovered during the advance in underground development and the latest detailed study of structures and petrography of the mining area.

II. PHYSIOGRAPHY

The Mule Mountains, in which the Bisbee ore deposits occur, constitute a chain with a northwest-southeast axis rising abruptly on the sides from the Sulphur Springs Valley on the northeast, as shown in Plate 1, and the San Pedro Valley on the southwest. The range starts at about the International Boundary near Christianson's ranch, with low hills covering a width of 1 or 2 miles, and extending for about 3 miles to the northwest. Here the Gold Hill overthrust fault has caused an abrupt rise in elevation and from this point on the range widens out and the hills become very much more rugged. The highest point is about 6 miles farther. The total length of the range from the International Boundary is about 23 miles and its maximum width, about opposite the town of Bisbee, is 10 miles. The range finally ends at Government Draw, which is a pass about 2 miles wide, connecting the San Pedro and Sulphur Springs Valleys. Beyond this the Tombstone hills commence.

The range is divided into two parts by Tombstone Canyon, a deep canyon running through the southwest side of the range. It is along this canyon that the Borderland Route road takes its course, and in which the town of Bisbee is situated.

Geologically, also, as seen in Plate 1, the Mule Mountains are roughly divided by Tombstone Canyon into two parts. To the southwest is the pre-Cretaceous tract and to the northeast the Cretaceous. The difference in the two tracts is very marked physiographically. The pre-Cretaceous is very much cut up and is formed of rocks of diverse compositions, consisting of schists, granites, shales, and limestones. These rocks, when subjected to erosion, form a rugged topography with deep canyons and steep cliffs. The Cretaceous tract, on the contrary, is uniform in composition, constituted, for the most part, by soft sandstones, conglomerates,

[1] Geology and Ore Deposits of the Bisbee Quadrangle, *Professional Paper of the U. S. Geological Survey*, No. 21 (1904).

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and shales, which are but little faulted. The topography carved from them is characterized by gently sloping hills and draws with almost no cliffs or deep valleys in evidence.

III. INTRODUCTORY GEOLOGY

The oldest rocks of the range are pre-Cambrian schists and an intruded granite, which are separated from the overlying Paleozoic beds by a profound unconformity.

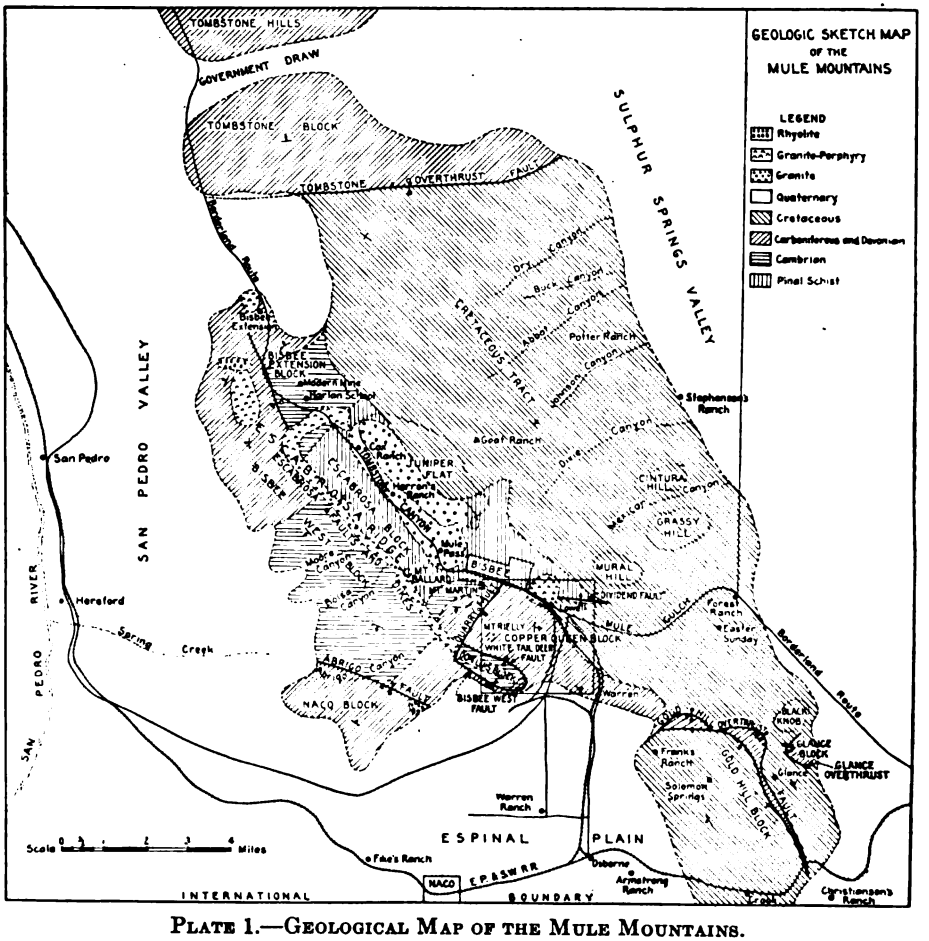

PLATE 1.—GEOLOGICAL MAP OF THE MULE MOUNTAINS.

The Paleozoic beds represent an era of apparently uninterrupted deposition of sediments, starting with 440 ft. of quartzites having a basal conglomerate, followed by Cambrian limestones, Ordovician-Silurian quartzite, Devonian and Carboniferous limestones. The total thickness of the Paleozoic beds is approximately 5,000 ft.

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At the end of the Carboniferous era a violent uplift took place accompanied by extensive faulting and intrusion of granite porphyry. During this uplift, and intimately associated with the intrusion, mineralizing solutions arose from which originated the orebodies of the camp.

Following this came a long period of erosion, which ended with a rapid subsidence, during which time 4,500 to 5,000 ft. of Cretaceous sandstones, shales, and limestones were deposited.

At the end of Cretaceous time a gentle uplift took place, lifting the range again above sea level. This was accompanied by some intrusion of rhyolite and monzonite.

During Tertiary and Quaternary ages, this land area was subjected to erosion, resulting in the present topography, the detritus filling in both the valley of the San Pedro and of the White River, one on each side of the range.

IV. ROCKS OF THE DISTRICT

A. Sedimentary Rocks

Reference to Plate 2, showing a generalized geologic section, will help make clear the following description.

Pinal Schist

Name.—The oldest rock of the district is the Pinal Schist. It is composed of a uniform series of thinly laminated siliceous mica schists of unknown thickness. From the similarity in texture and stratigraphic relation to the later sediments, it has been correlated by Ransome[2] with the underlying schistose complex of the Pinal Range, which he called the Pinal Schist.

Distribution and General Structure.—The Pinal Schist, being the basal crystalline rock upon which rest all the younger formations of the district, is exposed everywhere that erosion has stripped them off, and is encountered underground on sinking through them.

The best surface exposures are northeast of Mule Gulch and to the southwest of Tombstone Canyon. Smaller exposures are also found about 1 mile to the northwest of the town of Don Luis. Underground it is exposed where the Dividend Fault is cut, both in Copper Queen and in Denn ground. It also is found as dragged-in fragments in the contact-breccia mass around Sacramento Hill.

Lithology.—The color varies from light to dark gray, with tinges of green on fresh surfaces, and rusty in weathered specimens. The cleavage is rather imperfect, having a shiny satin-like surface.

The microscope shows the rock to be composed mainly of quartz and sericite. Sometimes the sericite is replaced in part by penninite, chlorite,

[2] *Loc. cit.*, p. 24.

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or serpentine. Folded into the schist, and forming an integral part of it, are occasional masses of diabase.

Origin and Age.—The age of the Pinal Schist is assumed to be the same as for the Pinal Schist of the Globe district which Ransome[3] has shown to be derived from arenaceous sediments, probably of Algonkian age. Both at Globe and at Bisbee a profound unconformity exists between it and the overlying Paleozoic beds.

COLUMNAR SECTION

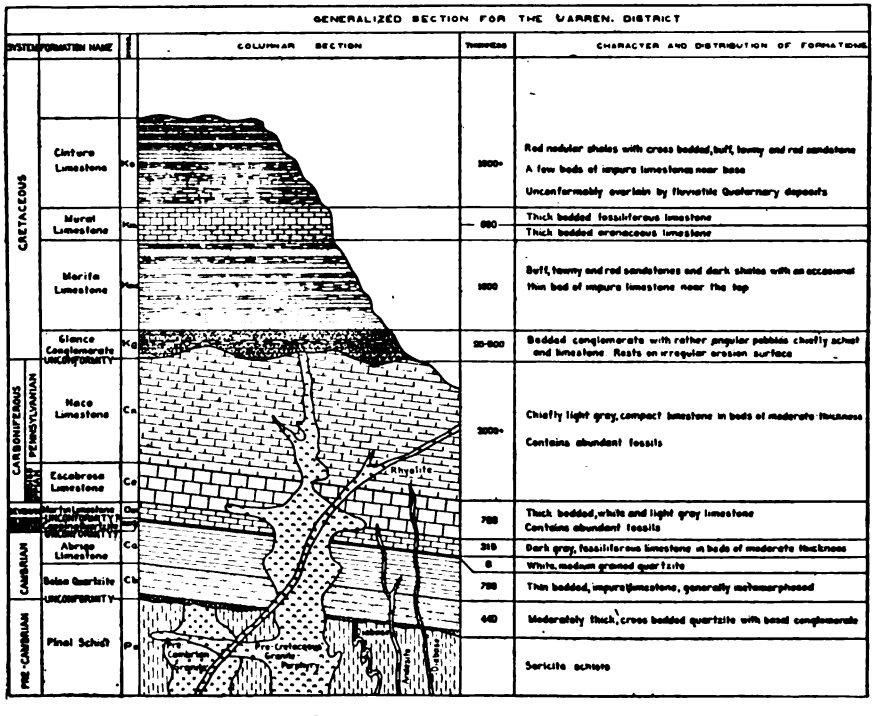

PLATE 2.—GENERALIZED GEOLOGIC SECTION SHOWING ROCK STRATA OF THE WARREN DISTRICT.

Bolsa Quartzite

Name.—The name of this formation, as well as those for all other formations in the district, were given by Ransome[4] and have been generally adopted.

Distribution and General Stratigraphy.—This Bolsa Quartzite lies on an evenly eroded plain of Pinal Schist. The main exposures of this formation are southwest of Escabrosa Ridge, especially in Bolsa, Abrigo, and Quarry Canyons. In the last-named canyon a complete unfaulted

[3] *U. S. Geological Survey*, Globe Folio, No. 111, p. 2.

[4] *Loc. cit.*, p. 28.

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section occurs. Underground, the formation has been cut by diamond-drill holes from the bottom of the Irish Mag shaft, and also by the workings from the Wade Hampton shaft.

Lithology.—At the base of the formation is a well-marked basal conglomerate, much iron-stained, containing pebbles consisting mostly of white, well-rounded quartz of up to a 4-in. diameter, and averaging about 1 in., the cementing material consisting of subangular sand. In the northwestern part of Tombstone Canyon some pebbles of porphyritic rock are found, together with some rounded crystals of feldspar. These feldspar crystals consist of pink microcline and microperthite, such as are found in the Juniper Flat granite less than a mile away. These feldspars, notwithstanding their being subject to such long exposure, are still relatively fresh.

The next 100 ft. of the Bolsa shows alternating beds of fine conglomerates and quartzites, with some arkose beds. Then follows a finer whitish quartzite with frequent cross-bedding, succeeded by some massive, dense beds of a maroon-colored quartzite. The total thickness of the Bolsa Quartzite as measured in Quarry Canyon is 440 ft.

Age.—No fossils have been found in the Bolsa except sparse worm tracks near the top, but as the next formation lies conformably on top of it and shows middle Cambrian fauna, the Bolsa has been assumed as lower or middle Cambrian.

Abrigo Limestone

Distribution and General Stratigraphy.—The Abrigo formation follows the Bolsa Quartzite conformably, and is the first fossil-bearing formation in the district.

The best exposure of Abrigo limestone is found on the northern slope of Mount Martin, where a complete unfaulted section is obtainable. Large areas of Abrigo are also found on the southwestern slope of Escabrosa ridge in Abrigo, Moon, and Bolsa Canyons, but the formation is here badly faulted. Small areas are also found in Quarry Canyon, north of Don Luis, and on the southern slope of the range from Mount Reilly to Gold Hill. As the beds are, for the most part, soft shales and shaly limestones, good exposures are rare, the formation being usually covered by talus.

Underground, the Abrigo formation is cut by the lower levels of all the mines from the Higgins to the Briggs, but in most of them only the top 100 ft. of rock has been cut. At the Spray, Oliver, and Irish Mag shafts, however, the formation has been cut by their lower workings to a depth of 400 ft. below its top, and diamond-drill holes from the Irish Mag shaft have penetrated into the Bolsa. At the White Tail Deer and Wade Hampton shafts the lower beds have also been cut. No complete section, however, has been exposed underground. Measured at Quarry Canyon the thickness of the Abrigo was found to be 798 ft.

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Geological section of Abrigo at Quarry Canyon:

Thin shales, greenish-gray to pink-brown, highly epidotised ..	0–72
Gray broken calcareous limestone, epidotised	72–83
Shaly calcareous sandstones	83–112
Medium-grained greenish-gray limestone with epidotised beds	112–132
Pure fine-grained pinkish-gray limestone	133–134
Arenaceous pinkish-gray shales	134–136
Purer fine-grained dark-gray limestone	136–139
Light-green shales	139–141
Fine-grained limestone, thin wavy bands of epidote	141–146
Green shales	146–151
Succession of impure lime and shales	151–183
Fine-grained gray limestone, beds of epidote ½ to 1 in. thick.	183–229
Purer light-gray limestone	229–245
Coarse sandy limestone with epidotised beds	232–245
Thin greenish shales	245–253
Coarse limestone, very fossiliferous (worm tracks)	253–261
Arenaceous green limestone	261–278
Coarse limestone, thin epidotised bands, some purer, 2-ft. beds forming steps on surface	278–412
Impure limestone green to brown, thin wavy bands of epidote very characteristic on surface by the weathering out of the epidote	413–497
Shaly arenaceous limestones	497–521
Coarse gray arenaceous limestone	521–629
Coarse light-gray crystaline sandy limestone	629–684
Coarse cherty-banded crystalline white bed-forming cliff	684–700
Sandy even-grained limestone and quartzite	700–798

The Abrigo formation consists of a series of shales and argillaceous and sandy limestones, with bands of epidote. These epidote bands weather out on exposure to the atmosphere, giving the formation a characteristic cherty banded appearance. In the top beds there are a few bands of true chert. Underground, this epidote appears as greenish bands giving a wavy banding which readily distinguishes it from all other formations, when the beds are fresh. About 100 ft. from the top there exists a 16-ft. bed of pure white limestone which forms a rather prominent cliff when encountered on the surface.

Capping Quartzite

Lying on top of the Abrigo, and with no apparent angular unconformity, is a persistent bed of pure quartzite varying in color from white to dark red, and varying from 6 in. to 12 ft. in thickness, and locally called the Capping Quartzite.

Distribution.—This quartzite being more resistant than the underlying and overlying beds, is exposed wherever the Abrigo outcrops, as a small cliff or shelf. One well-marked exposure is found to the east of Black Gap, where it runs up the side of the hill apparently as a siliceous dike. Other good exposures are found on Mt. Martin, and southwest of Escabrosa Ridge.

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Underground, this bed is important as a marker. Due to its character, it is resistant to metamorphism, and may be recognized easily even when the rest of the sediments have been completely changed. Its thinness, however, tends to make it elusive, as a small fault frequently will throw it out of an otherwise perfect section.

Age.—As one travels north through the State, this formation becomes thicker, and in the Globe district it attains a thickness of 500 to 700 ft., while that of the underlying limestone has decreased to 200 ft. Here there is a well-marked evidence of an unconformity between the limestone, which is capped by a flow of basalt of varying thickness, and the overlying Troy Quartzite, which is conglomeratic at its base. While no fossil evidence is available, its relation to the overlying and underlying formations seems to indicate that it is probably of Silurian age, and that the Ordovician is represented by a period of erosion, when the area studied was very little above sea level, since there is no apparent angular unconformity below or above this formation.

Martin Limestone

Lying conformably above the Capping Quartzite is 300 ft. of limestone known as the Martin limestone.

Distribution.—The exposures of the Martin limestone are widespread, especially along Escabrosa Ridge, north of Moore Canyon, in Abrigo Canyon, Escacado Canyon, and on the southern slopes of the hills east of Mt. Reilly. Owing to the fact that the overlying Escabrosa limestone is a prominent cliff-forming bed, the exposures are apt to be talus-covered. The best complete sections are obtained on the northern slope of Mt. Martin, and to the east of Black Gap. Underground, it is the formation most frequently cut of any in the ore zone. For this reason it is usually difficult to recognize from its lithological characteristics. The formation is so rich in fossils, however, that even when much altered, a diligent search will yield some trace of them. The formation is encountered underground in every mine in the district except the Wade Hampton, which is entirely in Cambrian, and the Denn which did not sink deep enough to strike it. The outcrop of this formation was first found in the Queen opencut near its top, and it has been encountered at progressively lower levels from here down to the 1,600-ft. level of the Lowell, and the 1,500-ft. level of the Briggs.

Lithology.—The Martin limestone is the most constant formation, both in respect to thickness, and character, of any in the district. Plate 3 shows a comparison of sections obtained over widely scattered parts of the district. As is seen, the formation as a whole is made up of fairly dense dark-gray limestone with occasional shaly and sandy members, but all with enough lime present to be called limestone. It is all more or less fossiliferous, and some beds are extremely so. The characteristic fossils are three brachiopods, one a spirifer, one an atrypa, and

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one a schisophoria, which all attain a length of about ⅓ to ½ in.; they tend to weather out on both surface and underground exposures. When the rock is altered, these fossils tend to resist metamorphism to a

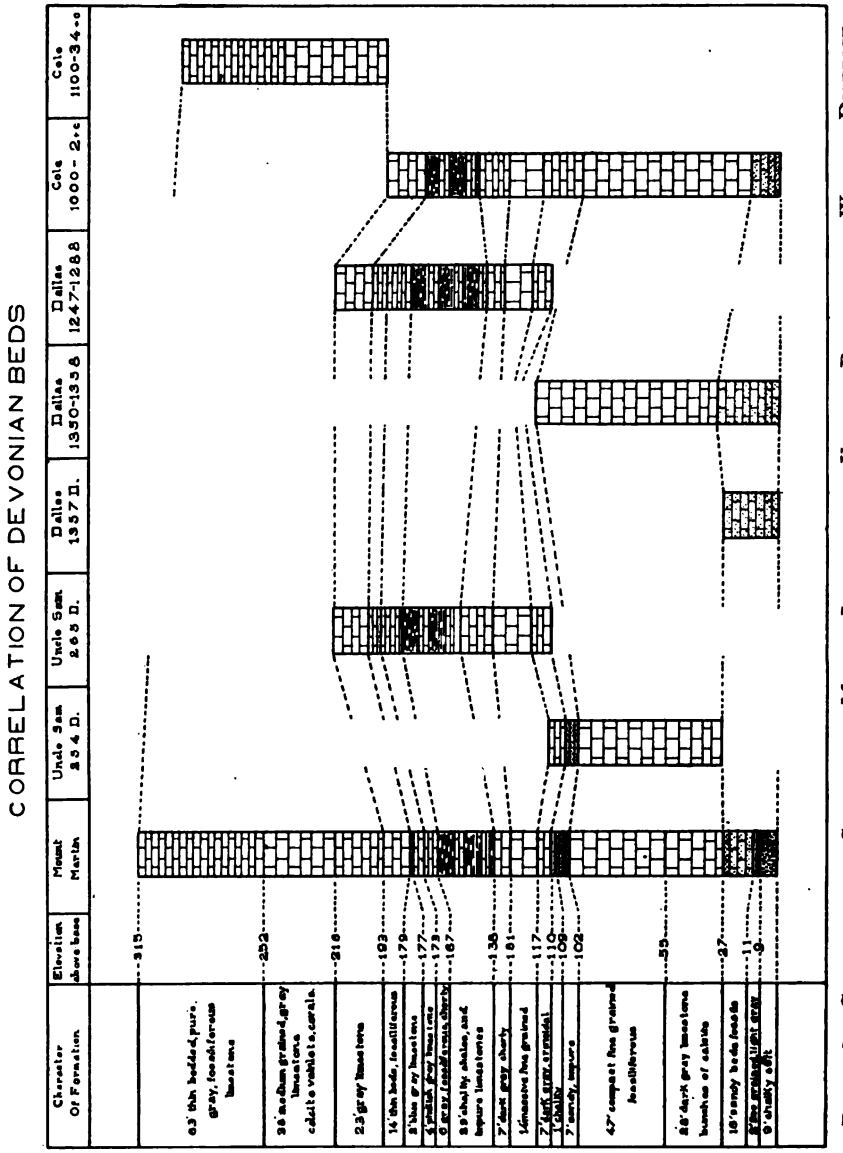

PLATE 3.—COMPARISON OF SECTIONS OF MARTIN LIMESTONE IN VARIOUS PARTS OF THE WARREN DISTRICT.

remarkable degree, and it is not at all rare to find almost perfect fossils left in a soft mass of completely altered limestone. These fossils, however, are frequently found completely pyritized when the surrounding

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limestone is untouched. In one case, within a completely oxidized orebody of the Martin, fossils were found almost perfect in form but completely replaced by azurite. Fortunately, the fossiliferous beds themselves seem to be more easily replaced by ore solutions, so that where metamorphism is most complete, some fossil evidence may usually be found.

The upper beds of the formation have few brachiopods, but are almost made up of corals. On altering, these beds tend to take a pseudo wavy banding, which makes it difficult to distinguish from the Abrigo. The succession of beds in these cases is the only determining factor.

The Martin limestone is not a pure limestone by any means. Almost the whole series is more or less dolomitic, more so than any other formation in the range. It is also much more aluminous than a superficial examination would lead one to suppose. The silica content is also high. Characteristic partial analyses of unaltered specimens of Abrigo and Martin limestones from both surface and underground exposures are given in Table 1. These determinations were made at the Copper Queen assay office.

TABLE 1.—*Analyses of Unaltered Abrigo and Martin Limestones*

	Abrigo					Martin							
	Mt. Martin Section Underground												
	Shales	72–232 ft. above base	232–521 ft. above base	521–700 ft. above base	Cherty Limestone 1,357 Drift	Black Gap[4]	500 Level Uncle Sam Mine[2]			265[3] Drift	1,300[1] Dallas	1,200[2] Dallas	1,000[4] Cole
SiO_2	68.6	30.8	35.3	16.1	49.4	10.3	39.4	46.0	21.7	15.5	45.5	35.5	25.2
Al_2O_3	9.1	4.0	3.0	2.1	4.5	2.0	9.1	10.8	6.5	3.5	12.9	5.0	5.9
MgO	4.6	3.6	2.3	1.7	2.4	14.2	8.8	4.4	13.8	22.9	3.8	5.4	8.5
CaO	9.8	33.5	31.5	38.9	17.8	28.5	6.0	10.5	17.8	10.7	13.1	26.7	24.7
Fe	5.1	2.7	2.8	1.5	2.2	1.2	3.7	3.2	2.0	1.8	3.0	2.1	1.8
S	...	....	....	0.2	0.6	0.2	1.5	0.4	0.7	1.3	1.9	0.8	0.8
Mn	...	....	....	....	....	....	0.6	0.4	0.4	0.3			

[1] Bottom beds. [2] 100 ft. above Capping Quartsite. [3] Somewhat metamorphosed. [4] Composite of section. [5] 200 to 300 ft. above Capping Quartsite.

As is seen, the two formations are of about the same purity, but the Abrigo contains more silica than the Martin. The Martin, however, is much more dolomitic, and the lower beds are very shaly. The composite from Black Gap shows that on the whole it is a fairly pure dolomite, with only 13 per cent. of silica, alumina, and iron. The impure beds are chiefly at the bottom 100 ft. and the top 50 ft. of the formation.

Age.—The age of the Martin has been well established by good fossil evidence as Devonian. The best exposition on the correlation of these beds is found in the previously mentioned Ransome's report on the Bisbee Quadrangle.

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Escabrosa Limestone

Distribution.—The best surface exposures of this formation are on Escabrosa Ridge from Mt. Martin to Mt. Reilly, and on the southern slopes of the hills from Mt. Reilly to Gold Hill. The formation again appears to the northward on Escabrosa Ridge southwest of the Modern mine, and an excellent section is obtained from where the Borderland Route turns out of Tombstone Canyon into the San Pedro Valley. Underground, the formation is cut by all the shafts from the Czar to the Briggs. It is usually in the oxidized portions of the mines, and as a consequence exposures are not very good.

The best exposures found are at the Junction and Sacramento mines. At the Junction, workings cut the formation from the top to the bottom. Here, except for marbleization, the exposures are good, as oxidation has not penetrated nearly as deep as in the other mines of the camp. At the Sacramento, exposures are obtained of the bottom 300 ft., but marbleization has so altered the formation that they are not good.

Lithology.—The Escabrosa and Naco formations represent really one period of deposition during Carboniferous time. The division of the formation as a whole into Escabrosa and Naco is largely an arbitrary matter. Roughly it is divided, from the fossil evidence, into Mississippian and Pennsylvanian, the former locally called Escabrosa, and the latter Naco. This line of subdivision lies at approximately 800 ft. above the top of the Martin formation. In this paper the thickness of the Escabrosa has arbitrarily been taken as 768 ft. irrespective of fossil evidence. Following is a detailed section of Escabrosa taken at Mt. Martin and northeast of the White Tail Deer shaft.

	Feet
Coarse crystalline pink non-fossiliferous limestone	0–6
Pinkish-white to gray medium-grained 6-in. to 12-in. beds	6–60
Massive grayish-white coarse fossiliferous cliff beds	60–140
Thin gray beds medium fine-grained fossiliferous	140–145
Coarse massive grayish fossiliferous cliff bed	145–185
Thin fine-grained fossiliferous gray beds	185–312
Fine-grained grayish-white cherty banded (½ in. to 3 in.)	312–479
Coarse massive white cliff beds fossiliferous	479–556
Fine-grained thin-bedded fossiliferous	556–608
Brown cherty fossiliferous limes	608–616
Grayish fine-grained thin-bedded fossiliferous	616–648
Pinkish beds medium-grained with some chert	648–686
Fine-grained thin-bedded soft grayish	686–758
Massive dark-gray medium-grained	758–768

The Escabrosa limestone rests conformably on the Martin limestone wherever exposed. The 6-ft. pink non-fossiliferous bed forming the bottom member serves in unaltered ground as an excellent marker. The pure coarse crystalline bed following is the most commonly exposed

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member of all, forming as it does the steep and rugged cliffs of Escabrosa and Higgins Ridges. At the Mt. Martin and the White Tail Deer sections, it contains very little chert and abundant crinoid stems. Underground exposures appear to contain considerably more chert in this horizon. In fact, the presence or absence of chert beds seems to be very local, and untrustworthy in identifying any particular horizon. Above this first 180-ft. cliff-forming horizon is a soft series of more dense fine-grained beds with some chert, followed by 127 ft. of extremely cherty banded medium-grained limestone, which seems to be very persistent. Above this is a second thinner cliff-forming horizon 77 ft. thick, followed by dense, relatively impure limestone, varying in color from brown to gray with quite abundant chert nodules. There is no definite marker between the Escabrosa and Naco. As previously stated, the Escabrosa is given arbitrarily a thickness of 768 ft.

Naco Limestone

Distribution.—This limestone is generally exposed on the top of the hills from Queen Hill to the east, and extends over the productive area from the Irish Mag shaft continuously east until covered by the later conglomerate. Good exposures of Naco are also found at the northermost range of hills just south of Government Draw, and on Escabrosa Ridge from the Modern mine, to the northwest, where it is finally covered by Quaternary wash. The best and least faulted section of the formation is found in the Naco Hills about 3 miles west of the town of Don Luis. Underground it is cut by all the shafts from the Irish Mag eastwards. Practically no drifts, however, have been driven in this horizon, except possibly at the upper levels of the Junction mine, and the 200-ft. level of the Lowell.

Lithology.—A thickness of 1,500 ft. has been measured in the Naco Hills. Assuming that the formation has been half stripped off we get a probable former thickness of at least 3,000 ft. Compared with the Escabrosa limestone, it is generally thinner bedded, and very much finer grained, and contains some distinctly shaly members, especially in the upper portions. Some coarse-grained beds do exist, however, which without fossil evidence would be impossible to distinguish from the Escabrosa. Abundant crinoids exist as in the Escabrosa, but large brachiopods and cephalopods are much more in evidence. The most common and characteristic Naco fossil is a large-sized member of the productus family.

Cretaceous Deposits

Glance Conglomerate.—Resting on an extremely uneven erosion surface of Paleozoic beds, schist and granite, is deposited the Glance Conglomerate. This formation is exposed over almost the whole of the southern end of the range.

Lithology.—This conglomerate is made up of partially rounded frag-

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ments of schist, porphyry, both fresh and sericitized, quartzite, and limestone. These fragments vary in size from ½ in. to 3 ft. diameter and are loosely compacted. Almost no cementing material exists. The conglomerate as a whole is stained a deep brown. The thickness varies from nothing up to 1,000 ft., depending on the old erosion surface. Where this old surface was fairly well worn down, as north of the Dividend Fault and on top of the Juniper Flat granite mass, the formation is a typical basal conglomerate with rounded pebbles cemented with sand.

Age.—The age of the formation is either late Jurassic or early Cretaceous.

Morita, Mural and Cintura Formations.—For a more detailed description of the Cretaceous beds above the Glance Conglomerate, Ransome's report may be referred to. As these beds occur outside the productive area, no additional study was made of them. The Morita consists of a series of buff and tawny sandstones, shales and calcareous sandstones, containing sparse fossil remains, and measuring 1,800 ft. in thickness. This is overlain conformably by 650 ft. of limestone with a 50 to 200-ft. member of pure limestone containing abundant Cretaceous fossils.

The Mural is followed by the Cintura formation, consisting of beds almost identical with those of the Morita, with a thickness of 1,800 ft., plus an unknown amount eroded off. These Cretaceous beds cover about two-thirds of the range, nearly the whole area northeast of Tombstone Canyon and Mule Gulch, and are finally buried beneath the Quaternary wash of Sulphur Springs Valley.

B. Igneous Rocks of the District

The most important igneous rocks of the district, those of granitic composition, have been grouped together in the United States Geological Survey publications on this region, and have all been ascribed to the same magma and general period of eruption, although differences in composition, texture and field relations were noted. Notman[5] has already separated the granite and the porphyritic rocks by placing the first in the pre-Cambrian, and Boutwell has found a basic variety of porphyry underground and on surface, which corresponds probably to the post-Cretaceous dike described by Ransome from the Glance mine,[6] this latter being a rock easily distinguished from the other porphyry.

Careful search for field and petrographic evidence has yielded results which allow the following classification of the igneous rocks of the Warren district.

Pre-Cambrian Granite

On Juniper Flats and along the north side of Tombstone Canyon, a mass of granite is exposed which forms the center of the Mule Moun-

[5] *Transactions of the Institute of Mining and Metallurgy*, vol. 22, pp. 550–562 (1913).
[6] *U. S. Geological Survey, Professional Paper* No. 21, p. 84 (1904).

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tains. The rock is a pink or purplish-gray coarse-grained granite whose component minerals are quartz, orthoclase, microcline, a little biotite and plagioclase feldspar. Accessory minerals can be seen only under a microscope in thin section, such as apatite, magnetite, zircon and tourmaline. The orthoclase contains microperthitic intergrowths of albite, and dusty inclusions of iron oxides.

This rock corresponds exactly to that described by Ransome as the Juniper Flats Granite, an analysis of which is given in *Professional Paper* No. 21 *of the United States Geological Survey*, as follows:

	Per Cent.		Per Cent.
SiO_2	75.86	Na_2O	3.60
Al_2O_3	12.17	K_2O	5.04
Fe_2O_3	0.85	H_2O-	0.27
FeO	0.36	H_2O+	0.72
MgO	None	T_1O_2	0.21
CaO	0.62		
			99.70

There are on Juniper Flats some porphyritic alaskite dikes that seem to be directly related to the granite, but aside from these, all other porphyritic rocks here have not been ascribed pre-Cambrian age.

Field Relations.—The Juniper Flats granite has been found cutting only Pinal Schist, and while it has not been found directly overlain by Paleozoic sediments but covered directly by Cretaceous, pebbles of microcline and of microperthite granite have been found at the base of the Cambrian quartzite. This granite nowhere cuts the Paleozoic sediments, but it is itself extensively intruded by dikes and irregular masses of the later porphyries that do penetrate the sedimentaries. These observations, as well as the similarity of this granite to other pre-Cambrian granites in Arizona has led to the belief that the intrusion took place before the beginning of the Paleozoic.

In the area economically exploited in the district for copper, the Juniper Flats granite has not been found, and the only ore associated with the rock comes in fissure veins filled with quartz and some fluorite with pockets rich in gold.

Pre-Cretaceous Granite Porphyry

The most important igneous rock in the district is the granite porphyry that is associated with most of the orebodies, and whose largest outcrop is the mass of Sacramento and Copper King Hills in Bisbee.

Distribution.—On surface this rock is found abundantly in the southwest, or older portion, of the Mule Mountains, but the distribution is by no means even. In the schist, granite, and Cambrian formations there are abundant masses of porphyry in the form of dikes and sills, while in the Carboniferous only a few well-marked dikes are found besides the main stock of Sacramento Hill.

In Juniper Flats and on Escabrosa Ridge there is abundant granite

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porphyry, some of the dikes cutting granite, schist and Paleozoics in their course from the north to the south side of Tombstone Canyon. On one of the ridges close to the mouth of Tombstone Canyon, about opposite the old Modern mine there is a continuous sill of granite porphry several hundred feet thick in Carboniferous limestones. This sill is remarkable because it is cut by dikes of rhyolite, and because on an old erosion surface of the granite porphyry and Carboniferous limestones a considerable thickness of rhyolitic lavas has accumulated.

The character of these later dikes and flows is exactly the same as that of the volcanic plug with flow structure which is found in the Naco hills and is mentioned by Ransome in his report on the district. The relations in the locality at the mouth of Tombstone Canyon leave little doubt as to the great difference in age between the granite porphyry and the rhyolite, as the erosion surface on which the rhyolite flowed is probably post-Cretaceous.

In the mining area of the district the Sacramento Hill stock has penetrated between the Paleozoic sediments on the south side of the Dividend fault and the schist on the north. The outline of the stock is irregular, and extends about a mile from north to south and an equal distance along the fault. The only other porphyry outcrop of any size in the mining area is the Shattuck dike, which runs from above the Spray mine for more than a mile westward. Another important dike, the Sacramento, outcrops a few isolated places in the Naco limestone, southward from Sacramento Hill almost to the Briggs mine.

In the underground mine workings, the relative abundance of porphyry intrusions in the formations lower than the Carboniferous holds about the same as on surface. Besides the main mass of the Sacramento Hill intrusion there is another smaller, laccolithic mass of porphyry around the Lowell mine. This body is connected with Sacramento Hill and Sacramento dike by masses of porphyry, none of which reach the present surface, but which spread out mostly at the Carboniferous-Devonian horizon. This mass, with its accompanying contact breccia is shown in Plate 6.

The mode of intrusion of the granite porphyry and its relation to the structure of the older rocks will be touched upon later. It will be enough to state here, however, that the intrusions took place in one general period, but not all at once, nor were the conditions the same at the beginning and end of the eruptions, or the composition of the rock exactly the same. Considerable alteration was also started in the rocks before the last porphyry was injected.

Lithology.—Although there is abundant fresh granite porphyry in the district most of the rocks around the ore zones have been so altered by mineralizing solutions that it is difficult to find material which represents the original granite porphyry of the main masses associated directly with the ores. Fortunately, the alteration processes have been so varied

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that they have sometimes left untouched some of the phenocrysts, at other times the base, so that from various sources a reconstruction can be made of what was for a short time the rock of Sacramento Hill and the Lowell mass. Of the dikes fresh rock is available.

The pre-Cretaceous granite porphyry is, when fresh, a light-gray rock weathering to a light-red color on surface. It has abundant phenocrysts, up to ½ in. in diameter of feldspar, quartz and biotite, the relative amounts of these being variable. The base is a fine-grained mass, almost glassy at times and composing from one-half to about eight-tenths of the rock. The microscope shows that the rock varies from a typical granite porphyry to a rather basic monzonite porphyry, sometimes in the same sill or dike. The main masses were probably of the more acid variety, with well-developed quartz phenocrysts that have embayments due to reabsorption. There are no noticeable inclusions in the quartz. In the more basic varieties of the porphyry, quartz phenocrysts become extremely scarce.

The feldspar phenocrysts are mostly orthoclase with albite and oligoclase increasing as the rock becomes more basic. The orthoclase is in large crystals when it predominates and in small ones when plagioclases are abundant. The only intergrowth of the orthoclase seen is in the form of a micropegmatite with quartz, and at the rim of some large crystals in the dikes where an aureole of quartz and orthoclase usually forms around a pure orthoclase of quartz center. The plagioclases are all between albite and oligoclase and sometimes are in greater abundance and size than the orthoclase.

Biotite is the only ferromagnesian mineral of any importance and comes in large well-formed crystals as well as fine shreds. In most of the porphyry it has been altered before any other minerals, and its traces cannot be distinguished megascopically, but in some of the more basic varieties, which are also chloritized, biotite is the most conspicuous phenocryst mineral and gives a decidedly dark color to the rock.

The base of the porphyry is usually a cryptocrystalline mass of quartz and feldspar, found sometimes with a micropegmatitic intergrowth, at other times with incipient spherulitic growth and a few times with a holocrystalline but extremely fine crystallization.

As accessories, there are: apatite in well-formed needles sometimes noticeably abundant, and generally in the biotites; very few grains of pyrite, and dusty undeterminable inclusions in the feldspars.

The foregoing description of the unaltered porphyry is rarely applicable to the rock encountered in the mining area, as the porphyry may be found in the form of a white mass of sericite and quartz, or a dense dark-green rock composed mostly of serpentine and penninite, with all variations between. These characteristics will be considered later under metamorphism and mineralization.

Age.—The granite porphyry distinctly cuts the Pinal Schist, granite

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and all the Paleozoic sediments, while pebbles of the altered sericitized and oxidized porphyry are found at the base of the Cretaceous Glance Conglomerate. It is, therefore, certain that the age of intrusion is post-Carboniferous and pre-Cretaceous. That the porphyry intrusion did not take place all at once, is shown by dikes of the slightly more basic varieties cutting the contact breccias of the more acid ones. This difference in age is further emphasized by differences in alteration, but, as will be shown later, there is no reason to believe that there was any great interval of time between the porphyry injections, but that they all belong to the same general period.

Rhyolite

This rock has not been distinctly recognized in the mining area of the Warren district, but due to its great similarity to the granite porphyry it may have been confused with the older rock, especially where both are found in dikes. The rhyolite from the mouth of Tombstone Canyon, whose field relations have already been given, is a gray to pink rock with decided flow structure in its surface forms. Phenocrysts are not abundant and are seen under the microscope to be entirely quartz and orthoclase, in an aphanitic or cryptocrystalline groundmass. Some of the dikes on Escabrosa ridge, and others northeastward from the mouth of Tombstone Canyon, have such a glassy groundmass, and so few phenocrysts of quartz and orthoclase, that they may very well be of the late rhyolite, especially as they are seen to cut other porphyritic dikes in places. The rhyolite of the Naco Hills described by Ransome is probably of the same age as that at the mouth of Tombstone Canyon.

Age.—Unfortunately no information is available at present for placing the age of the rhyolite any closer than post-granite porphyry. An erosion period separates the two, but whether it is the pre-Cretaceous or post-Cretaceous one cannot be definitely stated.

Other Dike Rocks

Hornblende Andesite.—At a few scattered places on the surface, and in the Shattuck, Cole, Wade Hampton and Wolverine mines, there are dikes of a dark-green, fine-grained rock, which is seen under the microscope to be composed of abundant small phenocrysts of andesine and labradorite and a few of hornblende, in a microcrystalline base of feldspar and fine magnetite grains. This andesite cuts all the pre-Cretaceous formations, including the granite porphyry, but its relation to the ores is not clear. The rock is considerably altered, and chloritic minerals especially penninite, as well as some pyrite, have been developed. In a specimen from the Wolverine shaft there is also some sericitization, so it is possible that the andesite may antedate the last of the mineralization period.

Diabase.—Olivine and augite diabase in small masses and dikes have

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been found to belong to two distinct periods. The earlier is folded in with the Pinal Schist and is pre-Cambrian, while the latest cuts the Carboniferous limestones and is entirely unaltered. There is no known exposure of them underground.

Monzonite Porphyry.—The dikes of this rock mentioned by Ransome as occurring around the Glance mine have not been studied during the preparation of this report, and no similar rock has been found in the mining area, unless it is the one classified as andesite porphyry, or the basic porphyry found by Boutwell.

V. STRUCTURAL GEOLOGY

General Structure of Mule Mountains

Fault Blocks and Other Divisions

In this report the structural geology will be confined almost entirely to the southwestern part of the range covered by pre-Cretaceous sediments, as these play the most important part in the relation to the ore-bodies of the district. Reference to the areal geology map, Plate 4, and sections made along its lines *A–A*, *B–B*, and *C–C* shown in Plates 5, 6, and 7, will help make clear the following description:

As previously stated, the range is roughly divided by Tombstone Canyon and Mule Gulch into two parts. To the northeast, and forming about two-thirds of the range, Cretaceous sediments cover the whole area, dipping gently to the northeast and north, and being very little faulted. Where cut by the above-mentioned canyons, they overlie a nearly level plain of schist and granite. This area will be alluded to as the Cretaceous Tract.

The area southwest of the two canyons is almost entirely made up of much faulted and intruded schists and Paleozoic sediments, covered at the southeastern end by Cretaceous sediments overlying a rough topography of Paleozoic beds. It is this area which is studied in detail in this report.

Referring to the accompanying sketch map (Plate 1) of the range, it is seen that in a general way the dips of the Paleozoic beds radiate around the Juniper Flat granite mass. At the northwestern end they dip to the north and northwest. At Mt. Martin and Mt. Ballard, the dip swings to the west and southwest. From Mt. Martin to the southeast, the dips change from south to due east. Each change in dip and strike, however, is generally marked by a fault, the beds seeming to withstand folding to a remarkable degree.

In our division of this area into fault blocks, the divisions given by Ransome[7] have been generally followed, with a few minor exceptions and additions.

Starting at the north is the Bisbee Extension Block with beds dipping to the northwest, and disappearing finally to the north under Quarter-

[7] *Loc. cit.*, p. 93.

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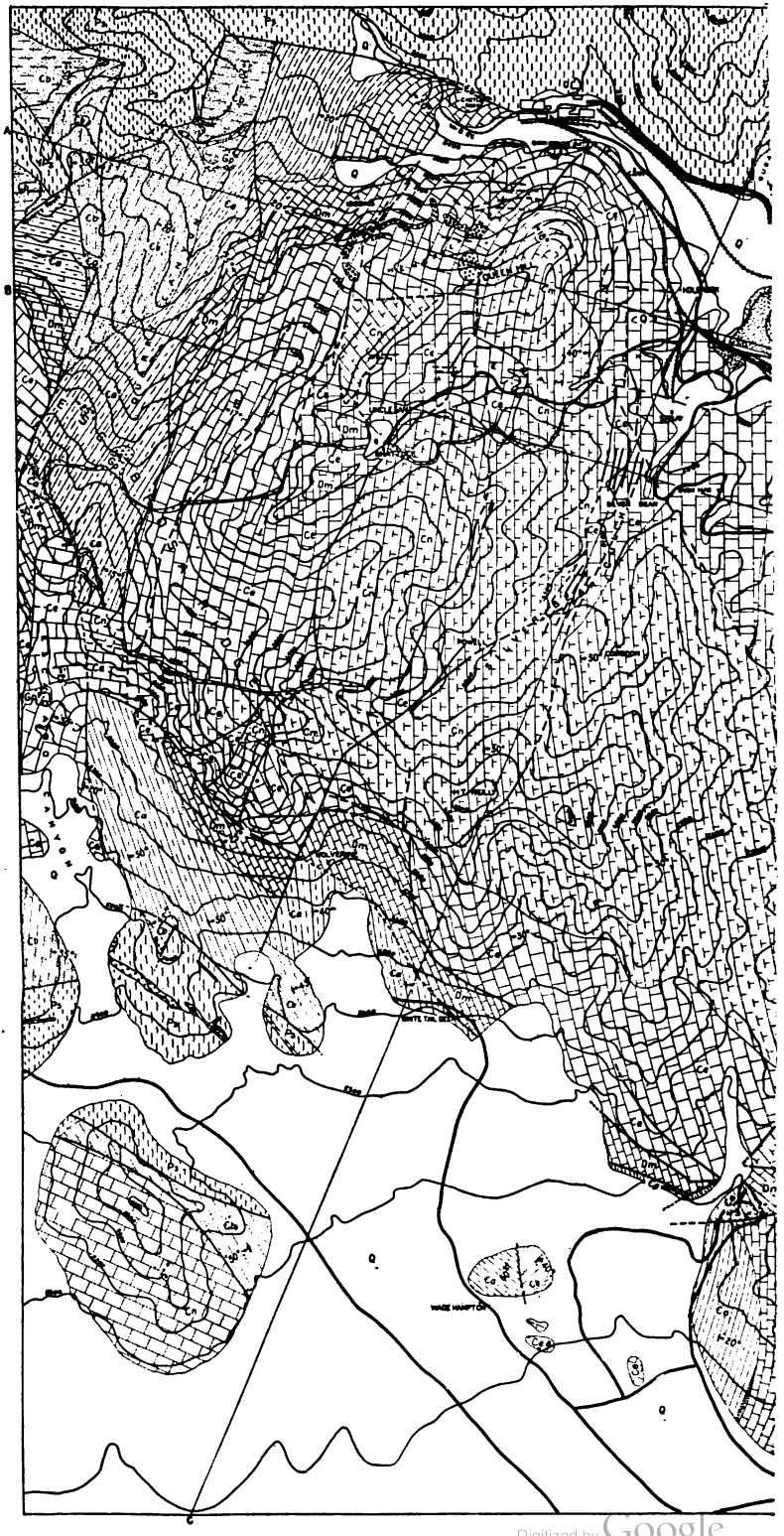

PLATE 4.—AREAL GEOLOGY MAP WITH SECTION LINES A–A, B–B

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AND C-C RELATING TO THE SECTIONS SHOWN BY PLATES 5 TO 7.

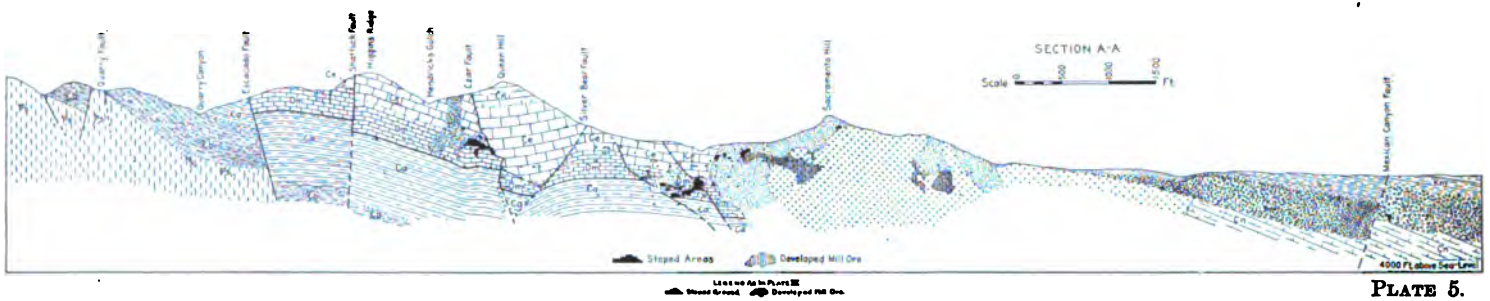

PLATE 5.

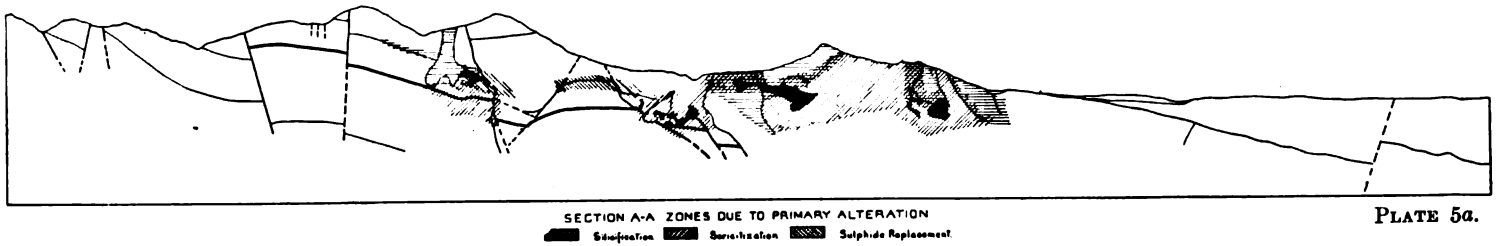

PLATE 5a.

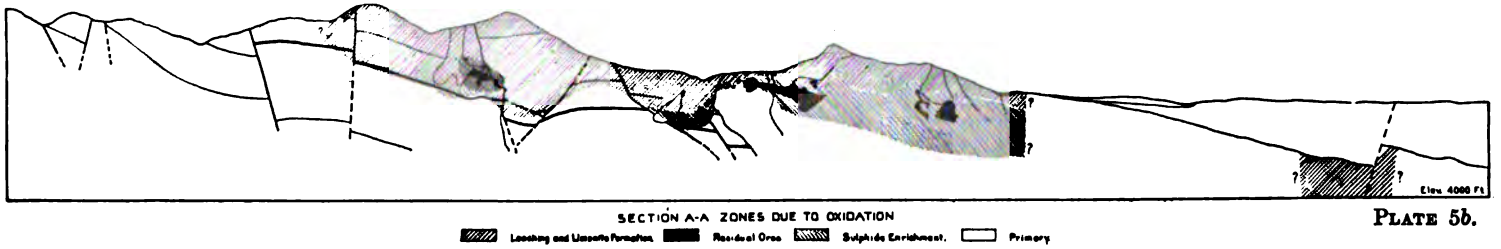

PLATE 5b.

PLATES 5, 5a, 5b.—GEOLOGICAL SECTION ALONG LINE A–A OF MAP, PLATE 4.

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nary wash. This block is bounded on the south by the Modern stockwork of granite porphyry, cutting through the schist and Paleozoic beds and striking northeast and southwest.

To the south of the Bisbee Extension Block are two blocks, the Escabrosa, extending from the Juniper Flat granite mass to the Escabrosa Ridge line of faults and porphyry intrusions, and the Bisbee West Block extending from Escabrosa Ridge to the southwest. The Escabrosa Block is up-thrown with respect to the Bisbee West Block, and consists almost entirely of schist and porphyry, until Mt. Martin is reached. Here we have the whole Paleozoic section exposed up to the middle of the Escabrosa limestone. The dips at Mt. Martin are southwest. The Bisbee West Block is covered at the northwest by Carboniferous beds dipping to the northwest and west. These are followed to the south by Cambrian beds, dipping to the west and southwest. The Escabrosa Block is bounded on the southeast by the Quarry Fault, beyond which are the Copper Queen and Don Luis Blocks. The Bisbee West Block is bounded on the south by the Abrigo Fault, to the south of which is the Naco Block; relatively very much down-thrown. To the northeast it is bounded by the Bisbee West Fault, to the north of which is the relatively up-thrown Don Luis Block.

The Copper Queen Block is down-thrown with respect to all the surrounding blocks. It is bounded on the north by the east and west Dividend Fault, which has a throw of at least 1,500 ft., throwing the Pinal Schist to the north against Paleozoic beds to the south. It forms a block with the Quarry Fault, a northeast to southwest fault which is the western boundary of the block. The Quarry Fault in turn, forms a block with the White-Tail Deer Fault which is nearly parallel to the Dividend Fault and throws Cambrian beds and Pinal Schist to the south against Devonian and Carboniferous beds to the north. The block is finally covered to the east by Cretaceous beds which are in turn covered by Quaternary wash. To the southeast, the block ends abruptly against the Gold Hill Overthrust Fault. The dips in the Copper Queen Block are generally to the east and northeast.

The Don Luis Block is a small up-thrown block between the Copper Queen and Bisbee West Blocks. The larger part of this block is covered by Quaternary wash, with numerous outcrops of schist and Cambrian beds. The dips in this block are variable but generally to the eastward.

In addition to these are four other blocks, three of which are evidently post-Cretaceous, and one probably so. These are the Tombstone Overthrust Block at the northwestern end of the range thrusting Naco limestone over Morita sandstone; the Naco Block bounded on the north by the Abrigo Fault; the Gold Hill Overthrust Block; and the Glance Overthrust Blocks at the southeastern end of the range.

The Cretaceous Tract, except where the sediments overlie the pre-

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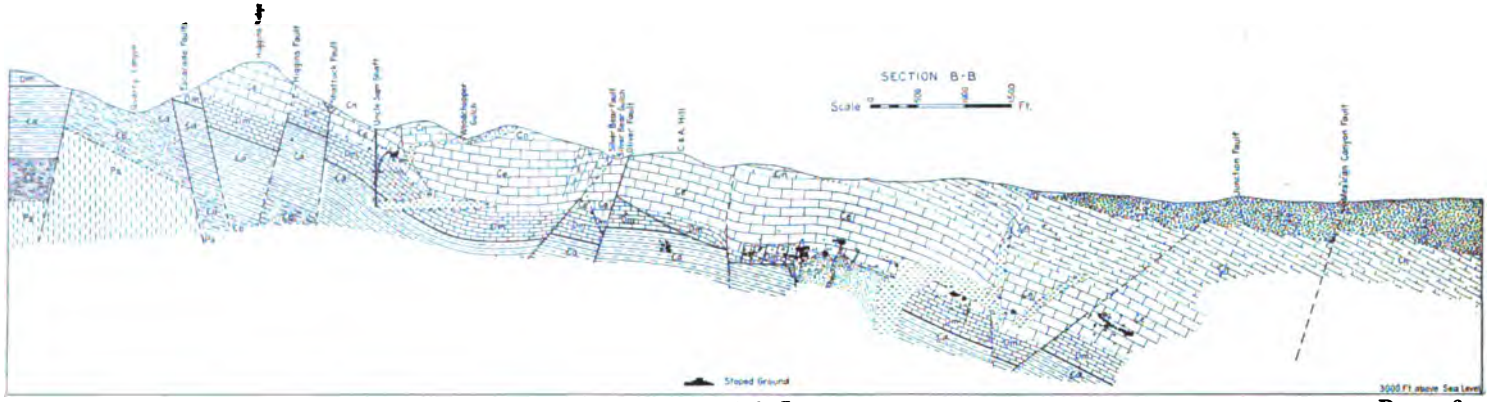

PLATE 6.

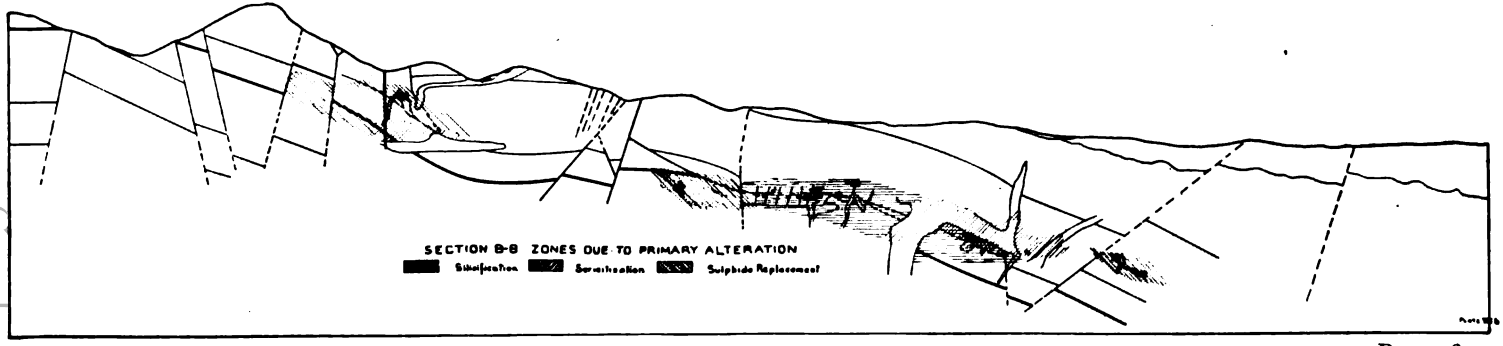

PLATE 6a.

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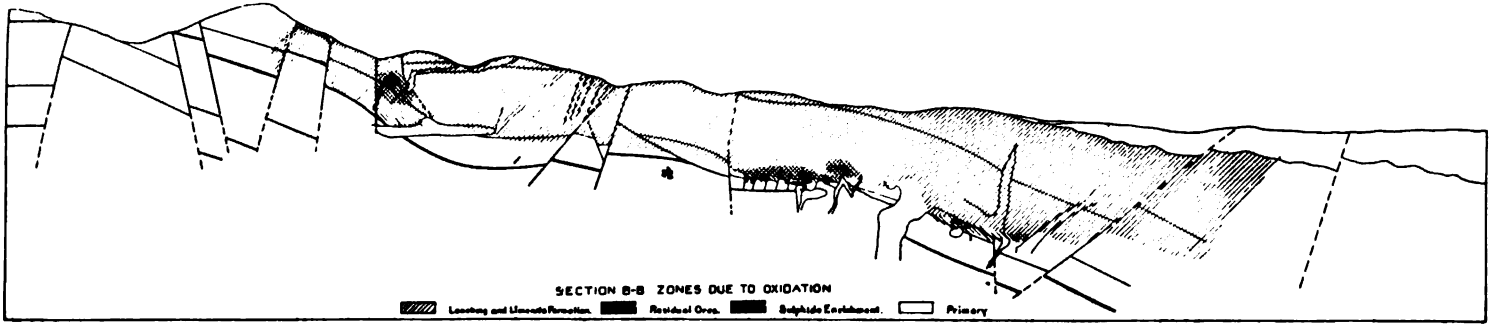

PLATES 6, 6*a*, 6*b*.—GEOLOGICAL SECTION ALONG LINE B–B OF MAP, PLATE 4.

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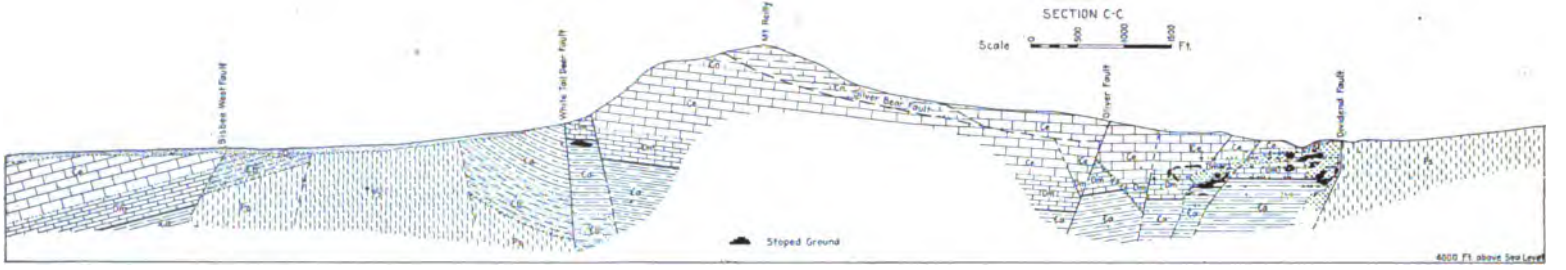

PLATE 7.

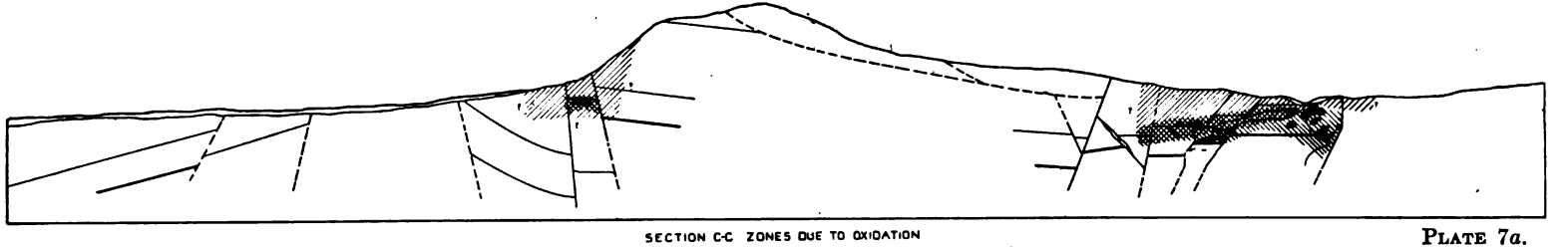

PLATE 7a.

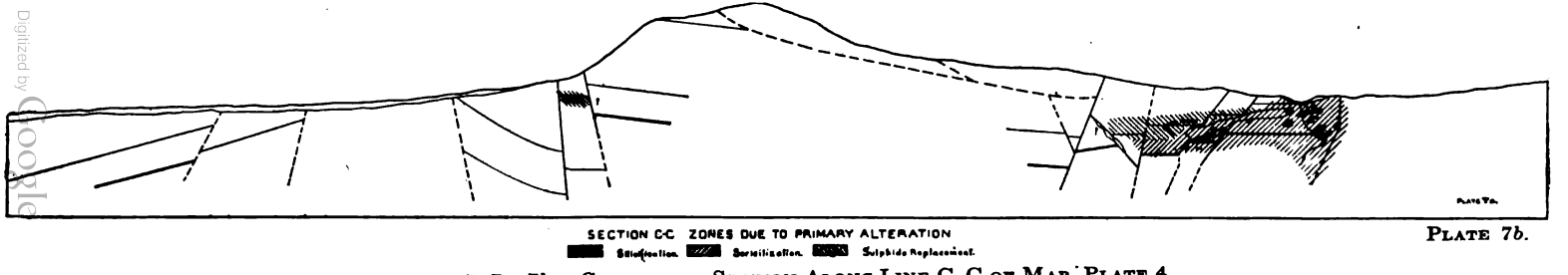

PLATE 7b.

PLATES 7, 7a, 7b.—GEOLOGICAL SECTION ALONG LINE C–C OF MAP, PLATE 4.

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Cambrian granite mass, and the Sacramento Hill porphyry, is devoid of igneous rocks. No intrusion has been observed cutting the Cretaceous over this area.

Igneous Rocks

The pre-Cambrian Juniper Flat granite mass has the form of a batholith. The contacts with the schist are clean cut. It probably forced its way up very slowly along some ancient line of weakness. There is no evidence of its having engulfed the schist, as no horses of schist are found inside the area exposed. From the nature of the schist, however, it is difficult to tell how it came in.

In all the pre-Cretaceous blocks mentioned above, extensive intrusion has taken place of granite porphyry, followed in some instances by extrusive rhyolites. These intrusions and extrusions came in for the most part along faults already in existence, but spread out in the less resistant sediments as sills and irregular masses. Although coming in at more or less the same time, some dikes clearly cut the others. The general habit of the porphyry was to follow some fault for a certain distance, and then cut across to another fault. The whole process suggests a very rapid intrusion.

In the Bisbee Extension Block, at the northwestern end of Escabrosa Ridge, there is an occurrence of extrusive rhyolite overlying an erosion surface of Naco limestone and an intrusive porphry sill.

Cutting through the Sacramento Hill porphyry, north of Naco Road, are several small dikes of andesite. This same andesite occurs cutting Escabrosa limestones near the Cole shaft, and has been observed underground near the Shattuck, Wolverine and Wade-Hampton mines. At the Shattuck mine, it is undoubtedly later than the porphyry. This andesite in all observed cases has come in as very small dikes in already opened fissures. In only one case has an outcrop been observed in higher beds than the Martin.

In the post-Cretaceous blocks, no intrusions have been seen except in the Naco Block. Here, at the north end of the block, entering along the big Abrigo Fault, an extrusive rhyolite mass has come in. This mass is similar in all respects to the mass mentioned above in the Bisbee Extension Block. Except for this one case, the large faults of post-Cretaceous age have not been accompanied by igneous intrusion.

The most important, economically, of all these igneous rocks is the granite porphyry, as it alone seems to play an important rôle in the processes of ore deposition. By far the greater amount of porphyry in the district cuts through with clean contacts. The one exception to this rule is the mass of Sacramento Hill. The mode of intrusion of this particular mass is somewhat different from all the others, due to the accompaniment of mineralizing emanations. This will be more fully treated later.

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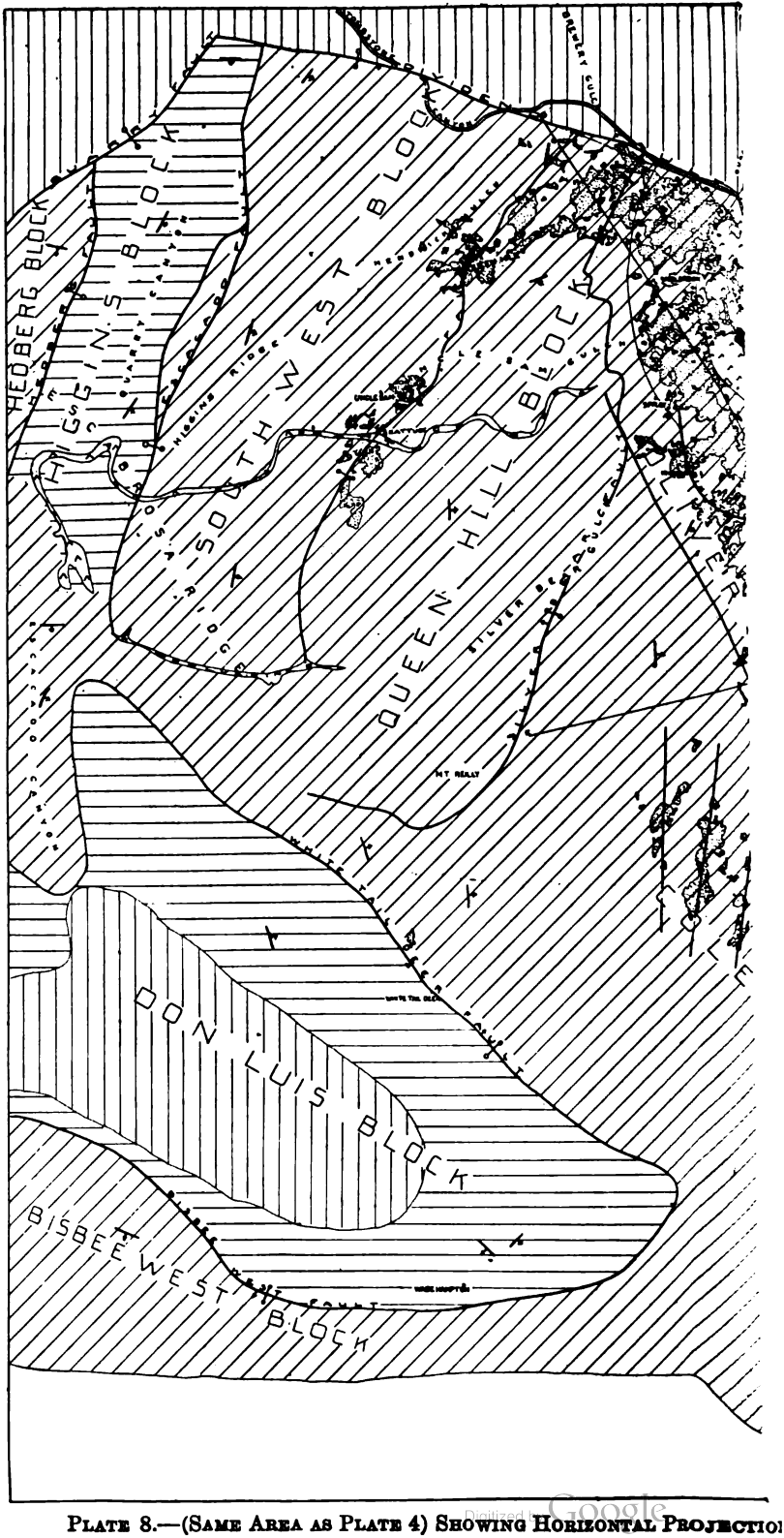

PLATE 8.—(SAME AREA AS PLATE 4) SHOWING HORIZONTAL PROJECTIO

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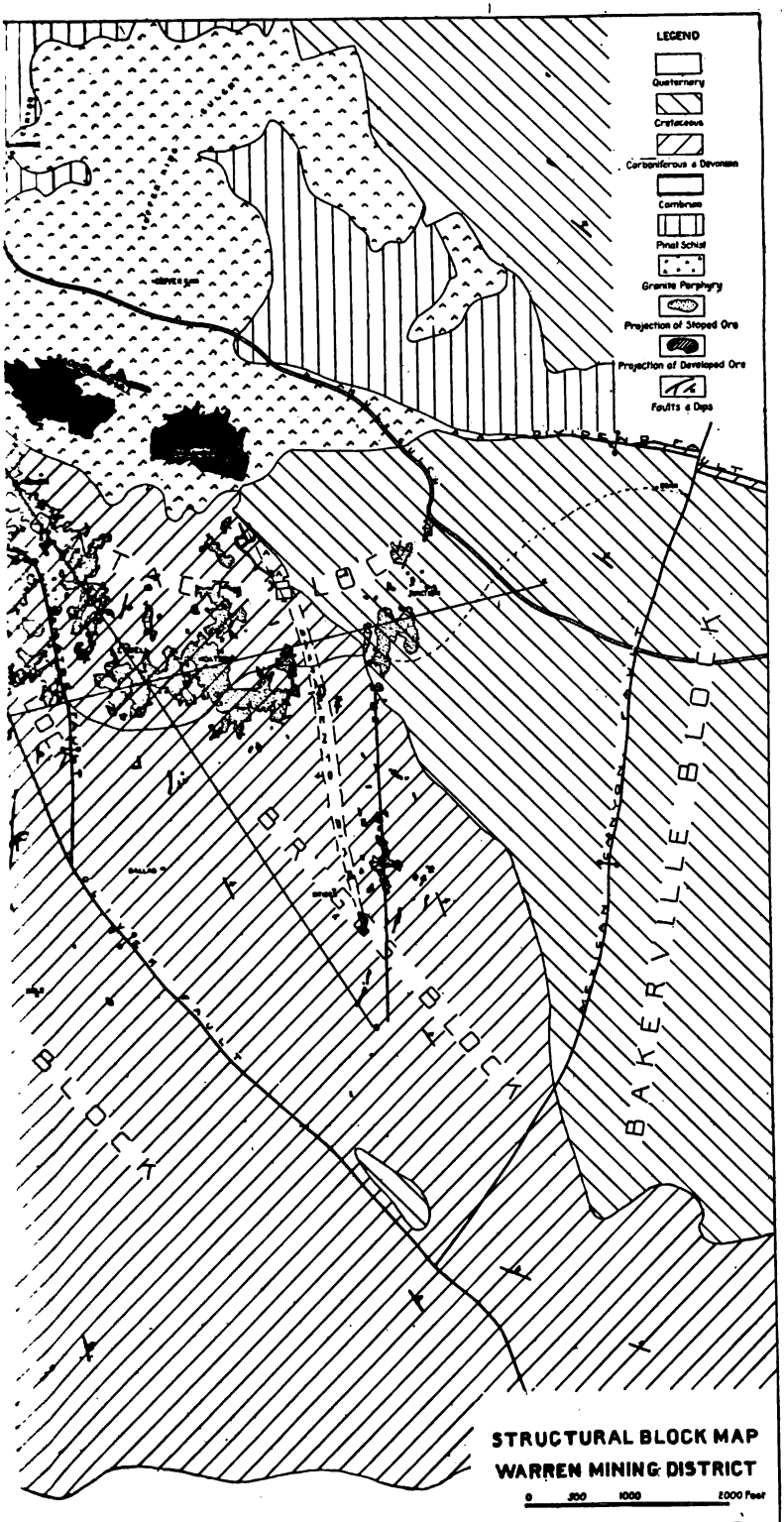

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OF STOPES AND SECTION LINES D–D AND E–E OF PLATES 9 AND 10.

Copper Queen Block

Fault Blocks

From an economic viewpoint, the Copper Queen and Don Luis Blocks are at the present time the only important ones in the range, and these two blocks, together with the mode of intrusion of the Sacramento Hill porphyry mass, will be more fully explained below.

The Copper Queen Block, whose details are best shown by Plate 8, lies in the heart of the range. It is a relatively down-thrown block, being bounded on the north by the Dividend Fault, a N. 75° W. fault dipping to the south from 60° to vertical. This fault has an estimated throw at the town of Bisbee of at least 1,500 ft. throwing Pinal Schist with a few outliers of Bolsa Quartzite on the north against Paleozoic sediments to the south. This fault apparently forms a block to the west with the Quarry Fault, a N.E.-S.W. fault which forms the western boundary of the area. To the south, the block is bounded by the White-Tail Deer Fault system parallel to the Dividend, throwing Bolsa quartzite and schist to the south against Abrigo and Martin limestones, to the north. To the east, the block is covered by Glance conglomerate and Morita sandstone, which are in turn covered by the Quaternary gravels of Sulphur Springs Valley. In the southeast, the block ends abruptly at the Gold Hill Overthrust Fault.

Starting somewhere near the Spray shaft, the block is cut by a N.W.-S.E. fault of considerable throw, the Oliver Fault, which dips about 60° S.W. This fault increases in magnitude toward the southeast and where it is finally covered by Quarternary wash, about halfway between Don Luis and the Country Club, it throws Abrigo limestone to the northeast against Naco limestone to the southwest, a throw of at least 1,500 ft. This fault is encountered in the Spray, Oliver and Cole workings. At the Spray the throw is about 200 ft.; at the Oliver it throws Abrigo limestone against Escabrosa with a throw of at least 500 ft. It finds topographic expression in the big draw between the Dallas and Cole shafts.

The Copper Queen Block, as seen by Plate 8, may be roughly divided into eight structural blocks. Starting from the west, we have first the Hedberg Block, relatively down-thrown, and with dips generally to the south. This is followed by the Higgins Block, up-thrown with dips to the east and southeast. Following this is the Southwest block, bounded on the east by the Czar fault system, and on the west by the Escacado fault system. This block is down-thrown relative to the Higgins Block but up-thrown in relation to the Queen Hill Block to the east. The dips here are nearly due east. The Queen Hill Block is a funnel-shaped block of Naco and Escabrosa limestone. It is well shown in section A-A, Plate 5. It is bounded on the west by the Czar fault system, and on the east by the Silver Bear Fault. The dips on the surface are irregular

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but, in general, to the east. Underground, below where the two faults wedge out, the dips are consistently east. Surrounding Sacramento Hill, to the south and southwest is a dropped block, which we may call the Contact Block. It is bounded on the north and northeast by the Dividend Fault and Sacramento Hill, and on the south and southwest by the Dixie-Howell-Senator Faults. To the southwest of this Contact Block and up to the Oliver Fault is the Oliver Block of relatively up-thrown beds. Southwest of the Oliver Fault is the Cole Block and northeast of the same fault is the Briggs Block which extends east to the post-Cretaceous Mexican Canyon Fault. The dips of all these blocks more or less surrounding Sacramento Hill are nearly due east. The Oliver Fault has tended, however, to swing the up-thrown side, especially where the throw becomes greatest, around to the northeast as is shown in the detailed geological map of the district, Plate 4.

To the east of all these blocks is an eighth block, the Bakerville Block, almost entirely covered by Glance Conglomerate. This block is bounded on the west by the Mexican Canyon Fault, striking about N. 15° to 30° east and dipping to the northwest. This fault is a late one and extends beyond the Dividend Fault, causing the prominent dislocations of the Mural Limestone seen north of Lowell and Warren.

As is shown in section B-B, Plate 6, the apparent dip of the beds in the whole block is very much less than the actual. This is due to minor faulting or slipping which has had the effect of dropping the beds very consistently to the west and raising them to the east. This is one of the typical structural features of the camp and seems to be caused by the extreme resistance of the beds to folding.

There are two ages of faulting, one pre-Cretaceous and the other post-Cretaceous. As has previously been shown, the porphyry in the district is pre-Cretaceous. The age of the faulting has therefore been determined in large part relative to the age of intrusion. The only large fault which is definitely post-Cretaceous is the Mexican Canyon Fault. As the fault apparently blocks with the Oliver Fault as shown in Plate 8, the Oliver is probably also post-Cretaceous. This is borne out by the character of the exposures underground. There, practically no mineralization accompanies it, and in Oliver ground it apparently cuts off an orebody, which as will be seen later, would make it post-Cretaceous. Both of these faults have the same effect, that is to raise the sediments to the east, and to drop them to the west. The probability is that the Junction Fault, shown in section B-B, Plate 6, is also post-Cretaceous, as it is reported to cut off certain parts of the Junction orebodies, and it is also possible that a large part of the north to south slipping in Lowell and Gardner ground is also post-Cretaceous. Here the throw of each individual fault is so slight, and the ground so highly metamorphosed, that it is not susceptible of proof one way or another.

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All other faults, so far as observed, have had their major movements in pre-Cretaceous time and before the introduction of the porphyry. Some faults have had movements in both ages. The Dividend shows some movement at the present time, and enough movement occurred in post-Cretaceous time to throw out the Glance Conglomerate. Some post-intrusion slipping has also undoubtedly taken place along the Czar Fault.

Intrusions

As has previously been shown, the block has been extensively intruded by granite porphyry. The largest and most important mass is that of Sacramento Hill, the other occurrences being more or less radial offshoots of dikes and sills from this central mass. It has also been shown that underground exposures below the Escabrosa horizon are much greater than surface exposures.

The path of the Sacramento Hill porphyry was the previously opened-up Dividend Fault. Underground development has shown that the porphyry acted as a plug, and that none of the later movements on the fault have taken place along the original line through the mass. The offshoots of the mass have generally followed major lines of weakness such as the Czar, and Silver Bear Faults. Some, however, such as the Shattuck dike, have cut across the country irrespective of fault lines.

The Sacramento Hill, and the connecting Lowell, mass differ from all the other porphyry occurrences in the range, in that they were accompanied by mineralizing emanations, which have completely altered them. These emanations, besides altering the porphyry itself, changed and broke up the surrounding sediments. In consequence of this, as the mass worked its way up, it dragged in pieces of schist, quartzite and limestone along its edges. The final result was a central core of metamorphosed porphyry, with a periphery of contact breccia, made up of highly silicified and altered fragments, sometimes rounded and sometimes angular, of schist, porphyry, quartzite and metamorphic limestones. This contact-breccia zone has a variable width, with a maximum of about 1,000 ft. It has fairly sharp contacts with the surrounding limestone, but tends to grade off into brecciated porphyry, and finally blocky unbroken porphyry, toward the center of the intrusive mass.

Don Luis Block

The Don Luis Block is an elliptical-shaped block up-thrown relative to the Copper Queen. It is bounded on the north and east by the White-Tail Deer fault system, throwing Abrigo and Martin limestone against schist, Bolsa, quartzite, and some Abrigo limestone. To the south it is bounded by the Bisbee West Fault, which is covered by the Quaternary

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wash toward the east and toward the west it is bounded by the Quarry Fault. The dips are variable, but for the most part are to the northeast and east.

Intrusions and Orebodies

The block is intruded both by granite porphyry and by andesite, the latter, however, not outcropping. There are two ore occurrences in the block, one at the northern boundary, and rightfully in the Copper Queen Block, and one at the southeastern end. The first of these, the White-Tail Deer orebody, occurs as a replacement of Abrigo limestone, about 50 ft. from the base of the Martin. It is not associated with any known intrusion, but is close to the White-Tail Deer fault system. The second occurrence is that of the Wade-Hampton. Here the ore occurs as lead and copper ore in a fault separating Abrigo limestone from Bolsa quartzite. The ore here is apparently associated with a dike of andesite and occurs in the crushed Abrigo limestone on the fault.

VI. GEOLOGIC HISTORY

The earliest geologic age is recorded in the district by the pre-Cambrian Pinal Schist. As has been previously stated, this was probably derived from arenaceous sediments.

Due to the extreme metamorphism, the geologic history subsequent to the deposition of these early sediments, and up to the deposition of the Cambrian sediments, is obscure. The sediments were probably subjected to extreme folding, intruded by diabase, and deeply buried. After the metamorphism was complete, the schist was intruded by a large mass of granite. From then on to Cambrian time, erosion worked on the whole complex, wearing it down to an even peneplain.

At the beginning of Cambrian time, a gentle subsidence took place, resulting in shore-line deposits of quartzites, the subsidence becoming more rapid toward the end of the period, in which time were deposited finer sands and some shales. The land surface toward the end of the Cambrian period became more distant, as evidenced by the deposition of the Abrigo shales, and shaly limestones.

At the end of Cambrian time, a gentle rise took place over an extensive area, bringing the flat sea bottom close to the surface, with land actually emerging probably toward the south. During the Ordovician and Silurian ages, this lowland surface of Cambrian beds was subjected to slow erosion simultaneously with the deposition in the district of an 8-ft. bed of quartzite. A slow subsidence then took place to the north of the district, allowing the thicker deposits of quartzite in the Yellowstone Range of 40 ft., and at Globe of 500 to 700 ft.

At the end of the Silurian, a sudden subsidence took place, resulting

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in deeper-sea conditions and thus the Devonian age is represented by fairly pure dolomitic limestones.

The sea evidently continued to deepen all through Mississippian time with the deposition of the pure Escabrosa limestone. During Pennsylvanian time, subsidence did not keep pace with deposition and the sediments became increasingly more shaly, and cherty showing a nearer approach of land surface and shallower seas. At the end of the Pennsylvanian, and probably during the Permian age, a violent uplift took place, resulting in the doming of the sediments around the pre-Cambrian granite mass. The sediments were shattered by extensive faults, the major lines of weakness being northwest and southeast. The granite mass itself resisted this shattering to a great degree except around its edges. Immediately following the faulting, or partly accompanying it, granite porphyry was intruded, following generally the major lines of weakness, but also shooting sills and dikes into the less resistant sediments. Accompanying one of these porphyry intrusions, that of Sacramento Hill, were mineralizing emanations, which metamorphosed the surrounding sediments and the porphyry itself, and were the source of our present orebodies.

During Jurassic and Triassic time, the whole area was subjected to erosion, during which time most of the upper Carboniferous beds were stripped off, and at the crest of the dome, around the granite and schist of Juniper Flat, the old pre-Cambrian complex was laid bare. During this erosional period, the orebodies were subjected to oxidation, which would place the age of primary ore formation definitely at or near the age of the uplift.

At the end of this period, a remarkably sudden subsidence took place, before any of the land except the top of the dome had been leveled off, and the rough topography surrounding was filled up with the extremely coarse Glance Conglomerate. The subsidence was so sudden that the fragments making up this conglomerate were hardly worked over at all, it being made up of angular boulders, some as large as a yard in diameter, of schist, quartzite and limestone. The roughness of this old topography is well shown in the basin covered by conglomerate, in which the town of Warren is situated.

After the whole country was leveled off by the Glance Conglomerate, the subsidence became less rapid, and shore-line deposits of sandstones and sandy shales were laid down, followed by deeper-sea conditions, during which the pure Mural Limestone was deposited, showing abundant Cretaceous (Comanche) fossils. Shore-line deposits followed this again to the end of Cretaceous time.

During Tertiary time, a second uplift took place, along almost the same axis as the previous one. This also took the form of a dome around the old granite mass, and the Escabrosa Ridge porphyry intrusion. This uplift, however, was not as violent. It was accompanied by extensive

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block faulting, with little disturbances of the beds within the blocks. The major faults took the form of overthrusts, the largest being the Tombstone and Gold Hill faults. Either accompanying the uplift or subsequent to it, some intrusion of monzonite porphyry took place, at the southern end of the range. Probably some of the extrusive rhyolite also belongs to this age of intrusion. Finally the whole range was subjected to a tilt to the northeast of about 15°.

From then on to the present time, erosion has been steadily at work, and has again laid bare the old pre-Cretaceous sediments along the crest of the dome, with Cretaceous beds exposed to the northeast. To the southwest, the Cretaceous has been all eroded, due to the final tilting. The only evidence left of the second doming is found at the southern end of the range, as shown by Plate 1. During this last erosion period the orebodies have been further subjected to oxidizing and enriching processes, so that we owe their condition as we find them today, to two widely separated periods, in each of which the original primary ores were worked over by surface waters.

VII. METAMORPHISM AND MINERALIZATION

General Metamorphism

In a region that has been subjected to as many geological changes as the Warren district, it was to be expected that the rocks would show many variations in texture and mineralogy from the unaltered types. The effects of regional metamorphism as well as dynamic are represented in the Pinal Schist as has been shown in describing the lithological characters of this formation.

The Cambrian formations have also been changed throughout their occurrence in the district by a process related to regional metamorphism in the wide scope of its action. The induration of the pure Bolsa quartzite has gone on until it is of glassy texture, but in the clayey, magnesian Abrigo limestone, besides the segregation of chert, the impurities have everywhere combined into such minerals as epidote, zoisite, chlorite, serpentine, and possibly albite. No such action is at all observed in the conformably overlying Martin limestone, which strongly suggests that the changes took place in the long period separating the middle Cambrian from the Devonian. But since field evidence excludes any great disturbance during the unrepresented period, it is probable that the general metamorphism in the Abrigo was caused by the deep burial under more than 4,000 ft. of overlying limestones after the impurities in the beds had had a chance to segregate and rearrange themselves due to the slow action of cold surface solutions during Ordovician-Silurian times. The necessary temperature conditions were probably obtained by the rise in iso-

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therms due to deep burial and to the post-Carboniferous intrusive period when a general rise in temperature could effect changes in the more susceptible impure sediments.

Whether this tentative explanation is correct or not, the importance of recognizing the general distribution in the Abrigo of minerals which may be mistaken for indications of contact metamorphism cannot be overemphasized. This formation is of considerable economic importance and the underground search for ores cannot be aided by the same indications as in the higher limestones.

Contact and Hydrothermal Metamorphism

That the granite-porphyry intrusion was the supreme factor in the ore deposition of the district cannot be doubted in view of the intimate association of the ores with the porphyry and the intense changes that have taken place in the rocks around the main intrusions.

It has been customary to separate the metamorphism of the rocks around a contact into contact metamorphism, caused by the igneous rock itself, and hydrothermal metamorphism, caused by the heated solutions emanating from the cooling magma. This distinction will not be made in the present article, because there has been nothing found by the authors to indicate that there is any genetic difference in the causes for changes in the rocks, outside of differences in intensity or quantity. The differences in *effects* are not to be confused with the causes, as the first are subject to the many variations of the rocks encountered, their physical and chemical differences, and their previous alteration. The metamorphosing agencies vary also with the distance from their source in the effect they may have on the same rock, but all changes observed so far have usually been graded and not susceptible to sharp definition.

In general, it may be stated that the two main centers of porphyritic intrusion are also the centers of metamorphism. Around Sacramento Hill the effects of metamorphism can be seen on the surface, but around the Lowell center the alteration did not extend through the covering of upper Carboniferous limestone. The effects around Sacramento Hill are more pronounced on the side of the calcareous sediments, which have been so much more altered than the schist of the north side. The porphyry on the south, as well as the covering remnants of contact breccia, are highly silicified and stand out as the crest of Sacramento Hill. The topographical depression around this hill is due to less silicified porphyry breaking through the contact breccia where this last turns from a more or less vertical to a horizontal body. Outside of this moat-like depression is a ring formed by the highly silicified contact breccia which grades into silicified metamorphic limestone, and this into limestone with a decreasing amount of metamorphic minerals until a belt of marbleized rock is

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reached, from 200 to 1,000 ft. away from the porphyry. The zone of recrystallization also fades outward irregularly to the unaltered sediments.

On the north, or schist side of the contact, there is some silicification of the older rock, accompanied by considerable pyritization, but this very soon fades out into the unaltered schist.

Underground, in tracing out the zones of rock alteration due to the intrusion and its accompanying solutions, it has been found that they roughly correspond to the ones just mentioned as appearing on surface, with the following general modifications: The zones extend farther than the general surface contours along extensions of the porphyry mass, especially the zone in Lowell ground, and along lines of fracturing which may or may not also be lines of faulting. The extent of the zones is remarkably influenced by the formation they are traced in, the changes going farthest at the top of the Devonian and the top of the Cambrian. Local variations are found to break the arrangement of the zones of alteration where there are minor centers of strong action, such as close to the Shattuck dike in the Uncle Sam and Shattuck ground, in the Southwest mine, and in several other places.

The Effect of Contact and Hydrothermal Metamorphism on the Formations

Before taking up the effects of contact and hydrothermal alteration on the rocks of the district it is well to state that hydrous metamorphism or oxidation has obscured or complicated in a great measure the results of contact metamorphism, and in many cases made the exact separation of the process involved too difficult and useless to be undertaken in the course of economic work.

The changes due to circulation of meteoric waters will be taken up later. The mine workings have now gone down far enough in all zones to disclose the fact that there is a depth below which the rocks as well as the ores are different from those closer to the surface. Here there are none of the minerals evidently due to the process of oxidation, and mineralogical associations continue the same indefinitely downward, without the vertical variations of the upper portions. This is the criterion that has been used in separating the effects of primary and secondary processes in the rocks and in the ores. Though not found in this order in the mines, primary effects will be considered first.

As a whole the impression given by the rock alterations in the Warren district is one of abundance and persistence rather than great intensity. There are no high temperature minerals developed in any great amount, or if they were developed at some stage, they have now been replaced. Garnet, diopside, wollastonite, scapolite, and vesuvianite have been observed, but in very small amount, and never forming an important part of a formation. Tremolite, actinolite, and edenite are far more common.

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The distinguishing minerals of alteration zones are quartz, sericite, chlorites, especially penninite, and the oxides of iron, magnetite and specularite, as well as the metallic sulphides of iron, copper, zinc and lead.

Metamorphism in Contact Breccia

Around any of the granite porphyry that has been accompanied by mineralizing emanations, such as that of Sacramento Hill, the contact breccia is marked generally by an extreme amount of silica which replaces all other gangue and rock-forming minerals. Pyrite and sericite are next in abundance, with chloritic minerals in variable amounts and calcite practically unknown. Schist and quartzite fragments have naturally remained the least altered in this mass and are consequently recognized most easily. In this contact breccia, or at the edges, there are bodies of intergrown magnetite, hematite and pyrite, with associations of the best-formed garnet, wollastonite, and other contact minerals. Silicification usually decreases toward the outer edge of the breccia, sericite and chlorites increasing. In many thin sections studied, the prevailing impression gathered is that in most of the breccia silicification is the last process involved, following sericitization and chloritization.

There are, however, distinctly later porphyry dikes cutting the breccia, as has already been stated, which have also been accompanied by metamorphosing emanations, the result being a complication and confusion of the processes involved. Certain highly chloritized portions of the breccia, where penninite replaces quartz in a noticeable way, are the result of these conditions, which are economically important, since copper minerals are concentrated in very pure form by the superimposed alteration. There are also other observed forms of this repeated alteration depending on the variations of either the first or the last. But it is generally true that the second process never reaches the point of adding silica or even adding sericite to the previously changed rocks.

Metamorphism in the Contact Limestones

The transition from breccia to limestone that can be recognized is apt to be very sudden in the Carboniferous horizons, and graded in those below. This is because the alteration of the lower limestones is more pronounced. The impure, and at the same time easily crushed, Devonian and Cambrian formations become a mass of sericite, penninite, calcite, and quartz, with some tremolite, garnet, diopside, wollastonite, and epidote, while the Escabrosa limestone usually has just a narrow fringe or a few bands of metamorphic minerals and the mass of the rock is simply recrystallized to marble. As extremes of alteration in this contact zone there are places where the limestone, especially the Devonian, is converted into a highly sericitic or chloritic mass, or into almost pure quartz.

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There is some evidence in this zone also that silicification is the last process affecting the rocks under ordinary conditions.

Farther from the contacts or the centers of mineralization the changes visible in the limestone are a decrease in silica and sericite in all the rock, while these minerals may persist in breaks and joints. Recrystallization of calcite, but not of the impurities, is very common.

In this contact zone the accumulation of sulphides is marked by the increase in either sericite, silica, or chloritic minerals around the sulphides.

Metamorphism Outside of Contact Zones

Sometimes rocks with considerable alteration are encountered far from any known porphyry, in which case it may be that solutions have traveled along fractures, or that they may have come with undiscovered porphyry. Usually, however, the copper ores have been found replacing the limestone very much farther away than any general alteration of the formations, in which case the transition from fairly unaltered rock to sulphides is sudden, but bears some relation to the local structure.

Metamorphism of the Porphyry

The primary solutions emanating from the granite porphyry affect that porphyry in about the same way as they do the adjacent rocks. If the nearby rocks, whatever they may be, are silicified, the porphyry will be also. If sericitized or chloritized, the same processes will have affected the igneous rock. The main difference comes in the relative amount of sulphides which have formed during these changes. Of course, the same minerals will not form in limestone as in igneous rock, but an increase or decrease in silica or magnesia or iron will be parallel in both cases.

The main mass of Sacramento Hill is so highly altered that a thin section of this rock shows only a mat of sericite plates, with variable amounts of quartz, pyrite, and chlorite, without much being left to distinguish between quartz, feldspar and mica phenocrysts, and the base. Close to the breccia, silicification is common, accompanied by disseminated copper deposits.

In the dikes radiating from the hill, the changes in alteration of the porphyry are very clearly marked, as well as the relative ages of succession of the processes. Receding from the silicified portions, sericite increases and the outlines of phenocrysts, first of quartz, then of feldspar, begin to appear. Lastly appear the outlines of biotite, marked generally by successive laminæ of sericite and pyrite. Farther away penninite is found in increasing amount, the quartz phenocrysts being unattacked and some of the plagioclase feldspars scarcely altered. The chloritic minerals appear as remnants replaced by sericite.

When sericite has almost all disappeared, and there is no silicification, epidote, zoisite and calcite are sometimes found as alteration or meta-

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morphic minerals in the porphyry. At this zone there are also found portions of the rock converted into almost pure penninite and edenite, with some serpentine, as if the ferromagnesian minerals lacking in the more sericitic rock had been driven to accumulate and replace other portions. Only the presence of some unaltered quartz phenocrysts and very resistant apatite crystals helps to determine the original rock, although transitions have been exposed in some of the workings.

Sulphides in some amount invariably accompany the contacts of porphyry that is altered to this degree.

Farther away from the centers of action slight chloritization of the base, of some feldspars and of micas, and eventually of the micas alone, is the only alteration noticeable. This fades out in a few observed cases to fresh rock, or rock affected only by meteoric waters. The contacts of this less altered porphyry have either just a little pyrite and sphalerite, or no sulphides at all, and even have had no appreciable effect on the immediately adjacent limestone when seen in thin section.

The alteration of the more basic dikes which cut the contact breccia is also similar to the one induced in the breccia, except that the earlier changes are not observed in these dikes.

Summary

To summarize, it may be said that the weakest as well as the farthest reaching alteration in the porphyry is chloritization. Next comes epidotization, which is of minor importance. Sericite and finally quartz are the additional minerals formed with the increase in the strength as well, probably, as the temperature of the emanations. The minerals formed by the stronger solutions are formed by replacement of the earlier ones, in the ordinary case when metamorphism is advancing in intensity from the center. Superimposed alteration may be caused by later intrusives, or in certain instances by the decrease in intensity in the dying-out period of alteration.

Some unaltered dikes in Holbrook and Czar ground are found in zones where the rest of the rocks show intense sericitization, including some masses of porphyry. These dikes are associated also with some sulphide ores, the contacts being far more distinct than ordinarily. If it were not for the slight sericitization of these dikes close to the ore, while away from it they show no hydrothermal action, it would be possible to believe that they are the result of post-mineralization intrusions.

The Shattuck dike shows similar phenomena, as it is very fresh for long stretches, and then, close to the ore zones in Uncle Sam and Shattuck ground is almost as altered as any rock around Sacramento Hill, showing that the altering solutions did not accompany all the porphyry but found their way through special channels.

The following partial analyses of altered porphyries are typical:

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	I	II	III
Au, ounces	*Nil*	*Nil*	*Nil*
Ag, ounces	0.04	0.08	0.04
	Per Cent.	Per Cent.	Per Cent.
Cu	0.24	0.11	0.11
Pb	Tr.	Tr.	Tr.
SiO_2	55.9	76.6	73.3
Fe	9.8	5.6	5.7
CaO	1.8	3.1	0.5
MgO	0.5	3.2	2.8
Al_2O_3	16.6	7.4	10.1
S	8.8	1.0	0.4
Mn	0.6	0.6	0.6
Zn	Tr.	Tr.	Tr.
P	0.08	0.10	0.09
Totals	94.32	97.71	93.60

I. Highly sericitized porphyry with pyrite, from Sacramento dike, close to Sacramento Hill.

II. Chloritized, partly sericitized and epidotized porphyry, from Sacramento dike, about 1,000 ft. away from the hill.

III. Slightly chloritized porphyry, Dallas mine.

The general zones of primary alteration are shown for the sections through the camp in Plates 5*A*, 6*A*, and 7*A*. Silicification and sericitization have been explained already. Chloritization is not marked because it is too irregular, and though important in the porphyry, the sediments in that zone are not generally affected. The zone of sulphide replacement was introduced, where the limestones surrounding sulphide replacements were affected so slightly immediately away from the orebodies that there can be said to be practically no metamorphism.

VIII. THE ORES OF THE DISTRICT

General Occurrences

The general occurrence of the orebodies can be best appreciated by reference to the horizontal and vertical projections of the stopes given in Plates 8 to 10.

In the horizontal projection (Plate 8), there is a well-marked crescent-shaped zone around Sacramento Hill, with an extension around the Lowell mass of granite porphyry. Along the Dividend and Czar zones the effects of major structural lines are well marked by the abundance of ore taken out. The Shattuck dike-Czar zone intersection has its group of orebodies, and extensions and outliers from the general zone follow both the Sacramento dike and the series of breaks in Briggs and Cole ground to the southward.

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In the vertical projections (Plates 9 and 10) the inclination of the ore zone is shown; also in Plates 5 and 6 the stratigraphical position is seen to follow in general the dip of the limestones, at horizons varying from the top 300 ft. of the Cambrian to the lower 300 ft. of the Carboniferous.

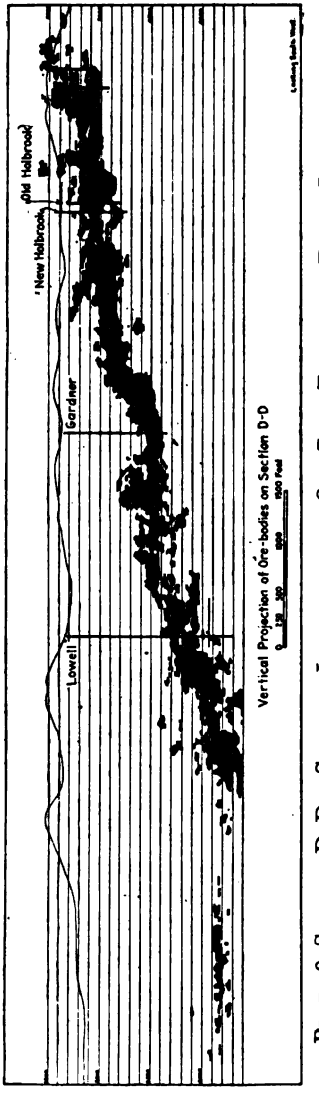

PLATE 9, SECTION D–D.—SHOWING INCLINATION OF ORE ZONE FOLLOWING DIP OF LIMESTONES.

CLASSIFICATION OF THE OREBODIES

As in the case of the metamorphism and alteration of the rocks, the authors have found no reason to separate the orebodies due to genetic differences. All the ores are believed to be due to one general period of

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primary mineralization which followed closely the intrusion of the main porphyry masses. Differences in the ores are ascribed mainly to the character of the formation that they replace and to the amount of general

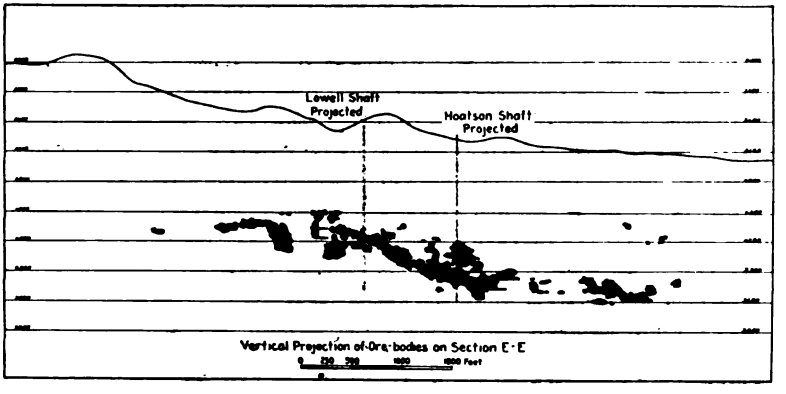

PLATE 10, SECTION E–E.—ORE ZONE BETWEEN LOWELL AND HOATSON SHAFTS FOLLOWING DIP OF LIMESTONE.

metamorphism in the surrounding rocks due to the distance from the centers of alteration. All the ores are also believed to be formed by metasomatic replacements of various rocks.

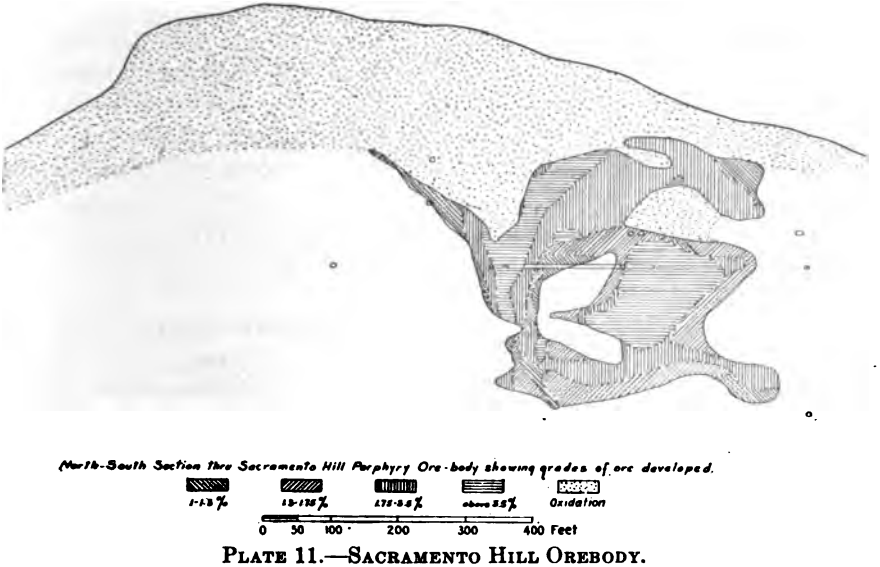

PLATE 11.—SACRAMENTO HILL OREBODY.

Four convenient subdivisions can be made, dependent on the formation replaced, as follows: ores in porphyry, in contact breccia, in contact metamorphic limestone, and in relatively unaltered limestone. In con-

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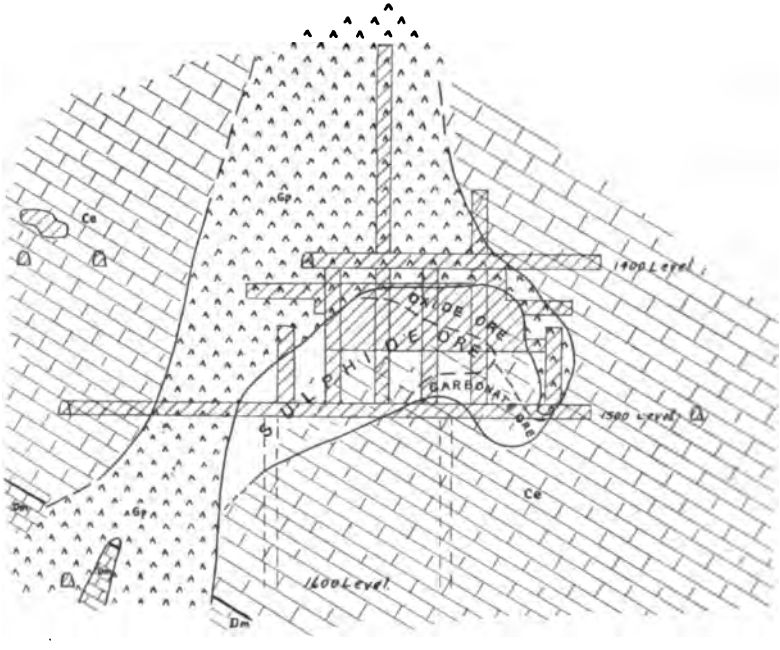
sidering these ores, the primary mineralization will be taken up first and the effects of meteoric water circulation left for later discussion.

Ores in Porphyry

To this type belongs the ore developed recently in Sacramento Hill. The ore occurs as a very irregular mass of chalcocite, pyrite and some bornite in brecciated, altered porphyry at the inner side of the contact breccia. The ore also penetrates the contact breccia, as shown in Plates 5 and 11, and connects with some typical bodies in this latter rock. The porphyry, in general, is sericitized, and somewhat silicified close to the



Section thru 15-12 Ore-body, Sacramento Mine.

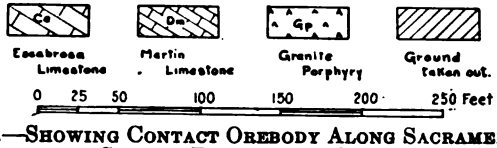


PLATE 12.—SHOWING CONTACT OREBODY ALONG SACRAMENTO DIKE AND SPECIAL FEATURES OF OXIDATION.

breccia, and in parts of the ore. The primary sulphide minerals are pyrite and bornite alone, this being the only type of ore in the camp in which chalcopyrite is not an important constituent. Partial analyses of this porphyry ore are given in Table 2.

Besides the ore in Sacramento Hill, ore occurs in porphyry in a

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great many places in the contact-metamorphic zone, where the sulphides replace limestone and in a minor, but very distinct way, the adjacent porphyry. A noteworthy example of this is the orebody known as the Dividend Slice No. 1, where most of the porphyry is entirely converted into ore.

Ores in Contact Breccia

This second type of ore is found from the Holbrook to the Lowell, Oliver and Sacramento mines. The bodies are extremely irregular in shape and size, and are not confined to any definite horizon, but are usually found in unenriched and unoxidized state. One of these bodies outcropped to the west of Sacramento Hill.

The breccia, as has been explained, is highly silicified, and the ores themselves may be in an extra siliceous portion, or in a chloritized zone due to one of the later dikes mentioned before. The value of the contact breccia as an ore zone has been realized only within the last few years, as the rock is perhaps the most difficult of all to prospect.

Ores in Contact-Metamorphic Zone

To this class and the next belong a great majority of the ores heretofore mined in the district. The contact-metamorphic zone, marked by many alteration products in the limestone, but mainly by sericitization of both intrusive and intruded rocks, forms a block around Sacramento Hill and the associated extensions of the porphyry. The ores occur as irregular bodies intimately connected with dikes and sills which are direct offshoots of the main porphyry masses. The replacements extend from some distance in the porphyry, through some of the unrecognizable altered limestones, into the distinct bedding planes of less metamorphosed sediments.

Faulting and crushing previous to the introduction of the porphyry and ore solutions played an important part in determining the location and limiting the size of individual orebodies. Very little post-ore deposition faulting has occurred, so little in fact, that it is a negligible factor for most of the district.

The contact-metamorphic zone occurs almost entirely in Martin and in the first 200 ft. of Escabrosa limestone. Some orebodies, however, are found in the top 100 ft. of Abrigo as shown clearly in Plates 5, 6 and 7. To this zone belongs the ore found along the Dividend Fault, which has played such an important part in structure and mineralization of the district. The rich ore mined early from the Czar and Holbrook was taken out along this fault, and rich ore is still being mined in the Czar from along it.

To this type of orebody belong also a great part of the ores immediately adjacent to the Sacramento dike, Lowell sill, and to dikes in the

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Junction and Oliver mines, though these bodies, being outside of the highly altered zone, generally grade into typical replacements of the unaltered limestones.

Ores in Relatively Unaltered Limestone

To this subdivision belong the orebodies in the Southwest, Higgins, Wolverine, Uncle Sam, Shattuck, the south end of the Spray, Irish Mag and Oliver mines, all of the ore in the Cole and Briggs, and the major portion of the ores in the Sacramento, Hoatson and Junction mines. A brief description of each of these occurrences is given below.

The Southwest orebodies, which include the original ore discovery of the camp, occur as replacements of Martin and Escabrosa limestones along the Czar Fault. The original Queen orebody outcropped at the surface, at the big glory hole at the base of Queen Hill. It occurred in the foot-wall side of the Czar Fault, replacing Martin and Escabrosa limestones. Following the Czar Fault to the southwest is a continuous chain of orebodies in the hanging-wall side of the fault, in the same stratigraphic horizon, until the present Southwest workings are reached. At this point the orebodies again cut across to the foot-wall side, but keep in the same horizon. It is these foot-wall orebodies that are being mined at present. The Czar Fault is here broken up and forms blocks with east-to-west faults. In this crushed zone occurs a large mass of brecciated iron-stained silica, which outcrops on the surface on both sides of Hendrick's Gulch. The orebodies are found at and near the contacts of this mass, as replacements of Martin and Escabrosa beds, and penetrate the breccia itself for short distances. The ore is both of copper and of lead.

The ore early mined along the Czar Fault (and now being largely remined), was closely associated with porphyry dikes and sills, which came up along the fault. The present orebodies, however, are not in contact with porphyry. The silica breccia mass, however, in depth is connected with porphyry.

Following the Silver Bear Fault, which forms the eastern boundary of the Queen Hill Block, another series of orebodies occur, also as replacements of Martin and Escabrosa limestone. These orebodies are directly related to porphyry dikes and sills.

The Higgins, Wolverine, Shattuck and Uncle Sam orebodies occur close to and in contact with the Shattuck dike. The Higgins and Wolverine are the farthest west, and therefore farthest removed from Sacramento Hill. Here the dike, as exposed underground, occurs as a dike of unaltered porphyry, and the orebodies are found on both sides replacing Abrigo limestone about 100 ft. below the top. The Uncle Sam and Shattuck orebodies occur as replacements of Martin and Escabrosa limestones, very intimately associated with a large sill of porphyry into which the Shattuck dike makes in depth. At the Shattuck, another mass

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of silica breccia occurs, more intimately associated with the porphyry than at the Southwest, with ore formed at the contacts, and penetrating into the mass. Here again the ore is both lead and copper as at the Southwest.

To the south and west of the Contact Block is a relatively up-thrown block, bounded by the Dixie-Howell-Senator Fault on the north and east, and by the Oliver Fault to the southwest, previously mentioned as the Oliver Block. In Irish Mag, Gardner, and Oliver ground, the block is much broken up by N.-S. fractures, and here there are orebodies in the foot-wall side of the Dixie-Howell-Senator Fault replacing Abrigo and Martin beds. In Spray, and in part of the Oliver ground where the block is less fractured, orebodies are found entirely as replacements of Abrigo limestone, from 100 to 300 ft. down from the top. In Spray ground no porphyry has yet been found anywhere near these orebodies. In Oliver ground, they are associated with small dikes.

In the Cole mine, orebodies occur in the Martin limestone and extend for a short distance into the top of the Abrigo, along N.-S. fractures. These orebodies occur in the hanging-wall side of the Oliver Fault. This fault, between the Cole and Oliver mines, has cut off one of these orebodies and is, therefore, post-ore deposition in age, one of the few breaks of its kind in the productive area of the camp. The Cole orebodies are not directly connected with porphyry.

The orebodies of the Briggs mine are similar to those of the Cole in structure, being related to a N.-S. fractured zone, not directly connected with porphyry. They are also replacements of Martin limestone.

The orebodies in the Hardscrabble claim of the Sacramento mine, and part of the Hoatson mine are directly associated with the Sacramento Dike. The orebodies exist as replacements of Escabrosa limestone, the horizon being about 200 ft. up from the base. The orebodies here are in direct contact with the dike on both sides, and make into it for short distances.

To summarize, the orebodies in the unaltered limestone occur for the most part at or near porphyry contacts, but they also occur away from porphyry in fractured country. In fractured country and where intrusions are numerous, the ore horizon tends to rise, and ore is found mostly occurring from the base of the Martin limestone to 300 ft. above the base of the Escabrosa. In unfractured, and relatively less intruded country, on the other hand, the horizon tends to drop and ore is found in the Abrigo limestone.

In the illustrations showing the zones of alteration (Plates 5*A*, 5*B*, 6*A*, 6*B*, 7*A*, and 7*B*) the orebodies in porphyry are seen to be between the sericitized and silicified portions of that formation. The ores in breccia are in a highly silicified zone. The ores in the contact-metamorphic limestones occur where the alteration is mainly sericitic.

Table 2 gives partial analyses of the orebodies along sections shown

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by the illustrations and the rocks that they are found in, as well as the character of the ore dependent on oxidation.

TABLE 2.—*Partial Analyses of Orebodies along Sections of Plate 4*

Section A-A

Rock Replaced	Location	Oxidation Features	Cu, Per Cent.	Fe, Per Cent.	S, Per Cent.	SiO_2, Per Cent.	Al_2O_3 Per Cent.	CaO, Per Cent.
Porphyry	East end Sac. Hill	Enriched	2.76	9.3	8.0	59.6	13.6	1.2
Porphyry	West end Sac. Hill	Enriched	1.8	9.4	9.8	60.5	11.4	0.9
Porphyry	West end Sac. Hill	Enriched	3.7	10.8	11.8	58.1	10.4	0.8
Porphyry	West end Sac. Hill	Enriched	6.6	13.6	15.8	50.1	7.8	0.7
Porphyry	West end Sac. Hill	Enriched	7.8	17.0	20.1	44.3	6.0	0.7
Porphyry	West end Sac. Hill	Enriched	12.2	16.9	21.3	43.7	3.8	0.4
Contact breccia	Breccia West of Sac. Hill.	Primary	6.4	19.1	22.4	40.7	5.3	0.7
Contact breccia	Breccia West of Sac. Hill.	Slightly enriched	6.1	14.7	16.4	46.1	9.1	0.5
Contact metamorphic limestone	Holbrook	Enriched	8.8	22.7	?	21.8	15.1	0.9
Contact metamorphic limestone	Holbrook	Enriched	4.4	24.4	25.3	13.0	15.1	1.0
Contact metamorphic limestone (slightly altered)	Holbrook	Slightly enriched	5.0	31.1	35.2	15.2	6.2	
Escabrosa limestone	Holbrook	Residual carbonate	6.3	22.3	0.2	10.1	5.3	14.8
Devonian limestone	Southwest	Residual carbonate	4.1	12.2	0.1	18.3	7.3	12.9

Section B-B

Escabrosa lime and porphyry	Uncle Sam Mine	Residual ore	14.7	26.3	...	20.6	7.5	2.6
Devonian lime	Uncle Sam Mine	Residual ore	6.5	22.5	...	19.8	9.5	7.0
Abrigo limestone	Gardner	Primary	6.1	28.5	29.3	20.7	4.3	1.7
Devonian limestone	Gardner	Primary	4.7	32.5	36.0	16.1	6.1	3.1
Escabrosa limestone	Gardner	Residual carbonate	8.6	25.9	...	20.0	5.1	3.0
Metamorphosed Devonian	Gardner	Primary	4.3	34.1	27.7	14.8	2.45	1.3
Contact breccia	Gardner	Primary	4.8	30.6	40.0	13.6	4.5	1.4
Metamorphic limestone.	Sacramento	Enriched ore	9.3	22.7	23.0	25.6	11.4	1.3

Section C-C

Abrigo limestone	White Tail Deer	Residual carbonate	5.25	13.8	...	45.9	5.2	2.6
Partly altered Devonian	Holbrook	Residual carbonate	8.1	13.2	Tr.	26.0	10.5	1.6
Partly altered Escabrosa	Holbrook	Residual carbonate	4.4	25.1	Tr.	19.4	9.5	1.5
Sericitised metamorphic	Holbrook	Enriched sulphide	5.5	21.1	?	14.9	23.0	0.4
Sericitised metamorphic lime and porphyry	Holbrook (Dividend).	Enriched sulphide	6.6	17.5	12.6	17.8	23.2	0.3

MINERALOGY OF THE ORES

As in the case of rock alteration, primary minerals in the ores have been changed greatly by the circulation of meteoric waters, and the criteria have already been given for determining the primary zone under

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the caption, "The Effect of Contact and Hydrothermal Metamorphism on the Formations."

The original sulphides of copper deposited are comparatively few in number, chalcopyrite and bornite being the important ones economically. Tennantite is the only other copper-bearing mineral so far found in the primary zone, if exception be made of one or two unknown minerals seen with the microscope. Pyrite is such an important constituent of the ores, that it must be considered along with the copper minerals, and magnetite, hematite, sphalerite and galena are common enough to be considered part of the ores, especially the last two, since they are the source of oxidized ores of importance.

Chalcocite is very abundant in many forms in the zone of enrichment but has not been seen in the undoubted primary zone, though this has been considerably explored. It is therefore highly probable that this mineral was not formed by the ascending solutions in this district, and conflicting microscopic evidence in the ores above has been considered insufficient to offset the field evidence of the primary zone.

Pyrite

Of the sulphides, pyrite is generally the first formed, and accompanies every one of the stages of alteration described under metamorphism. In the ores, very fine examples of the structures described by Graton and Murdoch[8] may be seen, grading from the perfect "porphyritic" relation between well-formed pyrite crystals and the other sulphides, to a mass of pyrite grains with little interstitial chalcopyrite or bornite.

As additional evidence for explaining the mode of formation of some of the sulphides the following observations will be given. Around any orebody, pyrite usually persists in the same general relations to the rock or gangue, that it has to the other sulphides in the ore, with the exception that in the ore the individual pyrite grains are broken up, penetrated or reabsorbed by the other sulphides in such forms as are illustrated in Plate 13, Fig. 2, and Plate 14. Thus, ore in porphyry has pyrite veinlets extending into the rock along joints and it will be found disseminated through the rock in irregular grains that usually started around a biotite. These same pyrite veinlets and disseminated grains will persist even when all the rock has been replaced by sulphides, as can be seen in Plate 13, Fig. 3. In limestone replacements, pyrite will extend out along bedding planes in layers that are continuous with pyrite bands in the copper ore. And pyrite crystals will be found to have the same "porphyritic" relation to slightly altered Abrigo limestone as they have to the rich ore in

[8] L. C. Graton and J. Murdoch: The Sulphide Ores of Copper, *Transactions*, vol. 45, pp. 26–93 (1913).

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FIGS. 1 AND 2.—"PORPHYRITIC" STRUCTURE OF ORES IN ABRIGO LIMESTONE (LEFT) AND IN BRECCIA.

FIG. 3.—REPLACEMENT OF ALL ROCK BY SULPHIDES WITH PYRITE VEINLETS PERSISTING.

PLATE 13.

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that formation, with the exception given above, that there is little breaking up of the pyrite (see Plate 13, Fig. 1).

FIG. 1.

FIG. 2.

PLATE 14.—ORIGINAL PYRITE GRAINS BROKEN AND REABSORBED BY OTHER PRIMARY SULPHIDES.

This information, coupled with microscopic observations concerning the age of crystallization of the pyrite has led to the belief that in the ore

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formation pyrite is the advance sulphide, formed partly by solutions that are not yet precipitating copper minerals. When these latter are formed, they take their place by replacement of the gangue and rock mostly, but also by slightly replacing the earlier formed pyrite. This may explain certain veinlet-like stringers of pyrite, which apparently cut the other sulphides, but which when examined in detail show that each grain of pyrite has been attacked by chalcopyrite or bornite.

Age Relationship of Pyrite.—In the porphyry ores, pyrite is found to have crystallized at about the same time as the sericitization occurred with some of the grains a little earlier. Pyrite deposition also extended into the period of silicification, but was undoubtedly also replaced by quartz. Bornite, the copper mineral, is found replacing the pyrite as already mentioned.

In the contact breccia, due to the many complicated interrelations found, the pyrite cannot be said to have formed at any definite stage. In general it holds true that pyrite is earlier than the rest of the sulphides, but distinct veins of it have been found through crushed pyrite, bornite and chalcopyrite ore. Magnetite, specularite, and sphalerite are of about the same age as the pyrite, and garnet, diopside and wollastonite slightly earlier.

In the contact-metamorphic limestone orebodies, pyrite and sericite again belong to the same period of formation, earlier than the copper sulphides, and slightly later than tremolite and other contact silicates. In this zone, pyrite veins are found cutting magnetite, specularite and earlier pyrite bodies.

In the orebodies that replace unaltered limestone the relations are the simplest, pyrite being closer in age to the copper, zinc, and lead sulphides than in any other place.

Chalcopyrite

This mineral is undoubtedly the most abundant primary copper sulphide in the district, and is prevalent in all the ores, except in the disseminated porphyry ore in Sacramento Hill. This ore is distinguished mineralogically from all others in the district by having as undoubted primary sulphides, pyrite and bornite alone.

Chalcopyrite is generally associated with the more siliceous ores, except in porphyry, its period of formation being slightly later than that of sericitization, but contemporaneous with silicification. In cases where there is late chloritization, some remarkably pure bodies of chalcopyrite are formed, as is also the case in certain vein-like bodies extending far above the general ore horizon, where galena is also abundant.

The intergrowths of chalcopyrite and bornite show them to be of contemporaneous precipitation, with a slight lagging of the bornite formation.

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Bornite

This mineral is universally found in the primary ores, though not as abundantly as chalcopyrite, as has been stated.

Although bornite occurs undoubtedly in the richer primary ores, there has been no reason found for its formation in preference to chalcopyrite, as it comes in rich and lean ores, and in those with a great amount of pyrite, also when there is scarcely any pyrite. Fig. 1, in Plate 18, illustrates a case where bornite and pyrite replace the cement in a calcareous Devonian sandstone, where the amount of copper is not enough to give 0.2 per cent.

Tennantite

This mineral is found usually in microscopic amounts only, in practically all the ores. Its age of crystallization is between that of pyrite and chalcopyrite. There may be some tetrahedrite, but as arsenic is more abundant than antimony in the specimens analysed, the mineral seen has been taken to be tennantite.

Sphalerite

The sulphide of zinc is found in all the primary ores in minor quantities, with zonal inclusions of chalcopyrite and galena in very fine grains (Plate 17, Fig. 1). In a few places such as between the Southwest and Czar mines, in the Gardner, and in Briggs ground, sphalerite is found in rather large bodies with pyrite and chalcopyrite. It generally replaces Martin limestone, and apparently its general relations are no different from those of the copper ores.

Galena

Like sphalerite, galena is associated with most of the primary ores, but is decidedly more abundant where the ores are in highly chloritized ground. The ground adjacent to the latest basic porphyry dikes, in contact breccia and in limestone, generally has more galena than the rest. An exception to this may have been the zones around the silica breccia masses of the Southwest and Shattuck mines, but unfortunately oxidation has obliterated thoroughly all primary mineralization. Lead carbonates and sulphates are here found in abundance.

Galena is found more abundantly also in the top zone of the ores, as a residual sulphide, so the general impression given by the occurrence of the lead sulphide is that it is more abundant in the last stages of mineralization and also that it was carried farther than the copper.

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Other Primary Metallic Minerals

Magnetite and specularite occur sometimes in large bodies that show a very perfect intergrowth of these oxides, as well as with pyrite, as has been mentioned. The occurrence of these bodies is limited to the contact-metamorphic zone, or is immediately on the contact of porphyry dikes. A few grains of pyrrhotite have been seen here, indicating the high temperature of their formation.

Specularite is also found associated with the orebodies in contact breccia that come with the basic, chloritizing porphyry dikes. The oxide is here a valuable indication of the proximity of ore, in a zone where all other features are baffling in their irregularity.

Silver and gold minerals have not been seen in the primary zone, with the possible exception of a few grains of polybasite found in galena. Gold and silver values are considerable in all the copper ores, and more so probably in the sulphides than in the oxides, but their amount could easily escape detection even under the microscope. Tennantite, as well as one or two brownish unknown minerals in the sulphides may be the precious-metal carriers.

IX. OXIDATION AND ENRICHMENT

General Features

The most striking feature of oxidation of the orebodies in the Warren district is the varying elevations at which the column, from oxidized to primary ores, repeats itself in going from the western to the eastern side of the mining area. At the extreme western end, the Higgins and Wolverine mines have primary ores in Abrigo limestone far above the level of the adjacent creek bottoms, and consequently far above the present ground-water level. In the Czar mine oxidation and enrichment go down to just under the 300 level, which is about 5,000 ft. above sea level. In the Holbrook division some primary ore outcropped at the west of Sacramento Hill, while some distance south, enrichment goes down below the 500 level. In the Gardner, primary ores are not found till the 9th level is reached, and in the the Lowell mine they are encountered, in some instances at the 10th level, but generally below the 13th, although the permanent water level was encountered here around the 10th level, or at an elevation of 4,325 ft.

In the Hoatson mine, oxidation features were found down to an elevation of about 3,800 ft. above sea level, and even deeper in part of the Sacramento and Junction mines, although the permanent water level was far above this before the mines were drained by opening up and pumping from the deeper levels of the Junction.

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The oxidation features shown by the illustrations of the sections through the camp (Plates 5*b*, 6*b*, and 7*b*), illustrate this general inclination to the southeast. It will be noticed that the oxidation follows in general the dip of the Paleozoic sediments, the average horizon reached being somewhere in the Devonian. There is a marked lagging of the oxidation at the eastern end, however, as here it does not reach the top Devonian. In this, the oxidation may be seen to correspond more closely to the pre-Cretaceous erosion surface.

It will be seen also that there are some sudden changes in the level of oxidation, as best shown in section A-A of Plate 5, and also that there are great variations in the extent of the zone of secondary sulphides. All these features have fairly definite relations to the geologic history and structure, to the nature of the formations, and to the primary alterations in them. The object of this chapter will be to explain what is known of those relations.

It will first be necessary to present the criteria for the subdivisions of the zone in which meteoric waters have circulated, in many directions, but generally downward. It has been stated before that the primary zone was marked by steady conditions and constant mineralogical associations. The deepest effects of the surface-water circulation are immediately and clearly marked in this district by veinlets, filled cavities and coatings of other minerals.

Just above the primary zone there is an extremely variable thickness where the prevailing secondary mineral is siderite with a little gypsum and an extremely small amount of chalcocite. Above this is usually a zone where halloysite and kaolin are most prominent among the secondary gangue minerals, and chalcocite is abundant as a replacement of primary sulphides. Next higher, limonite, and the staining due to iron oxides—gossan formation, in other words,—is the prevailing feature. Within this zone there are found the local bodies of residual ores, which have for some reason resisted the general leaching effect of strongly oxidizing waters. One or all the zones above the primary may be lacking at any point for reasons which will be explained later.

Relations of Oxidation to Structure and Geological History

Notman[9] has already pointed out the relations between the attitude of the oxidation and present and past water levels. That the present water level does not account for the oxidation is proved by the occurrence of unoxidized ores above that level in formations that are in other places completely oxidized also above that water level. This is illustrated by the primary ores in Abrigo at the Higgins mine, and the completely oxidized Abrigo ores in the Wolverine and White Tail Deer mines.

[9] *Transactions of the Institution of Mining and Metallurgy*, vol. 22, p. 561 (1913)

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Further proof is had in the presence of gossans several hundred feet below the water level in the Lowell and Hoatson mines.

The general plane of the oxidation corresponds very closely to the known pre-Cretaceous erosion surface, and the tilt of the oxidation is, therefore, the present tilt of the Cretaceous sediments. If the ground-water level at the time of oxidation had been an inclined plane that followed the Devonian horizon, due to the permeability of the purer limestone above, there would not be abundant unoxidized orebodies in Escabrosa limestone in the Sacramento and Junction mines. And if the ground-water level had stood several hundred feet lower in those mines at some post-Cretaceous time and had risen, say, due to valley fillings, it is probable that the ground water, now directly over that rise, would stand horizontal, and would keep to the old plane, or even a lower one farther west, which is far from being the case.

There is field evidence of another form to show that the oxidation was pre-Cretaceous, and this is the presence of gossanized fragments of porphyry, as well as fresher ones at the base of the Glance Conglomerate. The exposed pre-Glance topography also shows hard gossan reefs around the southeast side of Sacramento Hill.

Additional weight is given to the idea of pre-Cretaceous oxidation by the geologic history of this region. A very long erosion and a consequently long oxidation period is recorded between Pennsylvanian and Cretaceous times by the disappearance of the paleozoic column off of the north side of the Dividend Fault. And while the granite porphyry intrusion may not have taken place till a good part was gone, the nature of the rock as well as of the later minerals formed presuppose a considerable covering of sediments which were removed by erosion before Cretaceous time.

It is, therefore, probable that the main part of oxidation and enrichment as found at present took place when the Paleozoic sediments were almost horizontal, the only tilt noticeable being slightly to the southeast.

There is very little evidence of superimposed oxidation since Cretaceous times, due very probably to the later ground-water level being higher than the first. Narrow, limonitic seams, and very slight enrichment on primary sulphides in the west end of the camp are the only visible effects of this oxidation. The post-ore faulting which is also post-oxidation in most cases, is not found very frequently in the thoroughly explored portion of the district, but will very probably be found more frequently when operations extend into other blocks.

Relations of Oxidation to the Formations and to Metamorphism

The circulation of meteoric waters has been very much influenced by the material they traversed, and this was dependent upon the formation and the degree of alteration.

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In the Carboniferous limestones the massive purer beds had a tendency to crack and break rather than to fold, and waters could dissolve out channels through which extensive oxidation could be carried down rapidly. The same applies to the purer lime beds in the Devonian and Cambrian formations.

The more impure beds, which are also more thinly laminated, had a tendency to crush and fold, and while as a whole they may be found more altered by meteoric waters, it is because they retarded and distributed the effects all through the rock. This is noticeable in the Martin and Abrigo formations especially, the Abrigo being by far the most impervious.

The ability of meteoric waters to circulate and oxidize quickly in the Escabrosa limestone has made possible the abundance of residual ores in this formation. These are mostly carbonates fixed in this form against total leaching by the continually circulating cold waters. In a great many cases, however, the residual ore is found in the form of sulphide, mostly chalcocite, and also, in rare instances, as the original chalcopyrite and bornite. These sulphides are called residual because they are far above the level of normal enriched sulphides, and they have limonitic gossans or carbonate and oxide ores under them. Usually they are very pure, rich sulphides of copper, and are in the primary zone of sulphide replacement where the adjacent rock is pure calcite and contains little or no pyrite. Occurrences like this are found in the limestones all the way to surface where the main ore zone may be a thousand feet below.

The same quick circulation through the Carboniferous limestones has usually brought the zone of complete oxidation directly against the primary ores, with a thin intervening zone of secondary sulphides, which, as will be shown later, is not one of enrichment, but of leaching of values. The slower circulation in the impurer Devonian and Cambrian produced fewer residual orebodies, but a deeper zone of enrichment.

In the porphyry and contact breccia the differences in the effect of secondary alteration are far more noticeable, as can be seen from Plates 5*b* and 6*b*. Both the porphyry and breccia retard the meteoric water circulation, the breccia almost entirely, and the porphyry making it slow enough so that sulphide enrichment is at its most efficient point.

A special case which illustrates the difference in the penetration of oxidation through different rocks is given in Plate 12, which is an east-to-west section through an orebody on the 1,500 level of the Sacramento mine, at present being extracted. This ore occurs along the Sacramento dike, here cutting Escabrosa limestone and having a projection downward to the east as shown. The ore replaces the limestone, which is not appreciably metamorphosed at a distance from the porphyry. The replacement is very perfect along the beds, as certain cherty bands are left untouched in the midst of the ore.

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The oxidation feature that is noticeable is that the ore is almost unaltered close to the main dike, enriched farther away, converted into mixed oxide and sulphide under the thin portion of the porphyry, and entirely into carbonate next to the limestone. The water circulation here must have come partly from the east and partly from underneath through the limestone, as the porphyry ordinarily presented an effective barrier to circulation perpendicular to the beds before the tilting.

The oxidation features are even more noticeably related to the processes of primary ore alteration than they are to that of the formations. Intense silicification such as is found in the contact breccia and in a great part of the contact metamorphic limestone of the Gardner mine made circulation so slow that in these places the primary ore zone remains at a level far above those of adjacent occurrences. Also, the oxidation is so complete where there is any meteoric circulation at all that the zone of leaching comes to within a few feet of the primary zone, with only a thin layer between where covellite is most abundantly found in the district. These relations are well shown in Plate 5, Section A–A just west of Sacramento Hill.

Sericitization produced rocks in which the circulation was slow, but in which the secondary waters had opportunity to remain acid, as nearly all the lime was removed in the primary alteration and pyrite was abundant all through the rocks. Under these conditions the depth of the zone of secondary enriched sulphides was greatest, and the enrichment most efficient. At the same time, in the rocks very much sericitized there are no residual carbonates found, due of course to the lack of lime. If, however, the ore formation extended into unaltered limestones above those sericitized, carbonates might be found higher, as is the case in the Uncle Sam mine. Native copper and oxides are developed to a great extent in a shallow zone between the gossans and enriched sulphides in aluminous ground due to primary sericitization.

The ores in Sacramento Hill porphyry, and in the zone of contact-metamorphic rocks around the main intrusives are the best examples of the coincidence of sericitization and deep sulphide enrichment, as can be seen in Plates 5*a*, 5*b*, 6*a* and 6*b*. The amount of enrichment even in these cases has not been exceedingly great, as the original ores were rich to begin with, and very pyritic bodies have remained poor, first because pyrite is replaced with so much more difficulty by chalcocite than bornite and chalcopyrite, and second, because pyritic masses are generally quite solid, more siliceous and impervious.

The ore developed in Sacramento Hill is extremely irregular in shape and grade and owes its irregularity more to primary mineralization than to secondary enrichment, because so far as has been seen, the relative amount of primary bornite left in parts at the top of the ore close to the oxidation line is about the same as at the deepest levels. A section show-

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ing the ore grades is given in Plate 11, worked out from all information available. This body has not yet been mined extensively. The grades of ore can be seen to bear very little relation to the limonitic oxidation.

The orebodies classed as sulphide replacements of unaltered limestones are ordinarily found in their primary state or as residual carbonates. Oxidation is rapid through unaltered rocks, and waters usually carry enough lime to fix the copper as carbonate. In such a zone, carbonates are apt to be found to the very top of the mineralized zone, many times above hundreds of feet of gossanized ground.

TABLE 3.—*Showing Efficiency of Enrichment by Descending Waters*

Orebody	State of Oxidation	Tons per Cu. Ft.	Pounds Cu per Ton	Pounds Cu per Cu. Ft.
	Sericitised Contact Metamorphic Zone			
14-7	Primary	0.1143	126.2	14.42
13-13	Enriched sulphide	0.1008	189.8	19.10
	Sulphide Replacement Orebodies in Abrigo			
12-3-18	Primary	0.117	104.0	12.17
13-10	Enriched	0.0984	147.0	14.46
White-Tail Deer	Residual carbonate	0.0578	142.0	8.20
	Sulphide Replacement Orebodies in Martin and Escabrosa			
9-8-58	Primary	0.1255	130.0	16.31
8-17-11	Enriched and oxide	0.098	112.0	10.97
8-17-4	Residual	0.0609	113.0	6.88
8-16-5, 8-16-16	Residual	0.0609	137.0	8.34
8-16-18	Residual	0.0609	108.0	6.58
	Sulphide Replacement and Contact Bodies in Escabrosa			
1300 Junction	Primary	0.1111	116.0	12.90
16-3-39	Primary	0.140	140.2	19.63
16-3-68	Partly enriched	0.1267	192.4	24.38
15-12 Sulphide	Secondary sulphide	0.115	130.0	14.90
15-12 Oxide	Oxide	0.08	224.0	17.92
16-3 Oxide	Oxide	0.08	268.0	21.40
15-6-1, 15-6-9	Residual carbonates	0.065	170.0	11.05
15-13	Residual carbonates	0.065	188.0	12.20
14-20	Residual carbonates	0.065	114.0	7.41

Table 3 has been prepared to show approximately the efficiency (or lack of it) of enrichment due to descending meteoric waters.

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Orebodies have been taken for analysis which were originally of similar composition, as far as knowledge of them goes, and from mining records kept for a period of several years, the contents of copper in pounds per cubic foot in place has been figured. This is of course a figure for the ore mined, and must differ somewhat from the actual content of all the ground in an orebody, depending on the method of extraction used. All errors have been considered, however, and the relative value of the figures is about correct. Unfortunately the ores in the breccia and silicified ground are all primary and no comparisons can be given, and all the ore in porphyry is enriched.

From these tables it will be seen that the copper content per unit of volume is increased materially in the sericitized contact zone and in the more aluminous Abrigo formation where the secondary sulphides are found. Chalcocite formation is accompanied by leaching out of values in the replacement bodies, and all the residual ores show dissipation of copper contents. The narrow oxide and native copper zone is generally the richest of all.

Gossans

In regard to the gossans formed in the different zones of primary alteration, they can be said to extend downward for the greatest distances in the more permeable ore zones—this is true for those classed as sulphide replacements. These gossans are likely to contain residual ores anywhere. The thoroughly oxidized portions, where the formations were silicified or sericitized or chloritized are apt to be more completely leached out, and penetrate the ore zone least in highly quartzose ground. In composition the gossans of course reflect their source.

The only generalization that can be made about gossans derived from lean sulphides and from pyritic copper ores is as follows. The mixed sulphides of copper and iron are more readily attacked by oxidizing solutions than the straight iron sulphides, therefore producing stronger solutions. These stronger solutions have been found to have a segregating action on the components of the surrounding rocks, separating to some extent silica, alumina, and lime, and also depositing the iron oxides in purer form. The gossans from pyrite alone have been observed to leave the gangues and oxides of iron in about the same mixtures as in the original primary material.

These observations are of course susceptible of an infinite number of variations depending on the original composition of sulphides, of their gangues, and of the surrounding rocks, but in general they can be used to some extent in judging the kind of ground under an oxidized portion of an ore zone.

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Minerals of the Zones of Enrichment and Oxidation

Siderite

Under the zone of secondary enrichment the presence of siderite has already been noted. This mineral occurs sometimes in veinlets replacing either sulphides or chloritic minerals or calcite. At other times siderite is found in a curious box-work honeycomb structure as if it had replaced laminated rock along certain planes and the intervening material had been later dissolved out. The ferrous carbonate has a distinct selective action in replacing a rock, as it usually leaves intact pyrite grains, which then appear contemporaneous with the siderite. Sphalerite and galena are also found more abundantly with the siderite than around in the unaltered formation, and this fact has pointed to the possibility that these two sulphides may have been here deposited by secondary solutions.

Copper minerals in the siderite zone are very faintly coated with chalcocite. Gypsum is also found in vugs, even deeper than siderite, but apparently in late circulation channels.

Siderite probably formed as an advance mineral, as oxidation progressed downward. This may be proven by the closely similar forms taken by limonite in some gossans above enrichment, and the presence of siderite cores in that limonite. The iron carbonate has played an important economic rôle in serving as a precipitant for rich cuprite ores in the Czar mine, where copper-laden solutions have encountered a flat bed in Devonian limestone previously replaced by siderite and kaolin, and have precipitated cuprite and native copper in considerable amount. Throughout the district cuprite crystals can be found on oxidized or partially oxidized siderite.

Chalcocite

In the zone of sulphide enrichment chalcocite was the only secondary mineral of copper formed to any extent. It was usually formed by replacement of the bornite and chalcopyrite alone, leaving the pyrite untouched, or only slightly coated. The broken up, replaced pyrite grains in the primary ore remain in just about the same amount and with the same relations to chalcocite that they had to the primary minerals. When chalcocite and pyrite are the only remaining sulphides, it is difficult to believe that the iron mineral has not been abundantly attacked by secondary action. Pyrite has been replaced to a considerable extent, however, in the enriched ores of the contact-metamorphic type, by a sooty variety of chalcocite that possibly contains considerable covellite, since the latter has been seen coating pyrite grains in these ores.

In replacing the primary minerals chalcocite behaved in different ways depending on the general feature of the orebody. In the ore of

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Sacramento Hill, pyrite is barely coated in the very sericitic portions of the porphyry, and is not affected appreciably in silicified rock. The selective action in replacing bornite is better marked here than anywhere

FIG. 1.

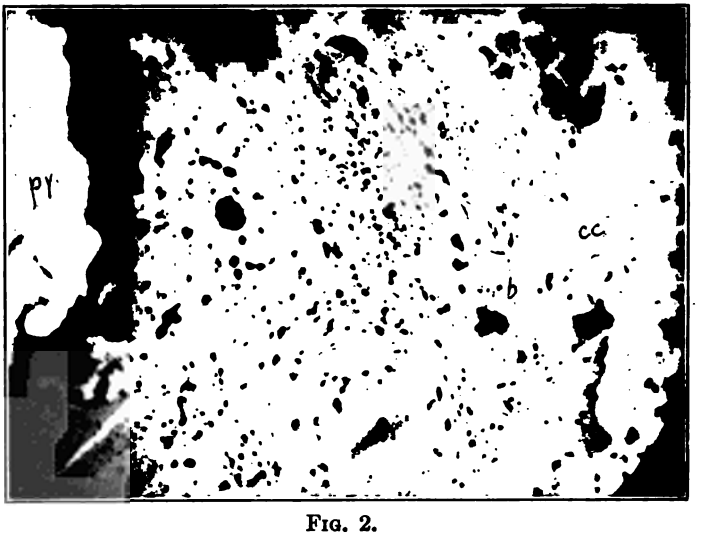

FIG. 2.

PLATE 15.—CHALCOCITE, *cc*, REPLACING BORNITE, *b*. PYRITE, *py*.

else as is also the replacement along crystallographic lines of the bornite in the arrangement commonly known as "lattice structure." This form of alteration grades into the usual one of veinlets and irregular remnants, as is shown in Plate 15. The field occurrence of this "lattice

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structure" alteration has been quite carefully followed in the district and found everywhere to coincide with slow or incipient enrichment in rich ores, and where the selective enrichment of bornite in place of chalcopyrite is well-marked. This has led to the conclusion that it is a delicate process in which crystallographic weakness can be of some influence. Etching of the chalcocite thus formed shows isometric figures, probably due to included or dissolved cupric sulphide,[10] but there are also stringers in which the chalcocite has orthorhombic cleavage, and these stringers can be seen to form a network of veins. The proximity and association of this orthorhombic chalcocite to major cracks and circulation channels, where even limonite is formed, points to its being due probably to the working over and recrystallization of the copper sulphides first derived from bornite.

Chalcocite of the "sooty" variety is found most commonly in the enriched ores of the zone of contact-metamorphic limestones. Massive chalcocite is here rare, and then only at the top of the zone of enrichment, with considerable native copper present.

In the ores that replace limestone, the formation of chalcocite is accompanied by leaching and formation of abundant cavities in the ore, a process that is reflected in the decrease of values as previously shown. This same thing is true in ores that have a very siliceous gangue, such as those in contact breccia. Pyrite is here oxidized to limonite even before the copper minerals are replaced by chalcocite, as is shown in Plate 16.

In the Southwest mine there is an occurrence of chalcocite that is somewhat different from the usual. The Devonian-Escabrosa contact beds are replaced by silica, almost chalcedonic in character, accompanied by abundant fine flaky specularite. This siliceous mass connects up with the main silica breccia replacement in this mine.

The top Devonian beds under those silicified have been hardly altered at all, except for a very slight amount of quartz addition, some recrystallization of calcite in stringers, and the formation of very fine veinlets as well as disseminated grains about a millimeter in diameter of what was apparently bornite and chalcopyrite, now replaced almost entirely by chalcocite. The original copper minerals have been found partially replaced by chalcocite in the center of coarse calcite veins that protected them. Pyrite is very rarely found.

The formation of chalcocite has taken place in this instance under conditions which would hardly allow the circulation of acid waters. A considerable amount of carbonates has formed, but in general the sulphide is relatively abundant. The possibility of chalcocite formation with the acid generated in the replaced mineral itself is indicated in this orebody, though this is by no means perfectly clear.

[10] Posnjak, Allen and Merwin: *Economic Geology*, vol. 10, No. 6.

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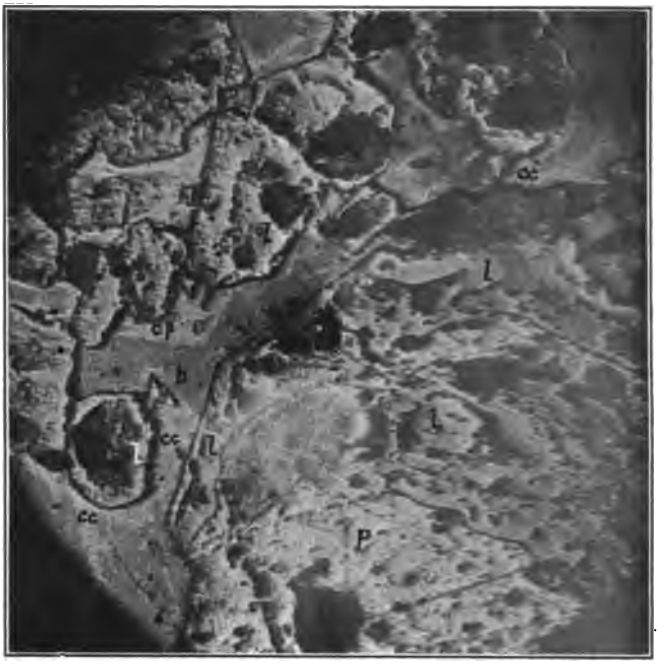

FIG. 1.

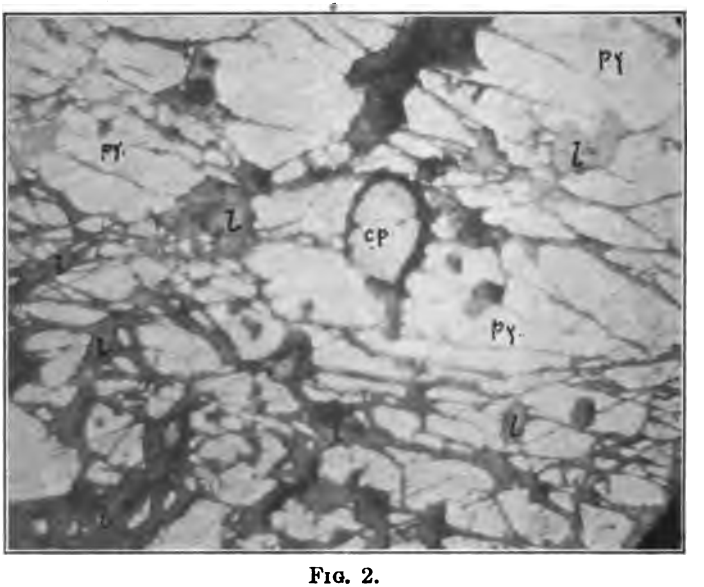

FIG. 2.

PLATE 16.—SHOWING OXIDATION OF PYRITE, *py*, TO LIMONITE, *l*, BEFORE REPLACEMENT OF COPPER MINERALS BY CHALCOCITE, *cc*.

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Covellite

Covellite is formed in very minor amounts especially in the narrow oxidized zone above the contact breccia ores. It replaces chalcopyrite in preference to bornite, and is found always in microscopic needles. The only other occurrences of this mineral are where the enriched zone touches highly silicified, not very sericitic ores.

Bornite and chalcopyrite have been found in many ores of the district, formed by secondary agencies, as transition products, but they are of no economic importance.

Gangue Minerals

The gangue minerals that accompany the enriched sulphides are halloysite, kaolin, gibbsite, alunite, and serpentine with mixtures of quartz, calcite and the oxides of iron. Of the aluminous minerals, halloysite seems to be the most common, especially in the zone of highly altered limestones. Where the rock is sericitized so that the mica flakes are well formed, sericite persists unaltered through the process of enrichment and oxidation.

Oxide Ores and Native Copper

In the oxide ores just above the secondary sulphides, cuprite, native copper, tennorite, melanochalcite and delafossite are found, but the first two are the only ones of economic importance. The association of cuprite and siderite has already been given. The zone of oxide has in many instances encroached entirely on the sulphides, leaving only a remnant here and there of the latter. Flat orebodies along the limestone bedding are especially susceptible to this process.

The minerals of the residual orebodies are mostly the carbonates malachite and azurite, with some chrysocolla, aurichalcite and brochantite. Besides these, there are the sulphides already mentioned.

Carbonates

The association and the occurrence of the carbonates has been very well described by Dr. Douglas[11] and by Ransome[12] and no additional features can be given, except to repeat that these minerals are found all through the oxidized portion of the ores that replaced unaltered sediments, and are still a very important part of the ore reserves of the district.

Other Ores

In recent years oxidized lead ores have become more and more important, especially from around highly siliceous replacements in Martin and

[11] *Transactions*, vol. 29, pp. 511–546 (1900).
[12] *Loc. cit.*, p. 125.

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Escabrosa limestone. Most of this ore has come from the Southwest mine and occurs at the edge of the siliceous breccia, as cerussite with some

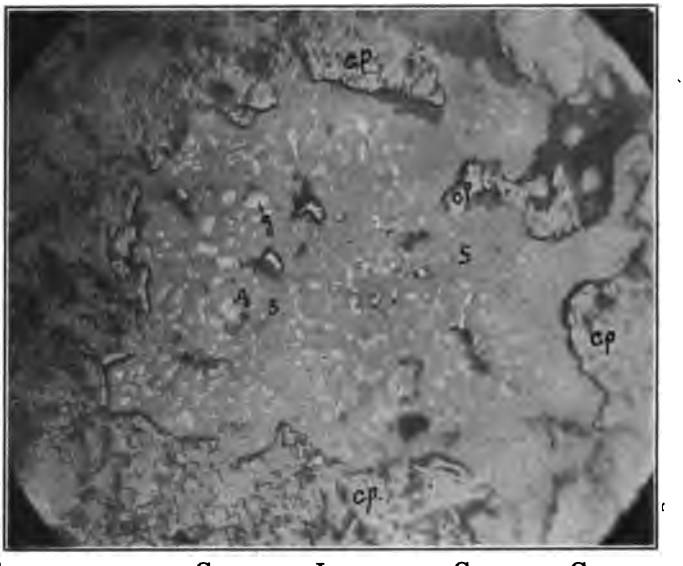

FIG. 1.—SPHALERITE, *s*, AND GALENA, *g*, INCLUDED IN GRAINS OF CHALCOPYRITE, *cp*.

FIG. 2.—CHALCOCITE AND MALACHITE. ALTERATION RESEMBLING INTERGROWTH. PLATE 17.

anglesite and variable silver and gold values. The primary minerals from which this ore was derived are unfortunately not present now, but

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it is very probable that the main one was galena, which, as has been before stated, is apt to come in vein-like masses above the general level of the copper ores. Lead ores have also been found with similar associations in the Gardner, Lowell and Briggs mines.

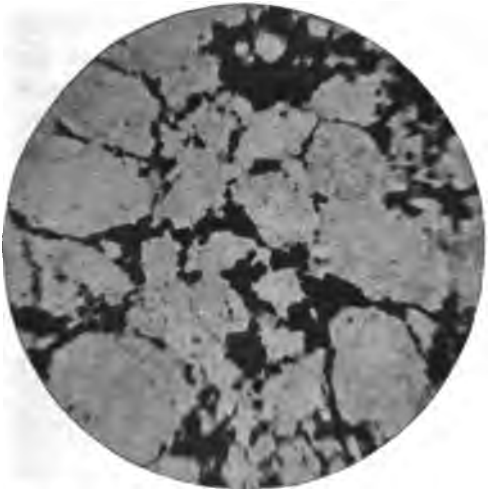

FIG. 1.—BORNITE AND PYRITE REPLACING THE CEMENT IN A CALCAREOUS SANDSTONE.

FIG. 2.—PYRITE REPLACED BY LIMONITE.
PLATE 18.

Vanadium, in the form of cuprodescloizite has been found in the Shattuck mine in considerable amount, and rarely in the Dallas and Sacramento. The occurrence may prove in future to be of economic importance.

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Manganese ore, in the form of psilomelane and braunite, with little pyrolusite, has been mined lately from several places on the surface of the district. It is found in irregular bunches as a replacement of limestone beds in the Naco or Escabrosa, close to the contact breccia, or to highly silicified outcrops of the sediments. The origin of this manganese is not yet known, for although manganese is found quite abundantly with some of the carbonate ores underground, it is here in the form of soft earthy pyrolussite due apparently to the concentrating action of oxidizing water.

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Mention must also be made of the report on the mining geology of the Copper Queen property by John Mason Boutwell prepared in 1908–1909 for the use of the mines, one of the results of this work being the establishment of the present Geological Department.

Acknowledgments

To the Copper Queen Consolidated Mining Co. for its support and encouragement in the preparation of this report, the greatest thanks are due.

Arthur Notman, Chief Geologist of the Company, has lent the utmost assistance, contributing from his store of knowledge gained by long and careful investigation in the district and in this part of the Southwest as a whole.

Former members of the Geological Department, and our colleagues of the Calumet & Arizona Mining Co., have also contributed materially to the information presented.

To Harvard University, and Prof. L. C. Graton in particular, we are indebted for being able to present the result of extensive petrographic and metallographic investigation, as 2,000 specimens from the Copper Queen geological collection were studied by one of the authors in the geological laboratories at Cambridge. Some of the photomicrographs were taken at the University of Arizona, for which thanks are especially due to Prof. C. H. Clapp.

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Assistance has been received from the engineering department of the company in the way of maps and stope records, and from the chemical department in the form of the analyses that are given in the report. Thanks are due to the operating department of the mines for a great deal of information, also for coöperation in pointing out and solving most of the geological problems of the district.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Mine and Mill Plant of the Inspiration Consolidated Copper Co.

BY H. KENYON BURCH, B. S.,* MIAMI ARIZ.

(Arizona Meeting, September, 1916)

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* Chief Engineer, Inspiration Consolidated Copper Co.

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INTRODUCTION

THE Inspiration Consolidated Copper Co.'s plant at Miami, Ariz., was designed and built to make possible the profitable working of a low-grade, finely disseminated copper deposit containing 100,000,000 tons of ore averaging 1.64 per cent. in copper.

From the beginning it was evident that the plant could not be kept integral but that a break would have to be made somewhere in the flow-sheet, removing at least the concentrator to a site more suitable than any available near the mine. It was finally decided, after considering numerous arrangements, to do the coarse crushing at the mine, to store the crushed ore in a bin from which it could be loaded into railroad cars and to haul it to the concentrator, an excellent site for which was available about 1¾ miles from the mine.

The original intention was to equip a plant to treat 7,500 tons of ore per day, but through the acquisition and proving up of additional ore reserves, the introduction of the Ohio caving system, and the excellent results obtained in the test mill (which made it possible to treat a lower-grade ore than had been thought possible) it was evident that a plant of much greater capacity should be supplied. It was, therefore, decided to treat approximately 15,000 tons of ore per day, the duties of the four main divisions to be as follows:

Division	Operating Time in Hours	Capacity in Tons per Hour	Daily Capacity in Tons	Available Capacity in Tons
Hoisting plant.....	15	1,000	15,000	
Crushing plant.....	15	1,000	15,000	
Storage bin.....	....	.....	.....	25,000
Concentrator.....	24	625	15,000	

It was realized from the beginning that with such an enormous tonnage to treat it would be well worth the time and cost to carefully work out a flow-sheet. Accordingly, a gravity test mill was erected and placed in operation near the Joe Bush shaft in November, 1910, and its operation was continued until August, 1911. Soon after this, flotation began to attract considerable attention in this country and realizing the possibilities that might arise through its systematic investigation, it was deemed advisable to go into the process in detail. A 50-ton Minerals Separation machine was first erected, which after a few months thoroughly demonstrated that flotation was applicable to the concentration of the Inspiration ores, but in order to better determine its proper place in the flow-sheet, a 600-ton test mill was designed and erected in 1913 on the benches of the concentrator site, the grading for which was at this time completed. Numerous flow-sheets were experimented with, the final result being the

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one now in use in the concentrator. A discussion of certain phases of the results accomplished in the test mill will be given in another section, mention being made of it here simply for preserving the proper sequence.

That the large-scale test-mill method for working out flow-sheets for large plants is the only logical method, is evidenced by the fact that in nearly every stage of the treatment either an entirely new machine has been adopted or a new application has been made of a standard machine; the result in each case being either increased efficiency or a more economical arrangement. The first mill was to have a capacity of 7,500 tons per 24 hr. This design covered an area of approximately 350,000 sq. ft., or a little over 8 acres. The mill now in use covers an area of approximately 125,000 sq. ft., or a little less than 3 acres, but has a capacity of 15,000 tons. The recovery in the two types of plants on favorable ores, that is, ores not carrying over 10 points of oxide, may be closely estimated at 70 per cent. for the first, and 85 per cent. for the second.

In order to account for the long period required for the design and construction of the plant, it may be interesting to note that six complete designs for the concentrator were executed, the idea being to keep this work abreast of the developments brought out by the test mill. Design No. 2 was completed and a contract entered into for structural-steel requirements. A portion of the steel had been fabricated when flotation developments pointed to the fact that wet gravity concentration could be greatly improved upon. At this point all work on the steel contract was stopped and that portion of the contract pertaining to the concentrator was cancelled. Although facts relative to flotation continued to develop, in July, 1913, the steel design for the present concentrator building was completed. This building was no sooner erected than very marked changes in grinding machinery began to develop, the ultimate result of which was another altogether new arrangement for the entire mill. As it stands today there are but three single pieces of machinery in the mill building occupying the places originally intended for them, these being the three electric cranes now in use. Considering these changes it is remarkable that such a good arrangement was found possible.

PLANT SITES

The sites for the plants as finally decided upon permitted of an arrangement entirely adequate to meet the proposed requirements, but the following important features may be noted:

The mine plant site is contiguous to the orebodies but at a safe distance from ground to be caved; it occupies a position as regards elevation well suited to a favorable hoisting arrangement and allows of a downgrade to the concentrator of 0.3 per cent. The shortest line from the shafts to the orebodies approximately bisects them, making possible a

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direct underground haulage system; the site occupies a remote corner of the property and does not lie in the general trend of the mineralized zone. The mill site has a slope favorable to the type of mill erected and is of sufficient area to allow of an economical arrangement of the various units of the plant. There is ample storage space for tailings and an excellent reservoir site available for the storage of water.

TYPE OF CONSTRUCTION

As the life of the property was estimated to be 17 years, or longer, depending upon new developments in the treatment of lower-grade ores, it was decided to make the various structures of a semi-permanent char-

FIG. 1.—MINE PLANT OF INSPIRATION CONSOLIDATED COPPER CO., MIAMI, ARIZ. LOOKING WEST. SHOWING COMPRESSOR HOUSE, MAIN SHAFTS, COARSE-CRUSHING PLANT AND STORAGE BINS.

acter. The buildings are of steel with corrugated steel coverings, except the concentrator which has a four-ply composition roof. All windows as well as skylights are of rubber glass. The floors throughout are of concrete, the retaining walls of reinforced concrete, and machinery foundations of massive concrete with little or no reinforcing. Reinforced concrete was used wherever applicable.

MINE PLANT

Under this general heading will be considered that portion of the plant with its adjuncts which takes the ore from the mine cars, hoists it to

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surface, reduces it to a size suitable for mill treatment and places it in storage, ready for transportation to the concentrator.

The relation between the orebodies and the mine plant can be described as follows: Conceive of a chain of orebodies the lateral extensions of which form a parabola with its vertex pointing to the north and having a length of approximately 9,000 ft. Place the main haulage ways on either side of the axis, 60 ft. apart, and open them to surface through two three-compartment shafts, 102 ft. centers, and about 750 ft. from the nearest approach of the orebodies. Group the mine plant symmetrically about the axis between the shafts and the orebodies and a general idea of their relation is obtained.

Dual Arrangement

As a result of the preliminary studies, the dual arrangement of the mine plant was evolved, the primary consideration being the necessity of handling 1,000 tons of ore per hour, which in itself precluded the use of a single shaft. Two shafts would also insure continuity of service. It was then considered advisable to follow this idea a step further and make the whole mine plant duplicate in its arrangement, which would give reasonable assurance against total shutdowns and also permit of a better load factor.

Underground Haulage

Underground haulage, for transporting crude ore from the stope chutes to the loading stations at the main shafts, is confined to two levels, the 600 or main haulage level, and the 400 level. The main haulage level is, in turn, made up of two distinct systems, each serving its main shaft. All haulage is to be controlled by a block signal system.

Two types of motive power for underground haulage were considered —electric and compressed air. On account of less danger to life due to the elimination of bare conductors, the compressed-air locomotive was chosen rather than the electric. The difference in the efficiency of the two systems is probably a small percentage of the total cost. "Safety First" was therefore the deciding factor.

The locomotives are of the two-stage, four-wheeled type, and have a weight on the drivers of 10 tons. The initial cylinder air pressure is 250 lb., and the charging pressure 800 lb. per square inch. The haulage capacity per charge was estimated at 50 ton-miles. The locomotives haul 25 cars, each of 5 tons capacity, on a 30-in. gage track over a 0.4 per cent. grade in favor of the load, and will negotiate a curve of 38 ft. radius.

All ore tapped from the stope chutes is broken to pass a 12-in. grizzly so that no further attention has to be given to it after it is once in the cars. At the shafts the cars, weighing 14,000 lb. each loaded, are dumped, five at a time, by motor-operated tipples, one for each shaft on the 600

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or main haulage level, with a third on the 400 level. All hoisting is done from the main haulage level.

Tipples

The tipples have an overall length of 56 ft., and make a complete revolution in 15 sec. The driving shaft is connected through suitable gearing to a 35-hp. motor which runs continuously in one direction, the starting and stopping of the tipple being accomplished by means of a friction clutch located on the intermediate shaft. Each tipple is provided with an automatic stop which brings it into the proper position for running on and off the cars, this stop being released by means of a foot lever on the operator's platform. The two main tipples are operated from a centrally located platform.

Underground Pockets

The tipples on the 600 level dump directly into two main underground pockets (Fig. 2), which serve as storage for the four automatic measuring and loading devices that are placed beneath them. The capacity of each pocket is 1,600 tons, or 800 tons for each loader, which, with full pockets, will run the crushing plant about 3 hr. They are of reinforced-concrete construction throughout and are lined with 2-in. planks. The upper pocket has a capacity of 500 tons and is connected to the lower pockets by an inclined chute.

Automatic Measuring and Loading Devices

To carry out the automatic feature of the hoisting equipment, it was desirable to make the loading of the skips automatic. There being nothing on the market which could be used for this purpose, it was necessary to work up complete designs in accordance with original ideas.

In a few words the device (illustrated in Figs. 3*a* and 3*b*) can be described as follows:

Ore from one of the 1,600-ton underground pockets rests on a roll feeder which is actuated by a pawl and ratchet wheel, and is fed into a 12-ton measuring hopper. This hopper is suspended between the legs of a U-shaped scale beam with fulcrums at the two ends. Through a link at the middle of the yoke the proportional weight of the hopper and ore is transferred to the main scale beam which carries a counterweight of 1,370 lb. Motion obtained from the counterweight scale beam when the hopper becomes filled is utilized to operate a system of levers and lift the pawl from the ratchet wheel, thus stopping the roll feeders. The hopper is now ready to be emptied.

As the returning empty skip settles on the chairs, it strikes an arm

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projecting out into the shaft. This arm is connected by a system of links to the shaft on which the yoke of the gate forming the bottom of the hopper is hinged. The motion imparted to the system by the skip trips the gate yoke, the weight of the ore in the hopper causes the gate to drop

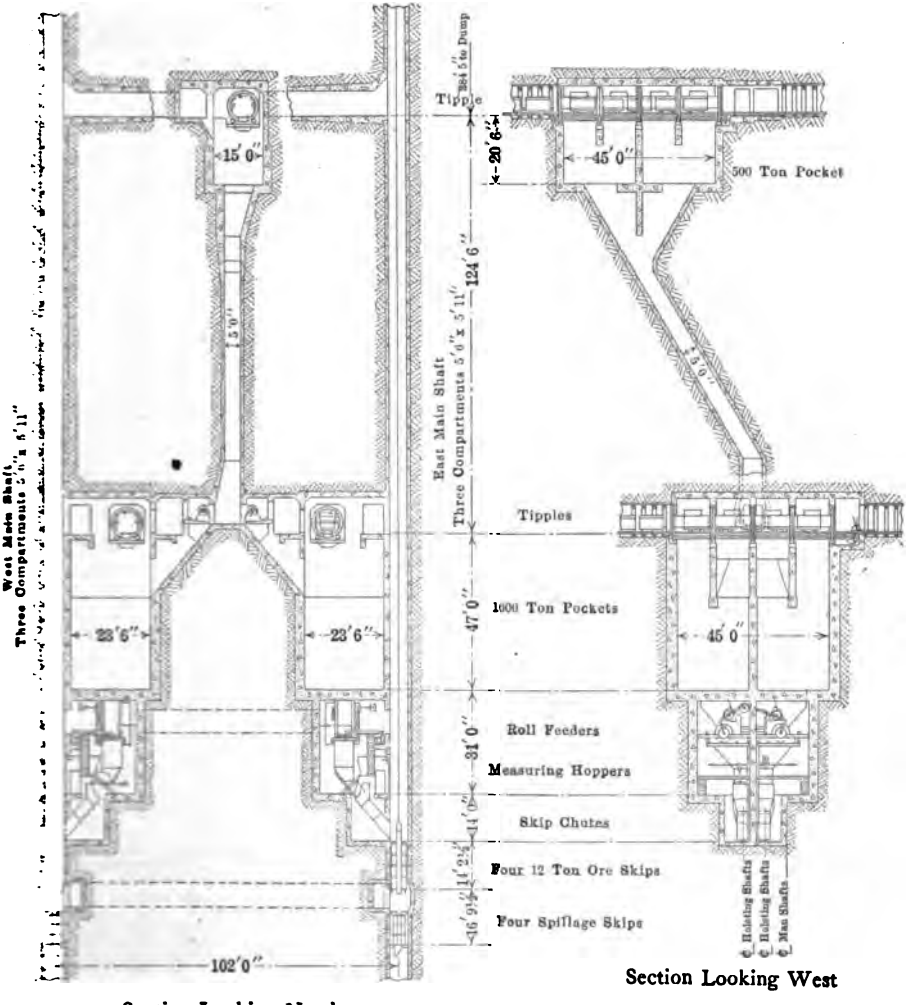

FIG. 2.—ARRANGEMENT OF UNDERGROUND POCKETS.

into the skip chute, and the hopper is emptied into the skip. As soon as it is empty, a counterweight on the gate causes it to close, all motions are reversed and the measuring hopper is again filled.

A cut-off gate prevents the roll feeder from becoming bare in case the pocket should be emptied. A 5-ft. layer of ore thus protects the feeder

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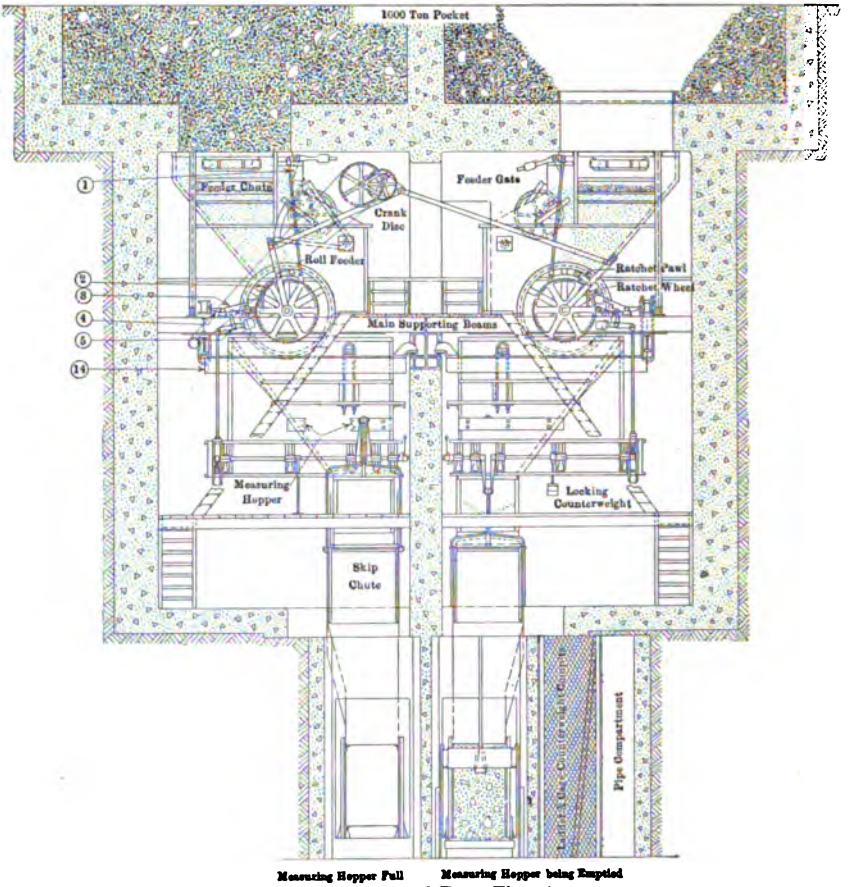
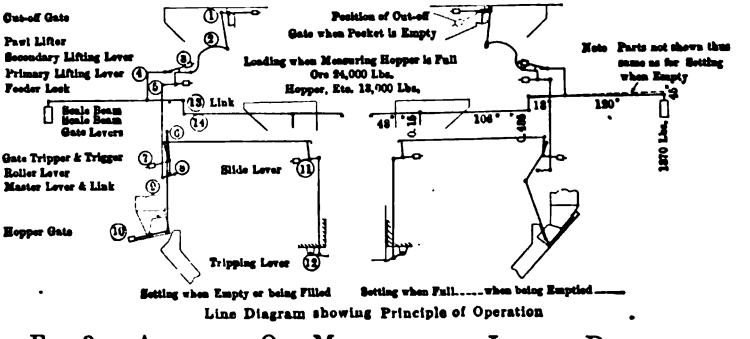


Sectional Front Elevation



Line Diagram showing Principle of Operation

FIG. 3a.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.

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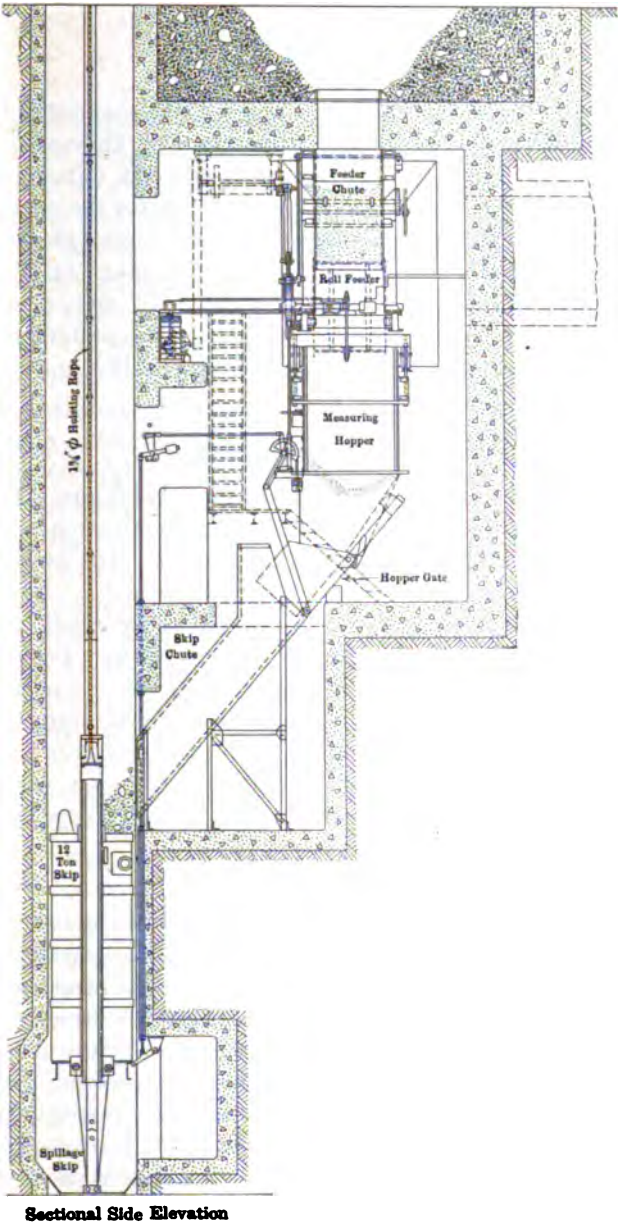

FIG. 3*b*.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.

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and does away with excessive impacts due to the dumping of the ore from the tipple 50 ft. above. A 5-hp. motor furnishes the required power for the roll feeders of each pair of loaders.

Shafts

Taken individually the shafts do not deviate materially from usual designs, being of reinforced concrete throughout. They are of the same size, each being made up of three compartments 5 ft. 6 in. by 5 ft. 11 in., the two south compartments of either shaft being for the ore skips. The north compartment in the West Main serves the man elevator and the corresponding compartment in the East Main is taken up by pipe lines, ladderways and the elevator counterweight. All platforms, supports, and ladders in this compartment are of steel. The conduits carrying the power and lighting circuits underground are in the end walls, being brought to junction boxes every 100 ft.

Ore and Spillage Skips

The ore skips are built according to special designs. They have a nominal capacity of 12 tons and weigh 17,000 lb. each. The overall length is 21 ft., the skip proper being 14 ft. deep. The body is of $\frac{3}{8}$-in. steel plate with liners to take up concentrated wear.

In order to keep the sumps at the bottom of the shafts clean and to facilitate the handling of ore which does find its way below the skips, spillage skips have been installed (Fig. 2). These rest on chairs 17 ft. below the ore-skip chairs and by means of hinged aprons completely close off the shaft. When one is filled it is attached to the bottom of the ore skip and hoisted to the top of the underground pocket, where by opening a hopper bottom it is discharged. They have a capacity of 9 tons each.

Automatic Hoists

At the time it was decided to sink two shafts the conditions seemed to favor the use of two entirely independent hoists, located symmetrically on either side of the main headframes. It was next proposed to put the two hoists under the same roof but to operate them independently of each other. Then on account of the advent of several new conditions it was deemed advisable to make the operation of the hoists automatic.

While the hoists are entirely automatic in their operation, either can be manually operated independently of the other if so desired. No attempt will be made to cover the automatic features of the installation as these are presented in detail in another paper.[1] Suffice it to say here

[1] H. KENYON BURCH and M. A. WHITING: Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co., *Bulletin* No. 111, pp. 583 to 596. (March, 1916.)

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that the problem as presented has been solved in a most efficient manner, sufficient proof being the entire satisfaction that the equipment has thus far given.

Man Elevator

When the question of handling men came up, several possibilities as regards point of distribution presented themselves. They could be handled through an existing tunnel, hoisting or lowering to the working levels through an inclined shaft, or entirely through an inclined shaft, or through one of the main shafts. Whichever scheme proved the most feasible, the idea was to have the men do as little walking as possible, ample provision to be made for transferring them from change house to working places. It was finally decided that distribution from the main shafts was the most logical and the shafts were equipped accordingly.

The next consideration was the type of hoists to be used. Standard mine practice favored a separate hoist, manually operated and working against a counterweight. This type made necessary the services of two attendants, one on the cage and one at the hoist, but in working out probable operating costs the idea was conceived of eliminating the hoist attendant by an application of the elevator principle. Studies along this line were at once begun, but no great amount of enthusiasm among the manufacturers could be aroused on account of the unusual conditions. Equipment was wanted to handle a counterweighted, double-deck cage, weighing 7,100 lb. and loads of 7,500 lb., the maximum speed of the cage to be 800 ft. per minute. The fact that the hoist and counterweight would have to be removed from the elevator shaft also presented a new feature in the design. It may be said, however, that placing the hoist in the headframes directly above the shaft, as is the usual practice in office building design, was at one time considered.

The equipment as installed is operated entirely from the cage, the hoist, motor-generator set, and control apparatus being in the hoist house about 220 ft. away. The motor driving the direct-current generator on the motor-generator set has a rating of 190 hp., the generator developing 130 kw. The hoist is driven by a 158-hp. direct-current motor. It is provided with a safety brake, so arranged that when the hoist is stopped the brake is automatically applied to hold the cage. This brake is actuated by spring pressure, is constantly in service except when electrically released during normal operation of the hoist and is, therefore, instantly applied in case the current supply is interrupted from any cause. Additional safety devices are provided which take care of overwind, overload on motor, slack cable, etc. Two oil-cushioned buffers are provided at the bottom of the shaft, being so designed as to bring the loaded cage to a gradual stop in case the cage from any cause, while running at normal speed, should not stop at the lower terminal. The cage is equipped with

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a telephone which keeps the attendant in touch with the hoist house, also with an annunciator system covering the several levels.

Each level and the collar are provided with two-story stations that make it possible to load both decks without shifting. The cage accommodates 36 to 40 men besides the attendant. The surface station, which is in the headframe, is connected with the change house by a covered runway. From the underground stations the men are transferred to the various working places in cars accommodating 12 men each and drawn by the haulage locomotives.

The flow-sheet in Fig. 7 should be referred to in connection with the following description of the surface plant of the Inspiration company.

Crushing Plant

The loaded ore skips assume a dumping position at a point about midway between the shaft collars and the center of the sheaves, leaving about 25 ft. for possible overwinding. The sheaves are 125 ft. above the shaft collars. They are 12 ft. in diameter and are pressed on 8½-in. shafts; 8-ft. sheaves on 6-in. shafts are used for the man-elevator ropes. The dumping tracks are of usual design.

Crude-ore Bin

By means of chutes leading away from one compartment of each shaft, an even distribution of the ore is effected throughout the length of the 2,000-ton crude-ore bin. This bin is of rather unusual design and very nearly self-cleaning. The bottom slopes down 45° each way from a hip center, toward the vertical back and front, a section resembling an inverted capital M. Gates are placed under the bin on the slope toward the back as well as on the front of the bin. This arrangement of bottom and gates not only allows the bin to be emptied but the ore is prevented from packing or hanging up by drawing off alternately from the rear and front gates. The gates are 48 in. wide and are operated by rack and pinion from runways.

For each of the four units of the crushing plant two gates, one under the bin and one in front, discharge on an apron feeder that travels across and under the bin, delivering the ore upon a 3-in. bar grizzly feeding a No. 8 gyratory crusher set to crush to 4-in. cubes (Fig. 4).

The apron feeders have a width of 48 in., a length of 25 ft. 6 in. between centers and travel at 7 ft. per minute; 20-in. belt conveyors traveling at 7 ft. per minute and driven from the feeder sprockets are placed beneath the feeders to take care of drippings. These discharge into chutes that join the undersize from the grizzlies. Wood scraps are removed from the ore at the discharge end of the apron feeders and also

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from the inclined conveyors leading to the disk crushers. These scraps are carried away by two 20-in. belt conveyors traveling at 75 ft. per minute, running across the building and dumping outside.

One of the four similar crushing units will be described in the following paragraph. Fig. 4 shows a cross-section of the crushing plant.

Description of Crushing Unit

The discharge from the gyratory and the undersize from the grizzly are conveyed by a 30-in. inclined belt conveyor to two 48-in. disk crushers,

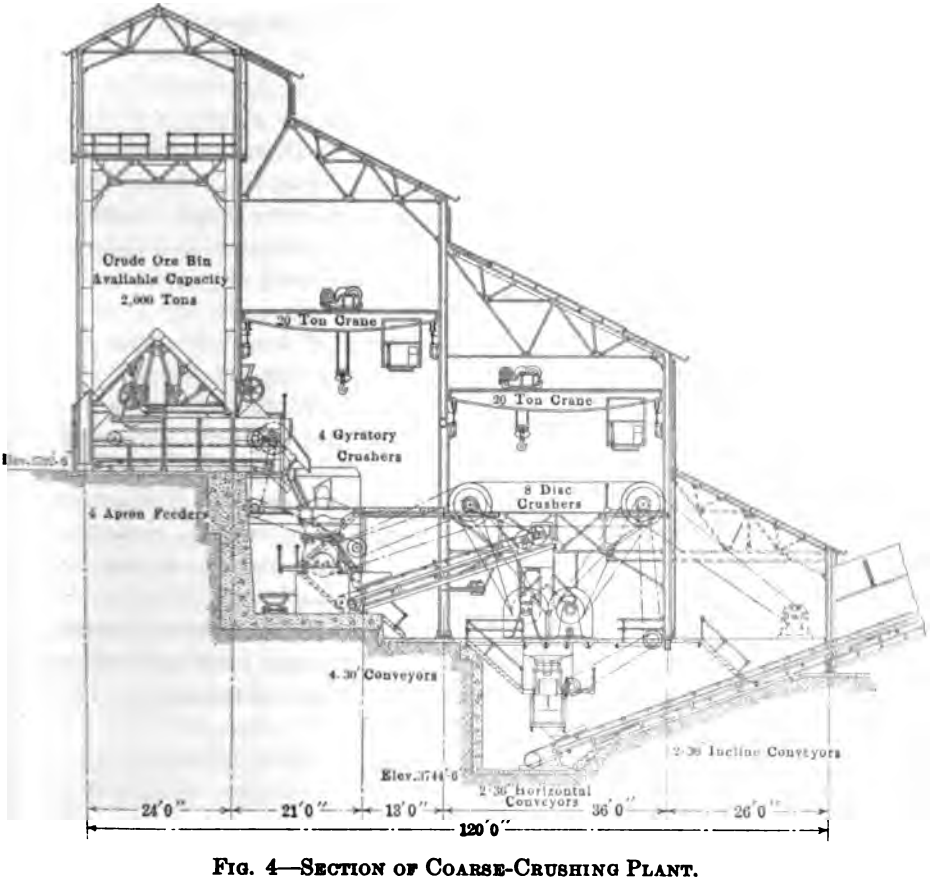

FIG. 4—SECTION OF COARSE-CRUSHING PLANT.

the ore first passing over a 1½-in. bar grizzly. This conveyor has a magnetic head pulley for removing tramp steel. The 48-in. disk crushers are set with a maximum opening of 2 in., their discharge and the undersize from the 1½-in. grizzlies joining on two 36-in. horizontal cross belts which in turn discharge at the center of the building on the two main

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inclined belts, also 36 in. in width. These belts are 300 ft. long, travel at 350 ft. per minute and are equipped with hand-adjusted tail-pulley take-ups and also with weighted tension carriages. They discharge on four 24-in. horizontal conveyors over the storage bin, each of which has an automatic tripper which distributes the ore uniformly throughout the bin. Weightometers are placed on the two main inclined belts, thus giving a record of the ore handled.

Storage Bin

The purpose of the storage bin in connection with a mine plant is identical with that of a flywheel on an engine: "To give out and absorb energy when variation in the load occurs suddenly." It does away with the necessity of maintaining a nice balance between mine and mill and allows either, for a short time, to cease running altogether, or to run at a lessened capacity. Where the mill is removed from the mine plant, it also furnishes an economical means of transferring the ore into railroad cars. Besides this it allows of a hoisting and crushing period considerably less than full-time running, a very important consideration, since there are always repairs to be made in shafts, to loaders, tipples, hoists, and crushing machinery.

The Inspiration storage bin is double tracked, has an available capacity of 25,000 tons and will load a train of 14 cars without switching. The base is of reinforced concrete, and is provided with four expansion joints. The bin proper and conveyor housing are of steel. The main dimensions of the bin are as follows: Overall length 465 ft.; width 40 ft.; depth 40 ft. The bin is divided into three compartments, the two end compartments being for the storage of special ores if any such ever be encountered. They are comparatively small, accommodating one car on each track and having an available capacity of 1,700 tons each.

The distributing belts are driven by two 50-hp. motors located at either end of the bin. The ore is drawn off through hand-operated gates of the swinging cut-off type, there being three gate openings for each car.

Sampling Plant

The heads sampler is placed at the point where the main inclined belts discharge on the horizontal cross belts (Fig. 7). This sampler consists of a rectangular cast-iron bucket, 9 by 30 in., on the end of a revolving arm 10 ft. in length. The bucket runs on a circular track 12 ft. 4 in. in diameter and is driven through a wormwheel mounted on a vertical shaft, making a revolution every 40 sec. The bucket has a swinging cut-off type of gate for a bottom. In its travel it cuts the two main ore streams, taking out $\frac{1}{120}$, and at a point 180° from the ore stream strikes a tripper

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which causes the gate to open and the sample to be discharged into a 10-ton hopper. From the hopper the sample is fed by a roll feeder and started on its way through the sampling plant, which is located against the center of the storage bin opposite the main conveyors. Here the sample is first passed through a 24-in. disk crusher. A $\frac{1}{15}$ cut is then made with a Snyder sampler and this cut is delivered to a set of 27 by 14-in. rolls. The roll product is cut by a second Snyder sampler, this $\frac{1}{15}$ cut going to a mixing drum that equalizes the flow. The resulting stream is delivered to a duplex Vezin sampler making two $\frac{1}{10}$ cuts. These samples are duplicates and weigh 110 lb. each, representing a day's run. The rejects from the three samplers go to an elevator that discharges into the storage bin.

Motors and Other Equipment

The four units of the crushing plant are driven by four 200-hp. motors located in two dustproof rooms, symmetrically arranged on either side of the plant. Except for some minor reductions, belt transmission is used throughout.

Two 20-ton electric cranes are installed in the plant, one to serve the gyratories, the other the disk crushers, motors, etc. These are in turn connected to the yard track by transfer cars and a 20-ton electric trolley hoist, running at right angles outside of the building. Each 200-hp. motor is served by a 6-ton crawl so arranged as to allow of crane handling after leaving the motor room. A repair car and track near the gyratories permits of easy handling of their driving mechanisms, or bottom plates.

Compressor Equipment

Compressed air is used for two distinct purposes: For underground haulage, and for mine drills. For the first, 1,000-lb. air is used and for the second, 100-lb. The haulage system is served by two compressors, each having a capacity of 1,125 cu. ft. of free air per minute at 107 r.p.m. These are of the duplex, four-stage, power-driven type, the cylinders being cross-compounded, water-jacketed, and having intercoolers between the stages. These compressors have an automatic regulating arrangement whereby the 1,000-lb. discharge line is cut out when a predetermined high-pressure point has been reached, and the four cylinders converted into a double-compound compressor, discharging directly into the 100-lb. main. Each compressor is driven by a 430-hp. self-starting, synchronous motor.

The low-pressure equipment consists of two 100-lb. compressors having capacities of 3,000 and 7,270 cu. ft. of free air per minute. Both compressors are of the parallel, two-stage type, all cylinders being water-jacketed with intercoolers between the two stages. The 7,270-ft. machine

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is equipped throughout with Hoerbiger-Roegler plate valves, which on account of the very short movement and exceedingly light construction permit of a high compression efficiency. The cylinders of this machine are 46-in. and 28 by 36-in. stroke. It is driven by a self-starting synchronous motor of 1,150 hp. The 3,000-ft. machine is driven by a 500-hp. self-starting, synchronous motor at 107 r.p.m.

Two 100-kw., 125-volt, motor-generator sets arranged to operate in parallel serve as exciters to the compressor motors. Cooling water for the jackets and intercoolers is obtained from a circulating system consisting of a duplicate installation of 750-gal. triplex pumps and a spray cooling pond.

The compressors and hoisting equipment are housed in a spacious and well-lighted building which has a 25-ton electric crane running its entire length connecting at one end with a well into which railroad cars can be run and unloaded. A 22-panel, remote-control switchboard, together with the low-tension (2,200-volt) bus equipment, occupy a room along one side of the building, the high-tension transformer station being outside near one end of the building. Only two attendants per shift are required, one being an oiler. Without the automatic features for hoisting ore and the elevator for handling men, at least four would be required.

CONCENTRATOR

The considerations which led to the type of plant finally decided upon will first be given; this will be followed by a description of the plant. The concentrator includes the concentrates filter plant. Fig. 5 shows the concentrator with its adjuncts.

600-Ton Test Mill

The underlying principle that pervaded the work in this mill was to carry on all experiments with the idea constantly in mind that the particular method or machine under test might eventually be installed in the new mill.

The test mill was put up primarily to try out flotation on a large scale in order to determine its proper application to Inspiration ores. Since the flotation process, as applied to the concentration of copper ores, was an innovation, it was logical to suppose that in its last analysis the flowsheet might differ altogether from wet concentration methods. This did not prove to be the case, but the assumption was made and it was for this reason that the test mill was made so flexible.

The test mill was started February, 1914. The concentrates sold have paid the expenses of erection and operation, a remarkable record for a test plant. It has treated from 600 to 1,000 tons per day, the amount depending upon the types of machines being tested.

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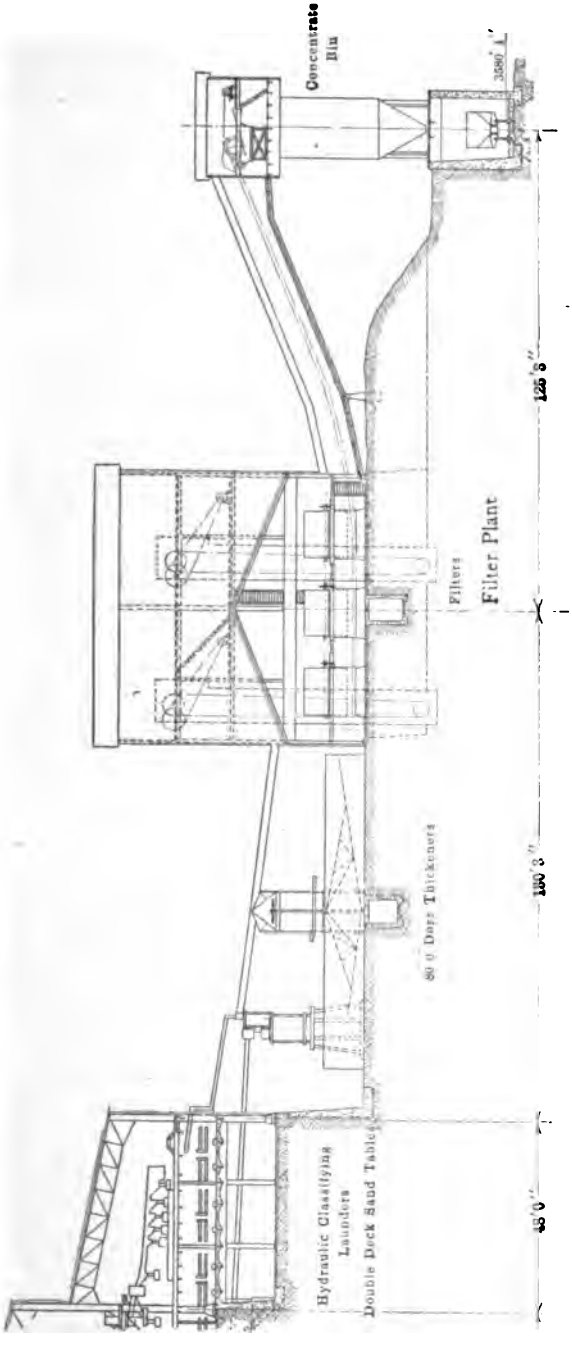

FIG. 5.—SECTION OF CONCENTRATOR AND FILTER PLANT.

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The building is of timber, covered with corrugated steel. This type of building was chosen because it would facilitate changes or additions, was the cheapest and quickest form of construction, and would still have considerable value when dismantled. All machinery was placed on concrete foundations.

The ore for the test mill was taken from a development shaft and reduced to 3-in. cubes in a No. 8 gyratory crusher located nearby, the crushed ore being loaded into cars which emptied into a 180-ton bin at the head of the test mill. The following description gives the way in which the mill was first run, the various additions and changes being noted in sequence.

Two 30-in. apron feeders delivered ore from the bin to an inclined 20-in. belt conveyor, which was equipped with a recording weighing machine and a magnetic head pulley. This conveyor delivered the ore to a 36-in. Symons horizontal disk crusher and a Symons 48-in. fine reduction disk, or vertical disk crusher. These two machines delivered to a second 20-in. inclined conveyor. If so desired the two crushers could be bypassed, the delivery then being made direct to the second conveyor. This permitted the crushers below the second conveyor to be tried either as primary or secondary machines. The second conveyor delivered the ore to another 48-in. fine reduction disk and a Symons 48-in. roller mill. Water could be added either above or below these crushers. The ore then went to four Hardinge mills, the feed to which was regulated within suitable limits by a mechanical distributor. Three of these mills were 8 ft. in diameter with barrels 36 in., 44 in. and 72 in. long, all being direct-driven by induction motors. The fourth mill was 10 ft. in diameter, had a barrel length of 28 in. and was belt-driven. All of these machines were pebble mills with either silex or El Oro linings.

A drag classifier followed each mill, the whole floor being so arranged that each mill could be placed in a closed circuit with its drag or two mills and their drags run in tandem, that is, the first mill discharged into a drag, the sand from which was delivered into the second mill which in turn discharged into the second drag. The sand from the second drag was returned to join the sand from the first drag to form the feed for the second mill. The overflow from the drag was delivered either to eight sand tables by two distributors, or sent to other distributors which fed eight slime tables at the lower end of the mill.

Between the sand and the slime tables were placed the flotation machines which could be fed directly from the drag overflow or by any table product. The original installation consisted of two Minerals Separation eight-compartment flotation machines, one of 50-ton and the other of 600-ton capacity. By means of elevators any flotation product could be retreated on the slime tables or in the small flotation machine.

The concentrates from the tables and flotation machines went to a

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filter plant containing an Oliver and a Trent filter. The dried concentrates were loaded into 1-ton cars, weighed and dumped into a railroad bin from which they were taken to the smelter.

Mechanical samplers were installed at important points and hand samples taken where necessary.

The crushing and grinding experiments were not only extensive but proved most interesting, the results being quite unexpected. The 36-in. disk crusher represented the 48-in. disk crushers now installed in the crushing plant following the gyratories.

As secondary or intermediate crushers, that is, machines between the 36-in. Symons disk and the Hardinge mills the following were tried out: The Symons 48-in. fine reduction disk; the Symons 48-in. roller mill; the Symons 56 by 48-in. ring mill; the Bradley 66-in. centrifugal roller mill; the Overstrom centrifugal crusher; the Allis-Chalmers No. 4 hammer mill; and the Marcy ball mill.

The final results indicated that the Symons fine reduction disk, with a little redesigning, would be an efficient machine for the limited field of crushing from 4-in. cubes to 4-mesh. It was found that this could be done with a single pass, no oversize resulting. The machine would not work well wet and it was found necessary to remove all fines from the feed to the machine, otherwise a packing would occur. Consumption of steel wearing parts appeared low, as also did power consumption, although no definite results as to these two points were obtained.

The Marcy mill also proved to be a most excellent intermediate crusher, handling as high as 800 tons of 3-in. feed and under to pass an 8-mesh screen without return of oversize.

To compete with the four Hardinge mills a 6 by 20-ft. Chalmers & Williams tube mill and an 8 by 5-ft. ball mill were installed. The latter was a Marcy mill using 5-in. diameter steel balls. It was placed in a closed circuit, as were the pebble mills, and gave good results. It was next tried on a feed from the 36-in. disk crusher and then on a feed direct from the gyratory. These experiments definitely demonstrated that only three reductions were necessary: A gyratory, followed by a disk crusher and ball mills in a closed circuit. This is the arrangement installed in the present plant.

Several different classifiers were tried in the ball-mill circuit, a specially designed duplex Dorr classifier being adopted.

The flotation machines and the tests conducted on them are described in another paper,[2] but it is interesting to note here that the following machines were tried out: Minerals Separation, Hoover and Hebbard types; the Callow flotation cells; the Cole-Bergman; and the Inspiration machine.

[2] Rudolph Gahl: History of the Flotation Process at Inspiration, *Bulletin* No. 117, September, 1916.

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The Inspiration machine was devised during the operation of the test mill.

The concentrator is equipped as follows: 4 units with the Callow machines; 13 units with the Inspiration machines; and 1 unit with the Minerals Separation machines.

The drag classifier as originally tried out proved to be too small for the tonnage required, the drag now in the concentrator having been developed from the small one.

The tables tested were the Deister Machine Co.'s double-deck slimers, double-deck sand tables, and four-deck slimer; the Deister Concentrating Co.'s single-deck slimer, double-deck sand table and No. 4 sand machine; and the Wilfley No. 6 table with decks arranged for both sand and slime. The Deister Machine Co.'s double-deck sand table was adopted, 198 being installed.

This sketch of the test mill does not bring out the many small things experimented with, such as shaking screens, Caldecott cones, samplers, feeders, slopes of launders and conveyors, or the necessary efforts to work out improvements on the new machines.

The 15,000-Ton Concentrator

The ore comes from the mine plant in 14-car trains, and is dumped into the concentrator bins, which have a capacity of 12,000 tons, or 670 tons per unit (Figs. 5 and 6). These bins are of the suspension type, are double tracked, and extend the full length of the mill, 300 ft.

The mill is made up of 18 units, similar in equipment except for the differences in the flotation department. Eighteen 30-in. apron feeders, located on the center line of the bin, feed the ore to 20-in. inclined conveyor belts traveling at 150 ft. per minute. Weightometers are installed on these belts, which record the amount of ore fed to each unit. The conveyors discharge into hoppers where a split is made, half of the feed going to the north ball mills and half to the south mills, the transfer being made through launders with cast-iron liners and having a slope of 45° (Fig. 7).

Grinding Section

Floor Arrangement.—One of the novel features relative to the layout of this plant is the arrangement of the grinding floor. With due regard for the individual capabilities of the machines used, it is nevertheless a fact that the arrangement has been important in bringing about the excellent results obtained. The following tabulation will serve to show that the method, besides being efficient in the work it is doing, is conservative of floor space as well:

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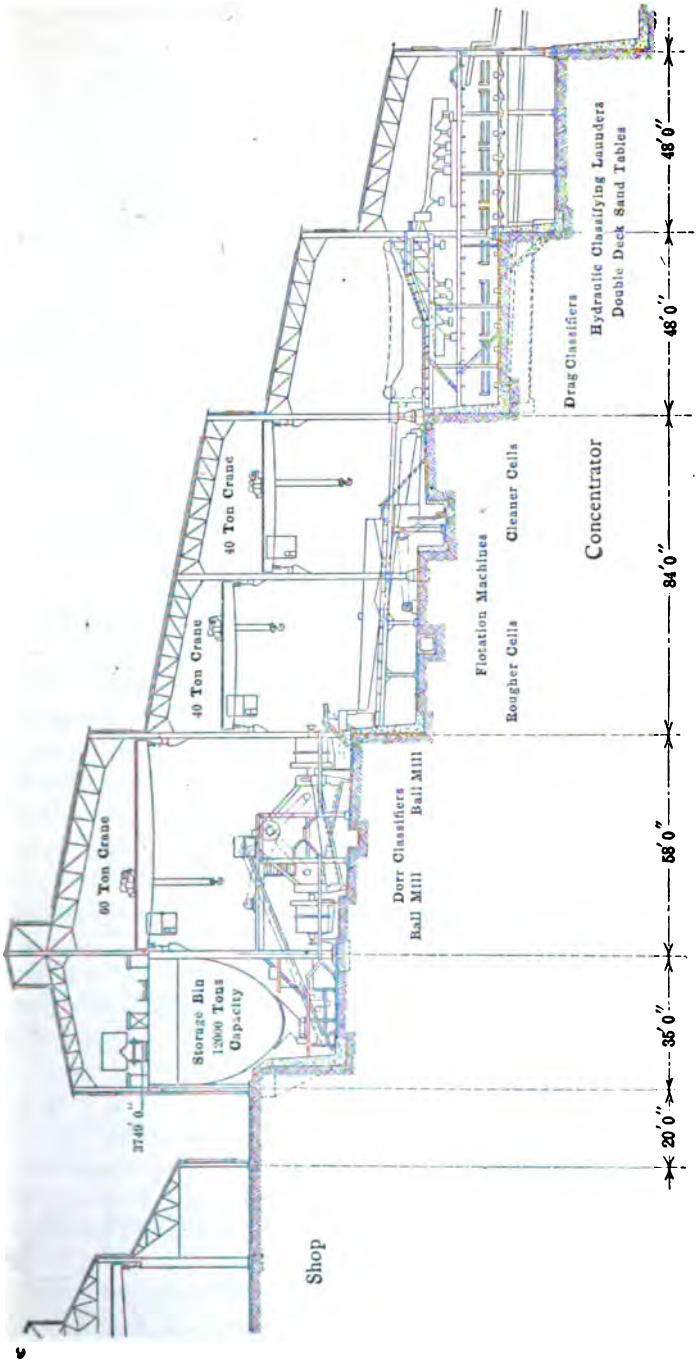

FIG. 6.—SECTION OF CONCENTRATOR.

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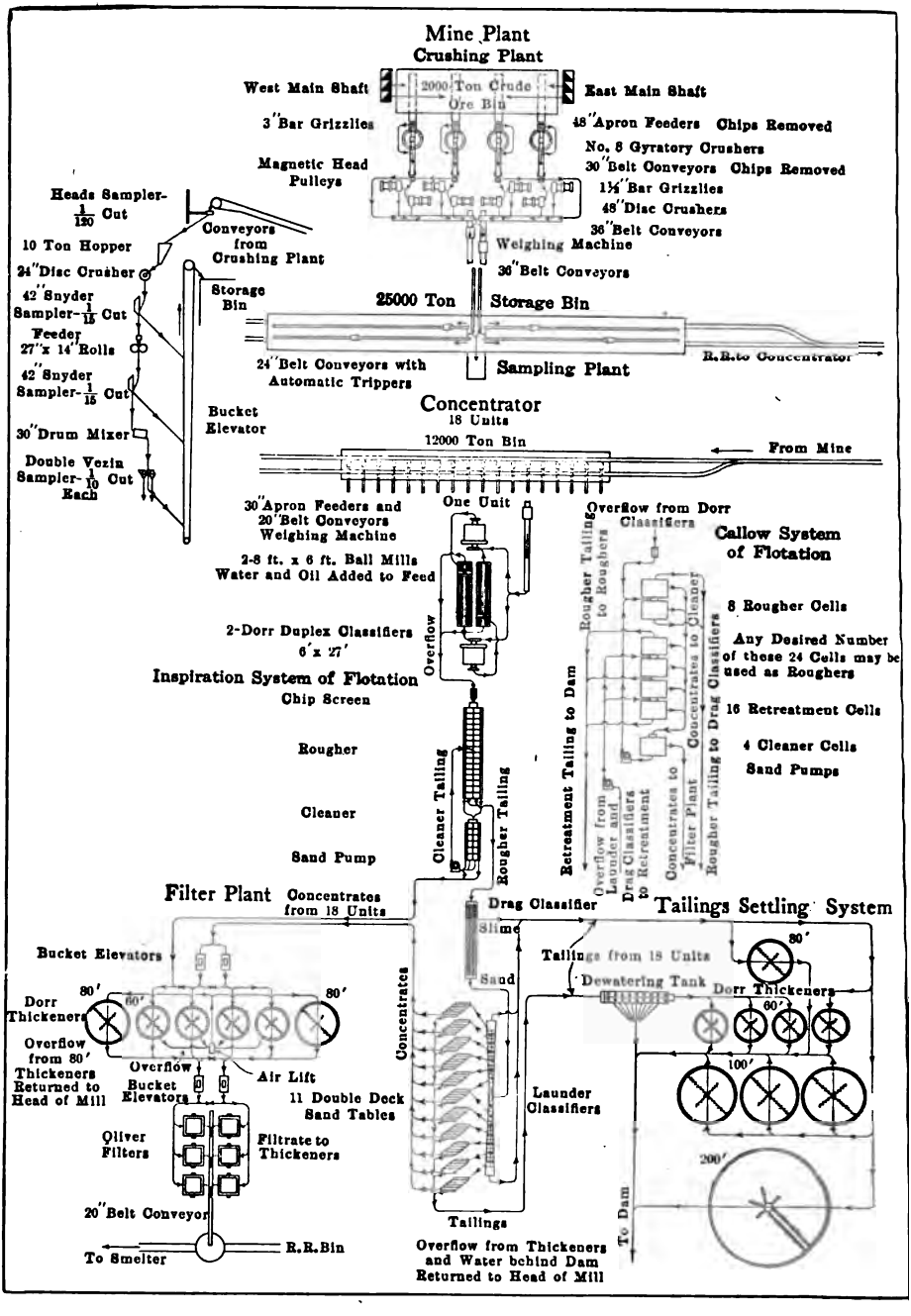

FIG 7.—FLOW SHEET OF INSPIRATION PLANT.

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Capacity of installation, 15,000 tons per day from 2-in. disk crusher opening to 1½ per cent. on 48-mesh.

Area of grinding floor, exclusive of motor platforms, 300 by 66 ft., or 19,800 sq. ft.

Floor space per ton capacity, 1.32 sq. ft.

The scheme can be briefly described as a parallel-series arrangement; two circuits—each with a Marcy ball mill and a Dorr classifier in series—are arranged in parallel, the oversize from each classifier being returned to the mill opposite for regrinding. The mills are symmetrically arranged in two rows extending the length of the mill with their feed ends facing each other, just enough room being left between them to accommodate the Dorr classifiers (Figs. 10 and 11).

The Dorr classifiers act as elevators and since all return is handled by them, the whole department is confined to a single level. There is a main runway between the mills above the classifiers, from which branch runways are taken off to lead to the motors, between mills, etc. This arrangement gives the attendant an excellent opportunity to watch his machines and to reach them when necessary. The conveyors bringing the feed to the mills are terminated in hoppers on a platform high enough above the mills to allow of a 45° slope for the launders leading to the feed boxes. The motors for driving the conveyors, classifiers, etc., together with their control apparatus, are located on this platform, as are also the control panels for the ball-mill motors. The switchboard attendant thus has a commanding view of the whole grinding floor. The floor is served by a 60-ton electric traveling crane which transfers the mills to and from the repair floors located at either end of the building. An inclined skip-way along one side of the building serves the whole concentrator, the cranes on each floor making direct connection with it.

Marcy Ball Mills.—The feed for each ball mill consists of ore from the bin, together with the sand from the discharge of the mill opposite, the final product of the system being the overflow of the classifiers. The feed to the ball mills has a consistency of about 1 to 2, and the overflow from the classifiers about 3 to 1, the latter being maintained by an automatic device which adds water as required in the discharge boxes of the ball mills.

The ball mills are 8 ft. in diameter and 6 ft. long. They are direct-driven through herring-bone gears by 225-hp. induction motors. Our experiments with the ball mill in the test plant led us to the conclusion that we could crush 400 tons of the coarse-crushing plant feed with each mill in 24 hr. to 2 per cent. on 48-mesh screen and 60 per cent. minus 200. Now that the Inspiration mill is running with all sections, it has been shown that our test figures were correct. Not only have we been able to crush 400 tons per mill, but with 18 sections running the average daily dry tonnage for the month of May was 15,358. We have a test

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section (of two mills) that showed for a short period a tonnage rate of 1,000 tons per 24 hr. While this is by no means the average, yet it indicates what may be done in the future.

Performance of Crushing Machines.—In view of the character of the ore and the fact that much of it is tough and more difficult to crush than the ordinary porphyry ores, we consider the power consumption for our crushing plants very satisfactory. The power consumption at the coarse-crushing plant for the month of May, taking mine-run ore, crushing it in gyratories and disk crushers, and conveying to the railroad bins, was 0.329 kw.-hr. per ton. At the mill, the average power consumed per ton during the month of May for the ball mills, including power for feeders, conveyors and classifiers, was 10.4 kw.-hr. Therefore, the total power consumed per ton for the month of May on ore taken from the mine bins, crushed and delivered to the flotation machines was 10.729. We have had one section of the ball mills in operation for a period of 10 days that has given a power consumption of 8.53 kw.-hr. per ton. If we add to this figure the power used at the coarse-crushing plant and of the conveyors, we would have 8.88 kw.-hr. per ton of mine-run feed delivered to the flotation machines. The above does not include the power required for transporting the ore by steam road from the mine to the mill.

The crushing and delivery of the ore for concentration is a matter of great importance, and one that absorbs a large portion of the operating costs in the treatment of ores. Too much attention cannot be given it. We are now conducting many experiments in connection with our crushing problem, including different types of steel for wearing parts and modifications that influence our tonnage and cost of production. We have equipped ball mills without the grates and the results of the mills so equipped have shown us that ball mills equipped with grates have advantages in increased tonnage and efficiency. Our experiments with the different shapes of pebble mills, our subsequent experiments with the ball mills, and finally our monthly operation, lead us to the conclusion that a good choice was made in the selection of our present grinding machinery.

Flotation Section

The overflow from the Dorr classifiers goes to the flotation machines, of which there are three types now in operation. Generally speaking, the flotation machines make a clean concentrate which goes to the filter plant, a middling which goes to drag classifiers and a tailing which goes to waste. As the subject of flotation is to be taken up in detail in another paper, just enough will be given here to bring out the part the process plays in the general scheme. The flow-sheets for the two processes differing somewhat, each will be considered separately.

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Flow-Sheet with Callow Flotation Cells.—A unit installation of these machines consists of a total of 28 cells which for sake of compactness are grouped into sets of four cells each, individual cells being 3 ft. 3½ in. by 10 ft. 2 in. Of this total eight are roughers, 16 are for retreatment and four are cleaners, the arrangement of the group being such that all feeds and products except two are handled by gravity. These two are elevated by sand pumps. The Callow cells, as installed in this plant, make two products, a concentrate and a tailing.

The overflow from the ball-mill classifiers first undergoes a roughing treatment in the eight roughers. Here the two products are a concentrate which goes to the four cleaners and a tailing which goes to a large, specially designed drag classifier, a detailed description of which will be given later. The concentrate from the cleaners is a finished product and goes to the filter plant. The tailing from these cells is pumped back into the rougher cells for a second passage through the system. The tailing from the rougher cells is classified in the drag into two products, a sand and a slime overflow, the sand being still further prepared for table concentration in hydraulic classifiers. The tables make two products, the concentrates joining the flotation concentrates in the filter plant, and the tailing going to waste. Further treatment of a middling product from these tables is contemplated and studies have been made along this line, but as yet nothing definite has been reached. The overflows from both the drag and the hydraulic classifiers are united and pumped back to the 16 re-treatment cells, the concentrate from which goes to the cleaners, and the tailing to waste. Throughout the mill adequate provision has been made for sampling.

Flow-Sheet with the Inspiration Flotation Machines.—Except for a special arrangement in one unit, the only difference between this and the Callow flow-sheet is in the flotation department.

From a metallurgical standpoint the difference here is not radical. In a few words the essential difference can be stated as follows:

The Inspiration system makes use of but two machines per unit, a rougher and a cleaner placed end to end (Fig. 9). These are each designed to allow a free passage of the feed from the upper or feed end, to the tailing or discharge end, during which passage the feed is subjected to the frothing action of several compartments. Because of the large number of compartments (16 for the rougher), it has been found that a tailing can be made, the slime of which does not require re-treatment as is the case in the Callow system. The sand contained in the rougher tailing is subjected to table concentration. The tailing from the cleaners is pumped back to the roughers for a second treatment.

Several mechanical features have been introduced which are expected to keep the cost of operation at a minimum. The arrangement is simple,

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FIG. 8.—BALL MILLS AND FLOTATION MACHINES.

FIG. 9.—INSPIRATION FLOTATION MACHINES.

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requires a minimum of floor space and is easily attended. Its large capacity is an important feature.

Flotation Air Supply.—The flotation department requires a large volume of low-pressure air. This is supplied by four single-stage centrifugal compressors occupying one end of the flotation floor. The rating of these compressors is as follows:

Capacity, 23,000 cu. ft. of inlet air per minute.

Discharge pressure, 5.75 lb. per square inch.

Revolutions per minute of impellers, 3,850.

Horsepower of driving motor, 720.

The step-up to the impellers is made through Alquist gears. One unit of the four is held in reserve.

Concentrates Filter Plant

Both the flotation and table concentrates receive the same treatment. From the concentrator they are either elevated or flow by gravity into the five 60-ft. and three 80-ft. Dorr tanks, located on either side of the elevator house. Here the moisture content is reduced to about 50 per cent. The thickened product is drawn off through the bottoms into launders and conveyed through tunnels to bucket elevators, which raise it to the top of the elevator house where it is distributed to six Oliver filters (Fig. 5).

The overflow from the Dorr tanks is practically clear and flows to the return-water sump. The filters are standard Oliver filters, the drums being 11 ft. 6 in. in diameter and having a length of 12 ft. After leaving the filters the concentrates have a moisture content of about 17 per cent. The arrangement is such that a single 20-in. conveyor belt traveling at 100 ft. per minute takes the concentrates away from the filters and delivers them to a railroad bin of the tank type. Here they are loaded into hopper-bottomed steel cars of 60 tons capacity. A 350-ft. train shed protects the loaded cars from heavy rains and winds.

Drag Classifiers

The drag classifier used on the flotation tailings for separating sand from slime is the result of experimental work in the test mill. It is worthy of a detailed description not alone on account of its ability as a classifier but also because of its large capacity. Except for the concentrates taken out by the flotation cells, each machine handles the entire tonnage of one unit, or 800 tons per day.

The classifier consists essentially of two 18-in. belts, running parallel to each other and so arranged that the return belts run entirely clear of the settling surface, the rough classification being carried on sufficiently

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far below the surface to cause no disturbing currents. This is made possible by the use of four pulleys, one at the discharge end, two (one vertically above the other) at the feed end, and one at the center near the bottom, where the slope begins. The distance between head and tail pulleys is 39 ft. The direction of travel of the return belts is horizontal,

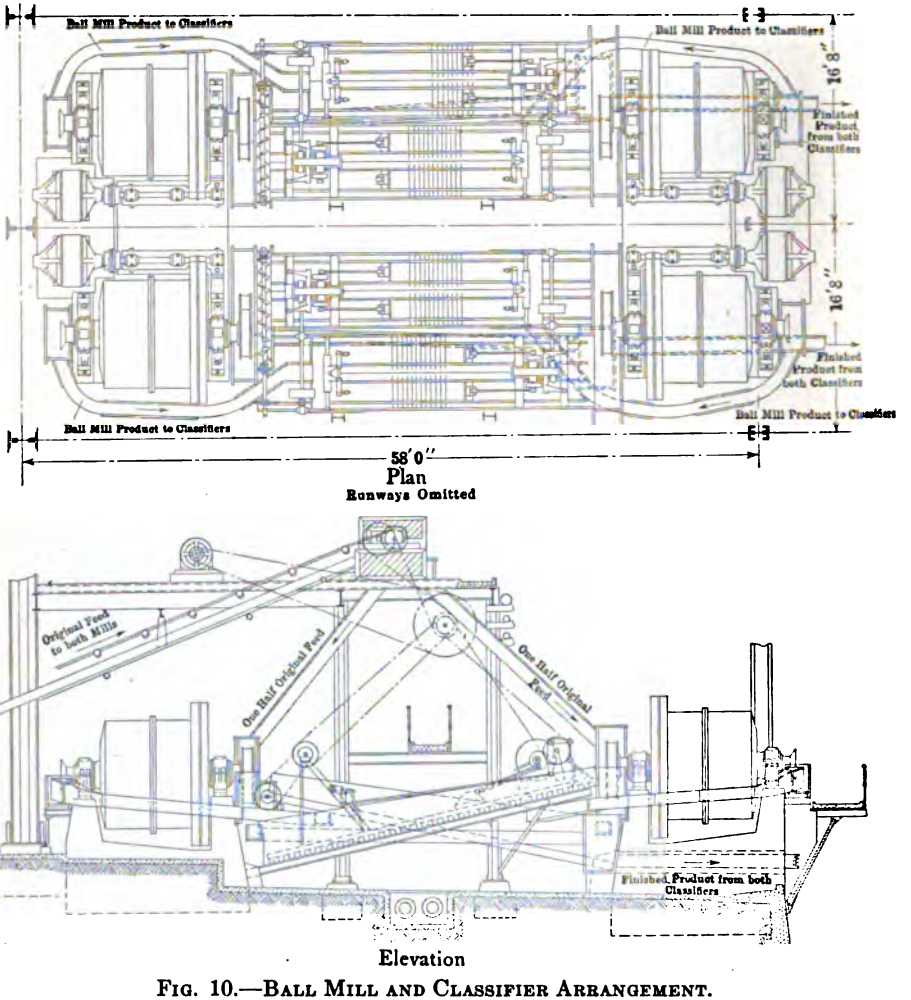

FIG. 10.—BALL MILL AND CLASSIFIER ARRANGEMENT.

but on passing over the upper tail pulley it becomes vertical. The belt thus enters the pulp perpendicular to its surface. At the bottom of the tank its direction is again changed to the horizontal by the lower tail pulley. It travels thus to the center of the tank where a break is made; the belt with its load passes under another pulley and starts up a 20° slope toward the discharge end. All pulleys are 32 in. in diameter.

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Two adjustable overflow launders on either side of the tank having a combined lip length of 52 ft., carry off the slime. The effective settling area of the tank is 25 ft. by 4 ft., or 100 sq. ft. The overflow is 5 ft. above the bottom horizontal belt. The two sets of submerged pulleys are mounted on shafts running in water-tight bearings supported by the sides of the tank.

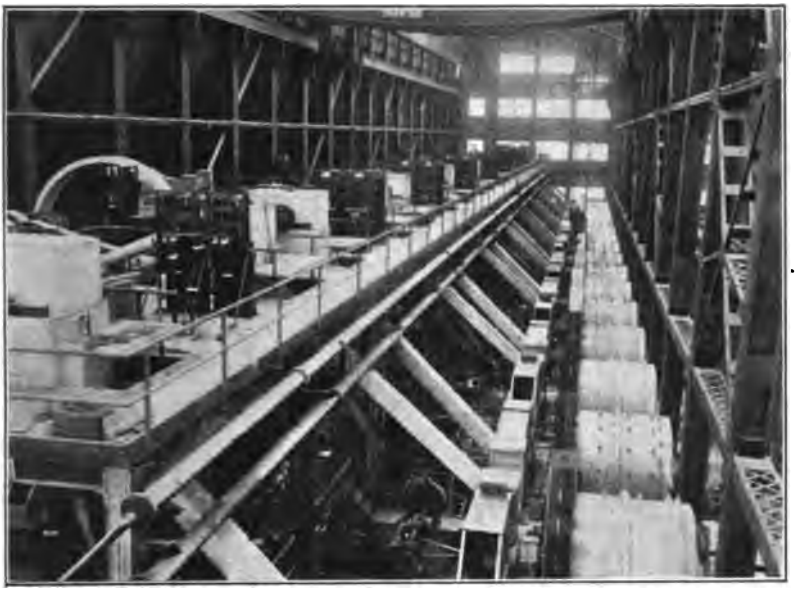

FIG. 11.—BALL MILLS AND MOTOR PLATFORMS.

Tailing Settling System

The conservation of the water supply is imperative, and it was for this reason that the extensive tailing-settling and return-water system was installed. The water is reclaimed at the concentrator and at the tailing dams. The equipment at the concentrator consists of three 100-ft. and one 200-ft. Dorr thickeners, and a large dewatering box 17 ft. by 109 ft. The overflow from the 80-ft. concentrate tanks is also turned into the return system. The dewatering box handles only the table tailing, the overflow from it being further settled in three of the 60-ft. Dorr thickeners. The overflow from all tanks goes to a 210,000-gal. concrete sump from which it is pumped to a 60-ft. diameter steel tank of like capacity on the shop level above the concentrator.

The return-water pump equipment consists of four vertical triplex pumps direct-driven through gearing by 100-hp. synchronous motors. These pumps each have a capacity of 2,000 gal. per minute and work against a head of 113 ft.

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The water reclaimed from behind the dam (Fig. 12) is at present being handled by a 3,000-gal. two-stage centrifugal pump, so arranged that it can be moved up a skidway as the water level of the pond behind the dam rises.

200-*Ft. Dorr Thickener.*—The 200-ft. Dorr thickener is larger than any single unit ever installed and is, therefore, interesting from a mechanical standpoint. The thickener has not yet been erected but at this time the designs are complete. It consists essentially of a reinforced-concrete tank 200 ft. in diameter with the bottom sloping toward the center and having a depth at the outside of 7 ft. 3 in. The feed is delivered at the center of the tank through a launder supported by a 107-ft. steel truss. At the center of the tank is a steel plate pivot, 18 ft. high, upon which is

FIG. 12.—TAILINGS DAM.

mounted the drive drum. To this drum are fastened four short rakes (two 24 ft., and two 35 ft.) and the driving truss, which in turn carries two long rakes, 10 ft. apart and 100 ft. in length. The slope of the bottom of the tank varies, it being $2\frac{3}{16}$ in. per foot near the center, $1\frac{3}{4}$ in. per foot for the short rakes and the inner 16 ft. of the long rakes, and the remainder 1 in. per foot. The driving truss is completely submerged except for the outer end, which rests on a circular track on the tank wall. The driving mechanism is placed at the outer end and consists of a 5-hp. motor which is connected through gearing to a single 24-in. plain-tread driver. Two similar wheels are placed 10 ft. on either side of the driver to distribute the weight. The weight of the driving truss provides the necessary traction. The thickened product is drawn off through an

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annular opening at the center, and the overflow at two diametrically opposite points on the periphery of the tank.

GENERAL EQUIPMENT AND SUPPLIES

Water Supply

Various possibilities were presented for supplying water to this plant. To insure a supply, sufficient ground and water rights were early purchased at Wheatfields, about 12 miles from the mill site and 1,000 ft. lower in elevation. Before developing this supply, however, it was decided to prospect on the flat below the tailing storage site. Wells were sunk at various points until the best location had been determined, which is 2½ miles from the mill, 430 ft. lower and at the junction of two fair-sized drainage channels, receiving their supply from the Pinal Mountains about 10 miles away and 4,000 ft. higher. Here six wells have been sunk and 24-in. multi-stage turbine well pumps installed. These deliver through wooden pipe lines to a common steel sump tank having a capacity of 235,000 gal. Each well pump is belt-driven by a 150-hp. vertical motor and delivers a maximum of 1,200 gal. per minute. Near the sump tank is located the pumping station which contains six 1,200-gal. pumps delivering into a common 20-in. pipe line. These pumps are horizontal, duplex, double-acting and are direct-driven through herringbone gears by 300-hp. synchronous motors taking current at 6,600 volts. Power for the pumping station and wells is supplied by the Inspiration-International power house. A 10-ton crane serves all parts of the building.

The 20-in. pipe line is 14,600 ft. long with a rise of 520 ft., and delivers water to the storage reservoir near the concentrator. From this reservoir water is delivered to all parts of the property. It is located about 80 ft. above the concentrator, 1,200 ft. away and has a capacity of 3,000,000 gal. An oval excavation about 20 ft. deep was made in the top of a hill and the sloping sides and bottom lined with concrete. Two 14-in. pipe lines deliver the water to the concentrator.

Power Supply

Electric power is obtained from two sources. The Reclamation Service of the United States Government, from its hydro-electric plant at the Roosevelt Dam, 40 miles away, furnishes energy at 40,000 volts, 25 cycles, three-phase.

In conjunction with the International Smelting Co., the Inspiration company built a power house that utilizes the waste heat from the reverberatories under one set of boilers. Another set of boilers, oil-fired, is also used when the power demand is greater than can be supplied by the waste-heat boilers alone. This power house contains three 7,500-kva.

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steam-turbine-driven generators, delivering energy at 6,600 volts, 25 cycles, three-phase; also three 15-lb. reciprocating blowing engines, each having a capacity of 15,000 cu. ft. of free air per minute. The air is for the smelter use exclusively, while the electric energy is for the mining, smelting and ore-dressing plants.

The distributing system is so laid out that any division of the plant can take power from either source. When available, the Reclamation power is used for the greater part of the Inspiration load. Each unit of the plant has its transformer station for stepping down the transmission line voltage.

Motors of 50 hp. or greater are run on 2,200 volts, smaller motors on 440 volts, and lighting circuits on 110 volts. Synchronous motors are used as required to maintain a satisfactory power factor.

Transformer Stations

There are two separate outdoor transformer stations, one at the concentrator and one at the mine plant. These stations are of rather unusual design, being skeleton-steel structures. The supports are of pipe poles and the trusses between which the busses are stretched are angle lattice construction.

As the power from the two sources is received at different voltages each station requires two sets of transformers. The Reclamation energy is delivered at 40,000 volts and the power house at 6,600 volts, both being stepped down to 2,200 volts for distribution about the plant.

Both high-tension transmission lines are in duplicate circuits. The incoming lines pass over electrolytic lightning arresters to electrically operated remote-controlled circuit-breakers, then through electrically operated remote-controlled automatic circuit-breakers to the transformers. Any one transformer may be cut in or out as desired, or all at one station can work in parallel. At the concentrator there are eight outdoor-type, 2,000-kva., oil-insulated, water-cooled, three-phase transformers, four on the 40,000-volt line and four on the 6,600-volt line. At the mine there are four similar transformers, two for each incoming line.

The 2,200-volt circuits are led from the transformers to the distributing stations, the one at the mine being located in the compressor and hoist house and the one at the concentrator in a separate building. Each distributing station consists of a concrete cell structure, carrying two sets of 2,200-volt busbars, one for each power line. This cell structure also carries the disconnecting switches, meter transformers and oil switches as required by the remote-control feature of the switchboard. In front of the cells is the switchboard with separate panels for the control of each incoming high-tension line, each set of transformers and each outgoing 2,200-volt circuit. Wattmeters, frequency meters, and power-

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factor meters, all graphic, and integrating watt-hour meters are installed for each power supply, and ammeters, watt-hour meters and graphic wattmeters for each outgoing circuit. Two-throw oil switches, controlled from the switchboard, make it possible to run any circuit on either power, or to disconnect it from both.

The transformer and distributing stations, besides presenting an attractive appearance, are unusually complete, convenient for the attendant and provide the greatest possible safety.

Shops and Railroad Facilities

All shop work, except that for the mine, is done at the concentrator shops where a complete equipment is installed. The equipment comprises about 30 different machine tools, all with individual motor drives, a 40-ton electric crane, traveling the entire length of the building, the necessary facilities for locomotive repairs, several welding outfits for various classes of work and a forge shop equipped with both oil and coal forges. The warehouse and electric shop and the locomotive roundhouse occupy opposite ends of the shop building. A standard-gage track through the center of the building permits cars to be run either into the shop or through the shops into the warehouse.

The shop level is 22 ft. above the upper floor of the concentrator. Communication between the two buildings is provided by the inclined skipway which occupies one end of the concentrator. This connects with the shops by a well through which material can be transferred either into the shops from the mill for repairs, or into the mill from railroad cars.

The railroad facilities comprise the main line and several spurs, the total length being about 10 miles. All equipment is standard gage. The lines connect the several units of the surface plant, except the pumping plant and the Live Oak Division. These are reached via the Arizona Eastern. On account of the topographical conditions, it was necessary to use switchbacks to make the grade between the Arizona Eastern on the flat and the concentrator, about 400 ft. higher. The switchbacks provide for handling six cars. The extremes of grade and curvature on this line are 4 per cent. and 12° respectively. The line which connects the mine and the mill has a grade of 0.3 per cent. in favor of the load.

The six-wheel, side-tank type of locomotive is used. Two are used on the mine line. These have a weight on the drivers when loaded of 112,000 lb. A third locomotive is used on the low line, which has a corresponding weight of 200,000 lb.; the latter being equipped with a superheater and Walschaerts' valve gears, oil burners being used on all. The ore cars are of the Ingoldsby patent-dump type, have a capacity of 60 tons and a train length of 33 ft. They are of all-steel construction and weigh about 22 tons each.

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Electric haulage on the mine division was at one time considered, but for sake of uniformity on the several lines the idea was abandoned.

CONCLUSION

The salient features of the mine plant can be summarized as follows: The dual arrangement of the entire plant; the automatic loading and hoisting of ore; the application of the elevator principle for handling men; and the reduction of the crude ore for mill treatment in two simple stages.

The final product of the plant is ore in storage ready for mill treatment. It carries roughly 1¾ per cent. copper, mainly sulphides, and has been crushed to pass a 2-in. opening of the disk crushers. The average moisture content is 2½ per cent.

To the mine summary can be added the following covering the remainder of the plant: A present mill recovery on sulphides of 90 per cent., or 85 per cent. on the total content; finished grinding in one machine; and flotation followed by table concentration.

Although the Inspiration plant is not quite the largest, it does embody what is today the latest practice in the treatment of low-grade chalcocite ores. In certain departments considerable pioneering has been done, which has resulted in marked economies, advantage of which will no doubt be taken where similar conditions make them applicable.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Ore-Drawing Tests and the Resulting Mining Method of Inspiration Consolidated Copper Co.

BY GEORGE R. LEHMAN, B. A., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

THE Inspiration Consolidated Copper Co. had an orebody at Miami, Ariz., of close to 100,000,000 tons of low-grade copper ore, and the method of mining this ore most profitably was of great importance. The selection of a method was practically limited to a "top-slicing," a "shrinkage-stope" or a "caving" method. The "top-slicing" method would give a high extraction of the developed ore, at a high mining cost, and the "shrinkage-stope" or "caving" methods a lower extraction of the developed ore, at a lower mining cost.

ORE-DRAWING TESTS

To decide on the method to be used it was necessary to know about what extraction of the ore could be obtained by the "shrinkage-stope" or "caving" methods. As no definite information regarding the question was available at the time, C. E. Mills, the general manager, thought that some information could be obtained by experimenting in drawing ore, covered with capping, from an experimental box. Some work had already been done along these lines by W. C. Browning for the Inspiration Copper Co. and by myself for the Live Oak Development Co.

The idea was to represent as nearly as possible, in the experimental box, the conditions within an area of caved stopes of caved ore all ready for drawing. That is, the area was supposed to represent a semibroken mass with the capping above it in the same condition and ready to follow the ore downward as it was drawn out of the chutes under the ore. To represent the above condition, in the experimental box, crushed ore from the mine was placed in it to a given height, and red barren capping from the mine was placed on top of the ore.

The Experimental Box

The experimental box shown in Fig. 1 was made of wood and glass. The glass was used for the sides so that the drawing action of the outside row of chutes could be seen and noted, the latter being cut in the center by the glass sides. The length of the box was 30 in., width 20 in. and the

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height 25 in., and built to a scale of 1 in. = 5 ft., the box represented an area of 150 by 100 ft. and a height of 125 ft. within the mine. The box, as shown in Fig. 1, was divided in the center and only half was used for each test. The bottom was bored with ½-in. holes, representing chutes, 1¼ in. center to center, thus indicating chutes of 2½ ft. diameter spaced on 6¼-ft. centers according to the scale. Thus the ore could be represented as drawn at either 6¼, 8⅞, or 12½-ft. chute intervals. The holes were fitted with wooden plugs through which wire nails were driven, so

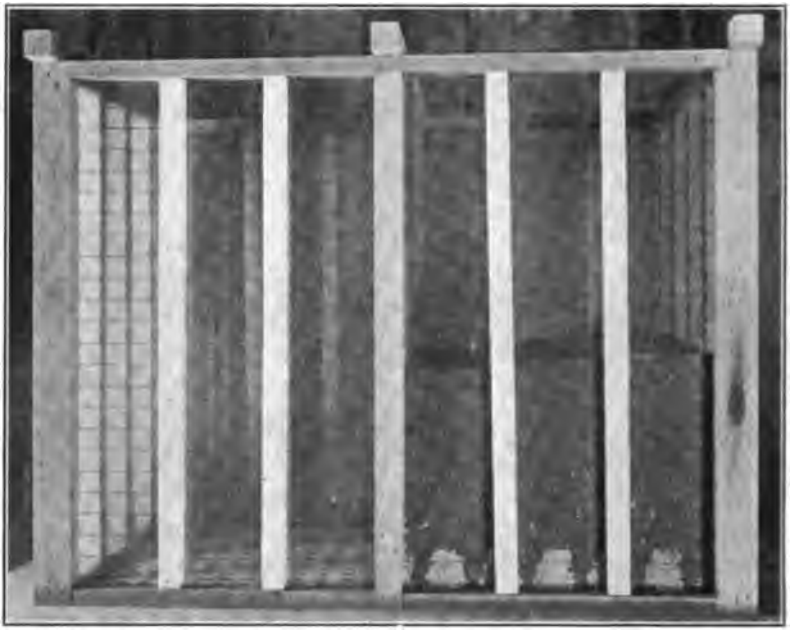

FIG. 1.—EXPERIMENTAL BOX FILLED WITH CRUSHED ORE AND CAPPING; DRAWN THROUGH HOLES IN THE BOTTOM REPRESENTING ORE CHUTES. SCALE, 1 IN. OF BOX =5 FT. OF MINE.

that when the plugs were in place the heads of the nails projected just the least bit above the inner bottom of the box. The nail was used to clear the chute when it "hung up."

Conditions of Tests

In all, 16 tests or experiments were made in a period of 6 months beginning in the latter part of the year 1913. In all the tests the rock representing capping was nearly barren, containing only from 0.03 per cent. to 0.08 per cent. copper. The color of the capping was a dark red and that of the ore a light gray, thus giving a sharp and definite line between them, as clearly seen in Fig. 1.

In the first two experiments, weights and assays were not used. The

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ore in the box and products drawn from the chutes were measured by volume. They were run to get an idea, in a short time, of what could be done. In all the other tests the ore was weighed, sampled and assayed for copper, as were the chute products drawn during the test.

Tests Nos. 3 and 4 were made with fine and coarse ore mixed, the capping being a similar mixture of coarse and fine material. The conditions of test No. 3 represented 35 ft. of ore and 35 ft. of "capping" and those of test No. 4 represented 70 ft. of ore and 55 ft. of capping. Both tests were drawn at the closest chute interval, 6¼ ft. In both tests ore was drawn from one chute equivalent to about 10 tons, on the scale adopted, and then 10 tons from the next chute, and so on until 10 tons had been drawn from each chute. The drawing of all the chutes was then repeated, 10 tons at a time, until capping appeared in each chute. The total chute-drawings product was then weighed, sampled, assayed and called the clean-ore product. Drawing was then continued as before until each chute showed that it was running one-quarter capping. To determine this point, a sample mixture was kept in a test-tube and the chute product compared with it. This product was then weighed, sampled and assayed. Again the drawing continued until each chute showed about one-half capping. This point was also determined by comparing with a sample mixture and this product also weighed, sampled and assayed. The tests were concluded by drawing 10 tons from each chute in rotation, weighing, sampling and assaying the product. The assay value for the combined products of the tests was calculated, as the portions removed for assay could not be returned. Details of test No. 3 are given in Table 1.

TABLE 1.—*Details of Test No.* 3

Weights and assays—35 ft. of ore and 35 ft. of capping.
Ore and capping—coarse and fine mixed.
Largest boulder—1 ft. diameter.
Smallest particle—powder.
Drawn at 6¼-ft. centers—10 tons at one drawing.

Ore in box 127.50 lb.....1.96 per cent. copper
Capping.....0.08 per cent. copper

Products	Pounds	Per Cent.	Per Cent. Cu
(1) Drawn clean to capping.....	81.00	63.52	1.96
(2) Drawn clean to ¼ capping.....	8.50	6.67	1.36
(3) Drawn clean to ½ capping.....	8.50	6.67	1.10
(4) Drawn 10 tons each chute.....	11.00	8.62	0.79
Totals and average.....	109.00	85.48	1.74

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TABLE 1.—*Details of Test No. 3 (Continued)*

Products	Per Cent. Total	Grade, Per Cent. Copper	Product		Per Cent. Total Cu Recovered
			Per Cent. Ore	Per Cent. Capping	
Clean ore.....(1)	63.52	1.96	63.52	.....	63.52
Ore and capping (2)	6.67	1.36	4.54	2.13	4.54
Ore.....(3)	6.67	1.10	3.62	3.05	3.62
	76.86	1.83	71.68	5.18	71.68

Estimate Applied to above Figures to Find Final Product of 1 Per Cent. Copper and Applied to Ores Below

Grade, Per Cent. Copper	Extraction, Per Cent. of Total	Grade of Product, Per Cent. Copper	Per Cent. of Total		Per Cent. of Product	
			Ore	Capping	Ore	Capping
2.00	79.500	1.842	72.96	6.54	91.77	8.23
1.90	78.340	1.763	72.44	5.90	92.47	7.53
1.80	77.180	1.681	71.84	5.34	93.08	6.92
1.70	75.490	1.605	71.07	4.42	94.14	5.86
1.60	73.490	1.529	70.06	3.43	95.33	4.67
1.50	71.490	1.449	68.92	2.57	96.41	3.59
1.40	69.290	1.367	67.54	1.75	97.48	2.52
1.30	67.290	1.283	66.35	0.94	98.61	1.39
1.20	65.290	1.195	65.02	0.27	99.58	0.42
Above Applied to Mining—25 Per Cent. by Development						
2.00	84.625	1.889	79.81	4.81	94.31	5.69
1.90	83.755	1.804	79.41	4.35	94.81	5.19
1.80	82.885	1.717	78.95	3.93	95.26	4.74
1.70	81.617	1.634	78.35	3.26	96.00	4.00
1.60	80.117	1.551	77.59	2.53	96.84	3.16
1.50	78.617	1.465	76.72	1.89	97.59	2.41
1.40	77.117	1.378	75.86	1.26	98.37	1.63
1.30	75.467	1.289	74.80	0.66	99.12	0.88
1.20	73.967	1.197	73.78	0.19	99.74	0.26

Besides estimating the extractions for different grades of ore, based on the experiments, the extractions for different grades of ore applied to mining were calculated, as shown in the lower part of Table 1, assuming 25 per cent. of the ore extracted by development and other work before the ore caves, when mining 35 ft. of ore, and 15 per cent. when mining 70 ft. of ore. For each grade, 1 per cent. copper was taken as the limit of commercial product. This point had to be interpolated from the product just above 1 per cent. and the product just below 1 per cent.

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This way of estimating the results possible in mining a 1.80 per cent. copper ore, according to test No. 3 indicates a total extraction of 82.88 per cent. of 1.72 per cent. ore, 78.95 per cent. of which was original ore, the chute product being 95.26 per cent. ore and 4.74 per cent. capping.

In the next two tests, Nos. 5 and 6, the fines were screened out of the ore and capping, making both ore and capping represent on the scale of 1 in. = 5 ft., coarse chunks of 4 to 12 in. diameter. The chute interval and method of drawing were the same as in tests Nos. 3 and 4. Test No. 5 represented drawing 35 ft. of ore and test No. 6 represented 70 ft. of ore. Compared with tests Nos. 3 and 4, as shown in Table 2, both tests show higher extractions.

Tests Nos. 7 and 8 were duplicates of 5 and 6 except that the chute interval was 8⅞ ft. instead of 6¼ ft. These two tests show lower extractions than Nos. 5 and 6.

Tests Nos. 11 and 10 were the same as Nos. 5, 6, 7 and 8 except that the chute interval was 12½ ft. As will be noted in Table 2, still lower extractions were obtained with these two tests, showing conclusively that the greater the distance between chutes the less the ore extraction.

To determine the effect on extraction if no care is taken as to the amount drawn at one time, two tests, Nos. 12 and 13, were run. They were both made with conditions representing 35 ft. of ore and 35 ft. of capping, drawn at a chute interval of 8⅞ ft. In these two tests each chute was drawn direct to capping in rotation, then direct to one-fourth capping and then direct to one-half capping. Comparing these two tests with No. 7, with which all conditions were identical, except the method of drawing, it was found that in No. 7 a greater extraction of clean ore, before capping appeared, was obtained, but that in Nos. 12 and 13 a greater total extraction was obtained to the limit of 1 per cent. copper, as shown in Table 2.

Tests Nos. 14, 15 and 16 were special experiments made with conditions representing a chute interval of 8⅞ ft. and the column of ore 35 ft. In No. 14 the ore was coarse, 12 to 30 in. diameter, and the capping fine; a low extraction was obtained. Test No. 15 was the reverse of No. 14, the ore was fine and the capping coarse; the extraction was but very little higher than in No. 14. The last test, No. 16, was made with conditions representing both ore and capping fine. This apparently was the worst condition, the extraction being very low.

Conclusions Derived from Experiments

The conclusions derived from the experiments were as follows: (1) The less the distance between chute centers, the greater the extraction; (2) the higher the ore column, the greater the extraction; (3) the less the amount drawn at a time from each chute, uniformly, the greater the extraction of clean ore before the capping appeared, but not the greater the total

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TABLE 2.—*Comparison of Extractions Based on* 1.80 *Per Cent. Copper Ore*

Test No.	Condition of Ore and Capping	Method of Drawing	Distance between Chute Centers, Feet	Depth of Ore, Feet	Extraction, Per Cent. Total	Grade of Product, Per Cent. Copper	Per Cent. Total		Per Cent. Product	
							Ore	Capping	Ore	Capping
3	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	77.180	1.681	71.84	5.34	93.08	6.92
4	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	93.950	1.705	88.76	5.19	94.48	5.52
5	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	86.400	1.717	82.30	4.10	95.26	4.74
6	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	95.030	1.765	93.13	1.90	98.00	2.00
7	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅞	35	78.570	1.717	74.85	3.72	95.26	4.74
8	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅞	70	89.550	1.764	87.71	1.84	97.94	2.06
10	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	70	85.110	1.741	82.24	2.87	96.63	3.37
11	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	35	67.390	1.707	63.81	3.58	94.69	5.31
12	Coarse, 4 to 12 in.	Direct to cap.—¼ cap.—½ cap. each chute.	8⅞	35	84.290	1.708	79.86	4.43	94.74	5.26
13	Coarse, 4 to 12 in.	Direct to cap.—followed by 10 tons each chute.	8⅞	35	83.980	1.703	79.33	4.65	94.46	5.54
14	Ore coarse, 12 to 30 in. Capping fine, 2 in. to powder.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	76.050	1.600	67.36	8.69	88.57	11.43
15	Ore fine, 2 in. to powder. Capping coarse, 12 to 18 in.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	80.040	1.564	69.24	10.80	86.51	13.49
16	Ore and capping, fine.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	53.940	1.635	48.85	5.09	90.57	9.43

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Above Tests Applied to Mining 1.80 Per Cent. Ore—25 Per Cent. by Development for 35 Ft.; 15 Per Cent. for 70 Ft.

3	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	82.885	1.717	78.95	3.93	95.26	4.74
4	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	94.857	1.720	90.52	4.33	95.43	4.57
5	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	89.800	1.740	86.72	3.08	96.57	3.43
6	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	95.775	1.770	94.13	1.65	98.28	1.72
7	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅓	35	83.927	1.742	81.15	2.78	96.69	3.31
8	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅓	70	91.117	1.770	89.55	1.57	98.28	1.72
10	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	70	87.343	1.751	84.90	2.44	97.20	2.80
11	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	35	75.542	1.738	72.87	2.67	96.46	3.54
12	Coarse, 4 to 12 in.	Direct to cap.—¼ cap.—½ cap.—each chute.	8⅓	35	88.217	1.734	84.89	3.33	96.23	3.77
13	Coarse, 4 to 12 in.	Direct to cap followed by 10 tons each chute.	8⅓	35	87.985	1.731	84.52	3.47	96.06	3.94
14	Coarse ore, fine capping.	25 tons each chute to capping—then 15 tons each chute.	8⅓	35	82.037	1.661	75.53	6.51	92.06	7.94
15	Fine ore, coarse capping.	25 tons each chute to capping—then 15 tons each chute.	8⅓	35	85.030	1.633	76.92	8.11	90.46	9.54
16	Ore and capping, fine.	25 tons each chute to capping—then 15 tons each chute.	8⅓	35	65.455	1.698	61.64	3.82	94.17	5.83
		Average			85.38	1.723	81.63	3.75	95.16	4.39

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extraction to a 1 per cent. copper final product; (4) the coarser the ore and capping without fines, the greater the extraction; (5) the higher the grade of ore, the greater the extraction, when a uniform limit of grade is allowed at the end of the chute drawing. (6) Applied to mining, the results of the experiments indicate that the greater the amount extracted by development and undercutting, the greater will be the total extraction. This is on the assumption that when the ore caves it occupies the same volume as in place.

Some suggestions deduced from these experiments considered desirable to carry out in mining were: (1) Chute centers for drawing caved ore should be as near together as the ground will allow; (2) ore should be caved in as high columns as practicable; (3) when capping appears at the chutes, no more should be drawn unless the bottom slice is being mined.

While all of the tests described were made with a barren capping, in the actual mining of a sulphide orebody having mixed or oxidized ore over it in place of barren capping, the recovery of sulphide ore would be considerably increased over the experimental results, provided the mixed or oxidized ore could be treated at a profit.

In deciding what extraction could be expected by caving from 35 ft. to 70 ft. of ore in average caving ground, the results obtained in tests Nos. 10 and 11, as detailed in Table 2, were considered conservative, provided the chutes were closely watched and ore of below the commercial limit not drawn. If the ground under the caved ore, the location for raises and chutes, is hard and strong, results of tests Nos. 7 and 8 could be taken as obtainable. This opinion is based on the supposition that the capping will break up in about the same way as the ore and not into a fine powdery sand.

Method of Mining

The Inspiration company's method of mining is one of the so-called Caving systems whereby the ore is caused to cave and crush itself, thus reducing to a minimum the blasting and handling. It is a modification of the method introduced by Felix McDonald at the Ohio Copper Co.'s mines in Utah, and was put into operation at the Inspiration under his supervision. The method consists, essentially, of undercutting the ore (taking out a horizontal slice), allowing the ore above to cave and crush, and drawing off the crushed ore through small inclined raises driven under the caved ore, into main inclined raises that lead down to the haulage-drift chutes.

Preliminary Development Work

By referring to the accompanying illustrations showing the progressive steps of mine development, a good understanding of the method will be obtained.

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After shafts have been sunk, the haulage drifts are driven under the ore of the section or sections to be mined at intervals of 100 ft. as shown in Fig. 2. These drifts are of large size, being 9 ft. wide at the base of the rails, 7½ ft. wide at the cap, and 7½ ft. high above the rail base. Where timbered, the above refer to inside timber dimensions.

After the haulage drifts are in, or during their driving, "pony" sets are put up every 25 ft. along the drifts. The "pony" set is about 5 ft. high, placed on top of the regular drift set and is the place from which the car loaders operate the chute gates discharging into the haulage cars.

Inclined raises are then started from the "pony" sets at an inclination of from 50° to 54°, depending on whether the sublevels are to be

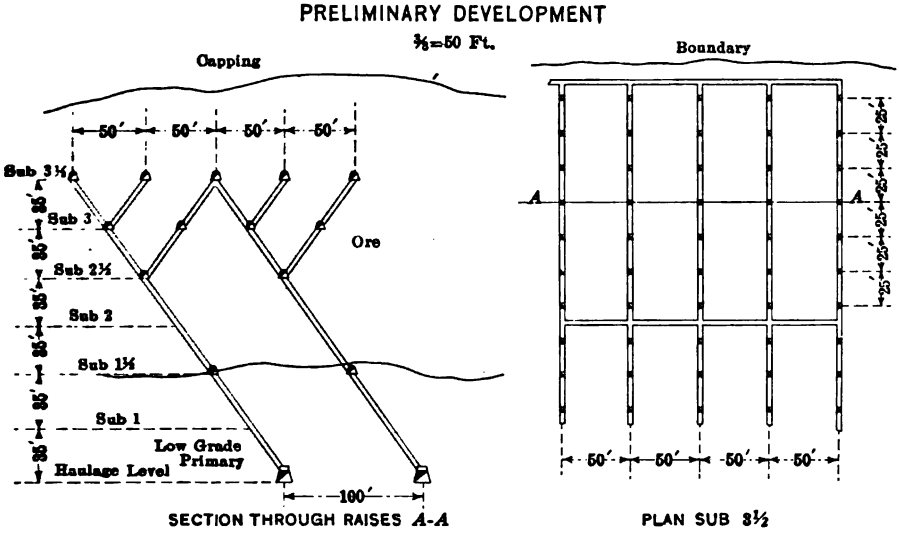

FIG. 2.—PRELIMINARY DEVELOPMENT UNDERTAKEN BY DRIVING INCLINED RAISES FROM HAULAGE-LEVEL DRIFTS.

30 or 35 ft. vertically apart, in a plane at right angles to the haulage drifts. When these raises are up from 10 to 15 ft., chutes are built at the "pony" sets and ore gates of steel are installed.

The raises are then advanced until the first sublevel is reached. "Sub" drifts are then started, from the first raises to reach the required level, and are driven parallel to the haulage drifts. These drifts break into the other raises every 25 ft., as they advance, thus cutting down the cost of handling the drift "muck." The raises are then again advanced to the next sublevel where drifting is to start. The drifts on this level are then driven, meeting the raises as they advance. All of these development raises are about 4 ft. in diameter, and the subdrifts 6 by 7 ft.

This method of raising and sublevel drifting is continued until the

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height is reached where the first undercut is to be made, and the drifts are driven as before.

The first undercut sublevel will be located below the top of the ore, at about the height at which it is intended to cave. When the preliminary development is finished, the sublevel drifts on the first undercutting level are 50 ft. apart, as they are on the "sub" just below, as shown in Fig. 2. On all the "subs" below, the drifts are 100 ft. apart. All the drifts on each "sub" are also connected with inclined raises every 25 ft. Cross drifts, connecting all the drifts on each "sub," are also driven at intervals of about 150 ft.

The next work, before undercutting is commenced, is to drive on the level to be undercut other "sub" drifts, between and parallel to those

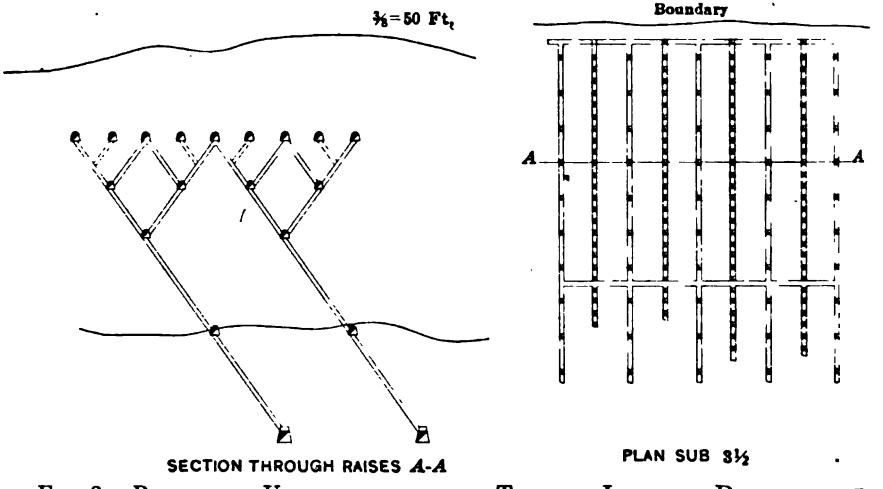

FIG. 3.—PRECEDING UNDERCUTTING, THE TOPMOST LEVEL IS DEVELOPED BY DRIFTS SPACED ON 25-FT. CENTERS AND CONNECTED WITH THE BRANCH, OR "FINGER RAISES."

already driven, as best shown by Fig. 3, making the drifts on this "sub" 25 ft. center to center. These drifts, as they advance, meet branch raises of the main inclined raises at 12½- or 25-ft. intervals as desired. These small branch raises, shown in Fig. 4, are called "finger raises," many of which are put up just before or ahead of the undercutting operations.

Undercutting

Undercutting the ore is accomplished by starting at a cross drift, on the boundary of the section to be mined, and in retreating from that cross drift, as best shown in the plan of Fig. 5, drilling deep holes, at nearly right angles to the drifts, into the pillars between them, and blasting out the ground. Three holes are drilled in each side at different angles in

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the same vertical plane, with one hole in the back of the drifts, thus making seven holes to the round. For this work a water-hammer one-man drill with large steel is used. The holes are from 8 to 10 ft. in depth. Usually the rounds are blasted one at a time, the undercutting receding from the caved ground.

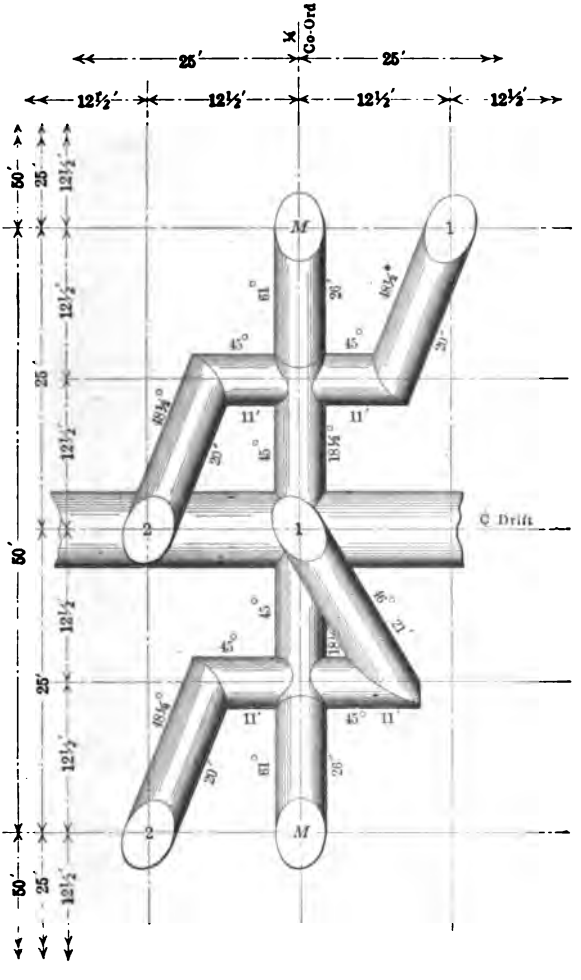

FIG. 4.—"FINGER RAISES", AN IDEAL TOP VIEW SHOWING THEIR CONNECTIONS WITH MAIN RAISES, *M*, AND SUBLEVEL DRIFT.

Ore from the undercut is drawn through the "finger raises," in each of which has been built an ordinary board chute to control the drawing. The "finger-raise" chutes are located about 4 or 5 ft. below the sublevel, so that they will not be blasted in shooting the undercutting holes.

While in some ground, the ore begins to cave as soon as it is undercut, in hard ground caving does not start until the undercutting has receded a considerable distance.

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Ore Drawing

After the ore caves it is drawn off as desired. Large boulders that will not pass the raise chutes are blasted. In drawing off the ore, the chute "tappers" work in pairs. One "tapper" goes up a raise, from the "grizzly sub," which is the first "sub" below the undercutting level, opens a chute gate and draws the ore, while the other "tapper" works the ore through the grizzly on the "sub" below.

All grizzlies have about 1-ft. openings and are made of timber or steel rails. They are placed over all raises on the "grizzly sub" and are set at right angles to the drift. Pieces of ore too large to pass the grizzly are broken with an 8-lb. hammer by the "tapper" tending the grizzly.

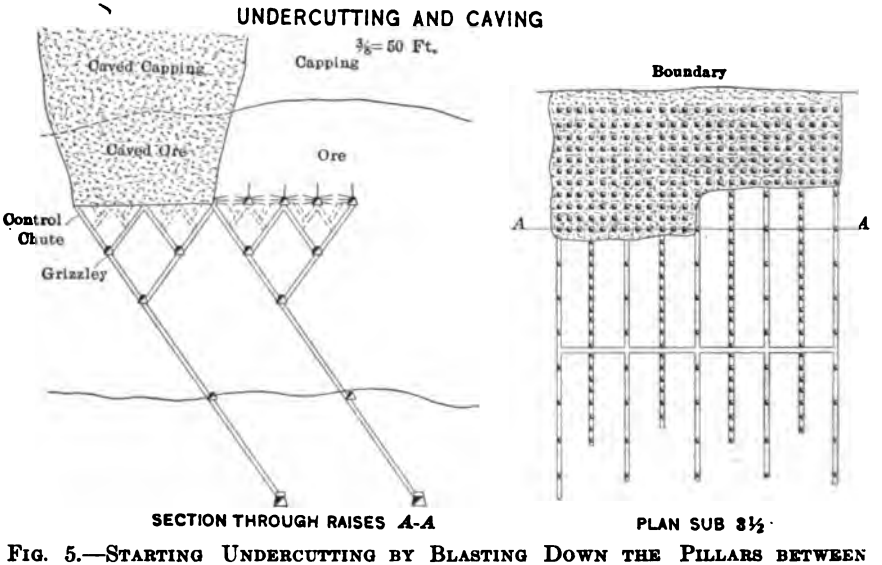

FIG. 5.—STARTING UNDERCUTTING BY BLASTING DOWN THE PILLARS BETWEEN SUBLEVEL DRIFTS.

After passing the grizzlies, the ore falls into the main raises and down to the haulage-drift chutes where it is loaded into 5-ton cars and hauled in trains, of 15 to 20 cars, to the shaft bins.

The second undercut can be located at one, or any number of sublevels below the first undercut, according to the height of ore which it is desired to cave. It could be located at the bottom of the ore if desired. In fact, the first undercut could be located at the bottom of the ore if it were known that the orebody to be mined could be caved throughout its total height.

Underground Haulage and Hoisting Arrangements

At the Inspiration mine the ore trains are hauled by compressed-air locomotives built by the H. K. Porter Co. of Pittsburgh. Arrived at the

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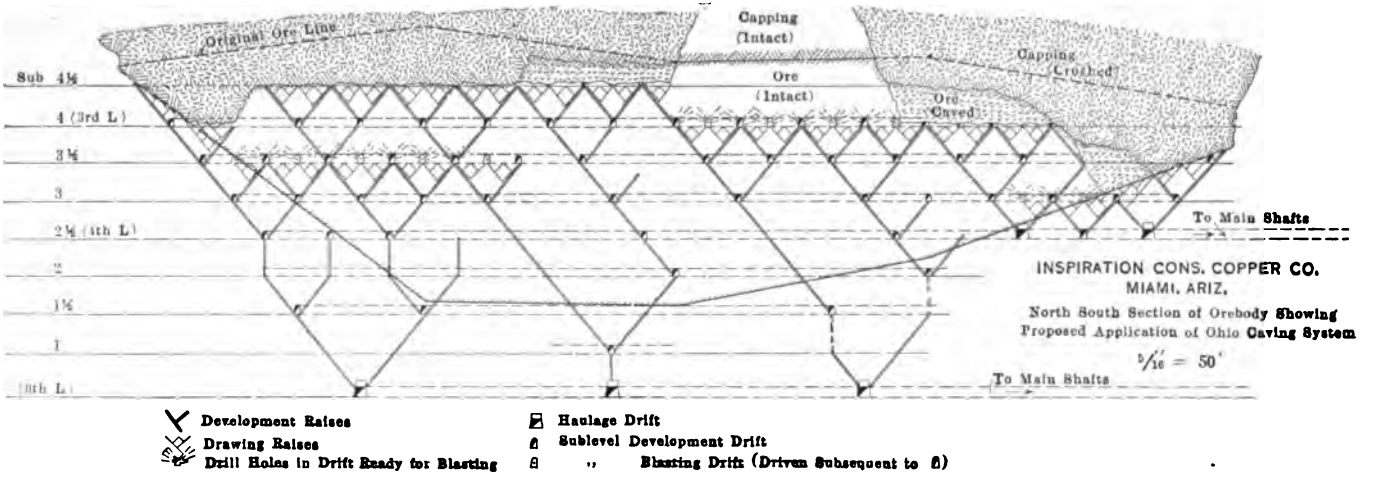

FIG. 6.—ARRANGEMENT OF MINE LEVELS AND RAISES UTILIZED IN INSPIRATION CAVING SYSTEM.

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shaft, the cars are tippled into the shaft bins by barrel-shaped tipples, five cars at a time. From the shaft bins the ore is loaded automatically into 12½-ton skips and hoisted by electrically driven automatic hoists to the steel bins on the surface.

For hoisting ore, two main shafts 101 ft. apart are used. Both shafts have three compartments, of which two in each shaft are skipways. The third compartment in one shaft contains the ladderway, pipes and electric conduits, while in the other shaft the third compartment contains an Otis double-deck elevator for hoisting and lowering men. Both shafts, the underground bins, and the stations are all lined with reinforced concrete.

Only two levels are used for hauling ore, the 4th and 6th. The vertical distance between these two levels is 130 ft. A good idea of the arrangement of the mine is obtained from Fig. 6. On the 6th level the shafts are connected by two drifts, one for each shaft. In each drift there is a tipple over the bin at the shaft. The arrangement of the drifts, bins, tipples and stations is symmetrical with a center line between the two shafts so that the tipples are operated by one man from a central location.

On the 4th level one double-track drift passes between the two shafts and is connected to the shaft stations by a small cross drift. The ore is dumped by one tipple, on this level, into a small bin which is connected with the 6th level bins by a concrete-lined inclined raise. This arrangement makes necessary only one loading level for the skips.

The Inspiration Consolidated Copper Co. commenced underground development on a large scale toward the latter part of 1913 but did not start regular mining operations until August, 1915. The tonnage mined gradually increased as the concentrator was able to handle it until at present (June, 1916) an average tonnage of 16,700 tons is being mined daily.

The cost of mining is now 60 c. per ton, including 20 c. for development and all fixed charges.

The percentage of ore and copper extraction obtained by the Inspiration method cannot be determined positively until some section has been entirely mined from top to bottom of the ore. But based on results in six sections of the mine from the first undercut to the capping estimated to contain 1,886,450 tons, and which have been almost completely drawn to capping, the ore extraction is 102.44 per cent. and the recovery of copper 86.52 per cent. The second undercut is expected to increase the copper recovery.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Block Method of Top Slicing of the Miami Copper Co.

BY E. G. DEANE,* B. S., E. M., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

A METHOD of top slicing has been devised at the Miami Copper Co.'s mine at Miami, Ariz., which differs radically in some ways from the customary methods of top slicing.

The area of that section of the orebody in which top slicing is used is about 800 ft. square. The ore, while for the most part soft, is, nevertheless, considerably harder than the capping. The latter is siliceous, seldom containing any clay or other binding material, and breaks into fine particles so that it runs like sand if given the opportunity. Because of these facts, and because the ore is above the average grade of the mine ore, it has been mined by top slicing.

Haulage levels are opened up 150 ft. apart, vertically, with two sublevels between at 50-ft. intervals, to facilitate the building of ore chutes. These sublevels are used during slicing for distributing air in the ventilation system. On the haulage level the drifts are spaced on 50-ft. centers, and raises along these drifts are also spaced on 50-ft. centers, except the incline raises as hereafter noted. The raises are cribbed where necessary. Where the wear will be excessive, ¼-in. iron plates are spiked to the top of every third set of cribbing, for its protection.

When top slicing was first used, an attempt was made to carry a slicing face from fifty to several hundred feet long. Timber and other supplies were brought in through long drifts from an auxiliary shaft. Great difficulty was experienced in keeping these drifts open, the side pressure breaking the posts and the top weight breaking both caps and posts. Furthermore, the men could not work efficiently while these drifts were being repaired. The slicing faces advanced irregularly, due to varying conditions, and in many ways the results were not all that could be desired. It was then decided to divide the slicing area into blocks 200 ft. square and this was later changed to 250 ft. square.

At the center of each block a two-compartment raise is put up as a supply raise, the compartment being 2 ft. 6 in. and 4 ft. by 4 ft. 4 in., the smaller being used as a manway. Station sets of 12 by 12 timber with

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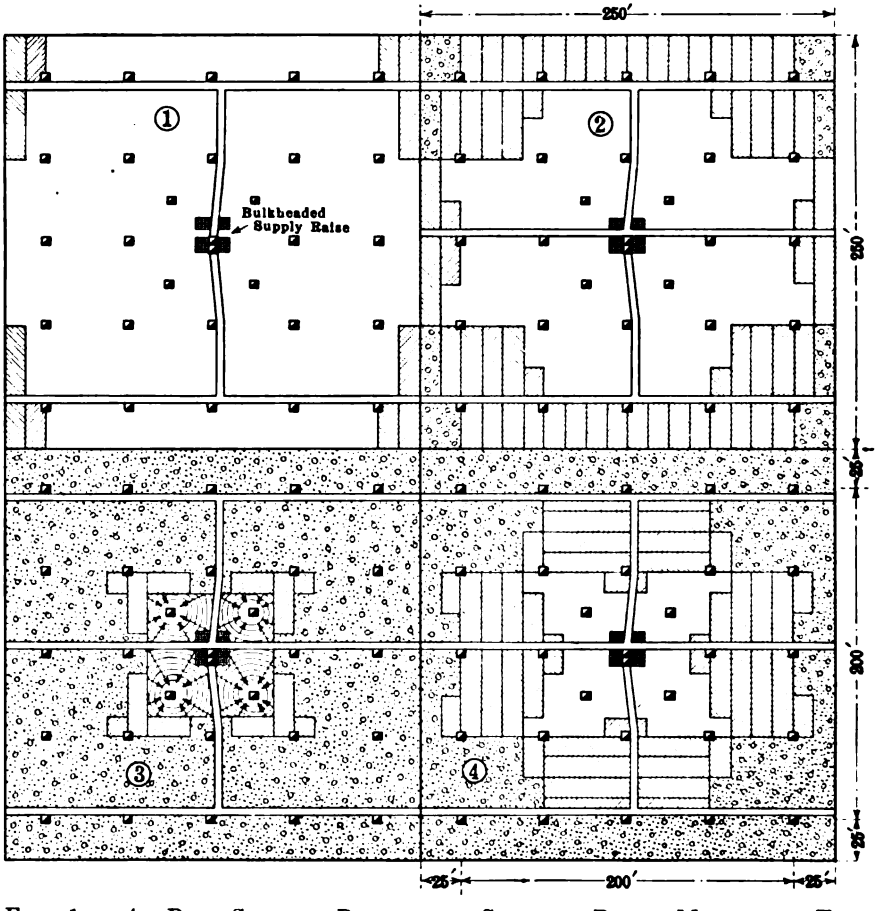
9- or 10-ft. posts are put in and an Ingersoll stretcher bar air hoist is mounted above the larger raise compartment to use in hoisting the timber, steel, etc. Four bulkheads, built solidly of blocks of square timber, are put in as shown in Fig. 1. Two of these are 7 by 11 ft. in size, and the other two 7 by 7 ft.

Two drifts, usually untimbered at first, are run out 100 ft. on the long axis of the supply raise. At the end of each of these and at right angles



FIGS. 1 TO 4.—PLAN SHOWING PROGRESSIVE STEPS IN BLOCK METHOD OF TOP-SLICING.

to them, two drifts are run 125 ft. to the block limits. When these limits are reached, slices are started toward the corners of the block. These slices are timbered either with single sets consisting of two 8-ft. posts and a 12-ft. cap, or with a double set consisting of three 8-ft. posts and two 7-ft. caps, depending upon the ground. The ore is taken to the floor above.

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As soon as the first slices have advanced a few feet, second and third slices are started, and also first slices toward the centers of the block limits. Every man who can work to advantage is put on and the ore mined with the greatest possible speed.

Where there is a sufficient mat of old timbers in the back to obviate the danger of the capping running into the ore there is no permanent floor laid, the planks used in shoveling and wheeling the ore being taken up later. Where there is no mat or it is not sufficient, a floor of 2-in. plank spiked to 2 by 10 sills is laid. Formerly 5 by 10 and 4 by 8 sills were used, but it was found that after being subjected to the pressure and heat of a completed stope, a 5 by 10 or 4 by 8 sill seemed to have practically no more strength than a 2 by 10.

As soon as the timbers in the slices show signs of taking weight, bulkheads are built of old timbers, obtained either from the mat in the back, or from repair work in other parts of the mine. As soon as possible, the posts are drilled and the slices shot down.

By the time slicing has started, four drifts have been run to the centers of the sides of the blocks as shown in Fig. 2 and slicing is also done from these. Working as intensively as possible, all ore except the four central pillars is quickly mined out as shown in the series of illustrations. As about the last of this ore is being taken, crosscuts are driven to incline raises, put up to about the center of the four central pillars, as shown in Fig. 3, and slicing continues, working from the outside of the remaining ore to the supply-raise bulkheads first put up as shown in Fig. 1, thus completing the stope. By this time, these bulkheads, which were 10 ft. high when put in, have squeezed to from 4 to 6 ft. in height. Upon completion, the stope is shot down and another may then be started below, though it is best to let the ground settle for a few weeks.

At first thought the criticism suggests itself that with such weight the mining method used intensifies this weight as the block approaches completion. But experience has shown that, as a rule, the maximum weight is taken by the timbers at about the time the outside pillars are completed, and as mining progresses a larger proportion of the weight is taken by the bulkheads in the outside slices. At no time does the weight on the remaining ore and the slices still necessarily open get beyond control. These central pillars constitute our cheapest ore, not only because of the pillar raises but because the ground has been fractured by the weight, and lifters are the only holes necessary to break the face.

All drilling is done with plugger machines, using a water spray attached to a 5-gal. can. No cars are used in the slices, all ore being shoveled directly into the chutes or wheeled in barrows. Round timber is used almost exclusively in the stopes, because of its superior strength.

Ten feet has been taken as the standard height of a slice. If a greater

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height is taken the ore sloughs off the top of the slicing face faster than it can be mucked out and the bottom shot, thus shortly caving the slice. It is possible that later, when a good mat has been formed, sublevel caving may be used, but so far, where tried, it has not been successful.

The ventilation of these slice blocks is very important because, without it, the heat coming out of the mat is excessive and prevents efficient work. It is accomplished by connecting one of the sublevels below the slicing floor with the discharge end of a 60,000-cu. ft. fan. Openings are maintained from the sublevel to raises through which it is desired to force air.

As a result of the change to the block method of slicing, and to using forced ventilation, the production per mucker-shift has been raised from 9 to 20 tons, the production per man-shift from 5 to 10 tons.

While the details of this method have been fully worked out and a large tonnage of ore has already been extracted by its use, the work to date has to a large extent been preparatory to systematic work for lower lifts. It is not possible to give representative costs, but the following is an estimate of what is expected:

Mining Costs per Ton

Preliminary development	$0.035	
Haulage development	0.025	
Other development (raises, etc.)	0.080	
Total development		$0.140
Stope Costs:		
Miners, at $3.75	0.080	
Muckers, at $3.75	0.160	
Drills	0.030	
Explosives	0.040	
Timbering, labor	0.070	
Timbering, supplies	0.130	
General—bosses, nippers, etc.	0.040	
Total stoping		0.550
Haulage		0.055
Hoisting		0.04
Pumping		0.005
General underground		0.025
Ventilation		0.015
Engineering and sampling		0.016
Underground lighting		0.004
Mine surface		0.030
Total		$0.880

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Cost and Extraction in the Selection of a Mining Method

BY C. E. ARNOLD, B. S., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

In attacking the problems of mining and treating large disseminated copper orebodies such as those occurring in the Miami or the Ray district of Arizona, one of the vital questions to be decided is, "What is the value of the orebody per ton in place?" The importance of this question lies in the fact that its answer helps to reach a decision concerning the method of mining to be employed.

Let it be assumed for the moment that milling and smelting practice, especially the latter, is more or less standardized, and has a cost incapable of much variation, while the cost of mining has a considerable range owing to the large number of mining methods by which a mine may be exploited. For instance, a method may be adopted whereby all the available ore can be extracted perfectly clean (*i.e.*, unmixed with barren material) at a relatively high cost; or a method may be used involving the necessary dilution of a portion of the ore with waste, in order to recover that portion, thus giving a relatively poorer extraction but nevertheless a lower working cost than the former method. Naturally, between these two extremes there lie intermediate methods.

In order to arrive at a solution of a mining method problem that is of any value, it is necessary that accurate and complete estimates of mining, milling, and smelting performances and costs shall be established. By way of illustration is given a case of mining, by the system of undercutting and caving, a large orebody with an average content of 1.50 per cent. copper supposedly all in sulphide form, to which case has been applied those results in mining operations which one may fairly assume are to be expected when viewed in the light of G. R. Lehman's ore-drawing experiments, which form the subject of a paper to be presented at this meeting.

It is assumed that the total costs will be as shown in the following table:

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Costs on 1 Ton of Clean Ore

Mining, including prepaid development	$0.600
Coarse crushing	0.024
Freight on ore	0.015
Concentrating	0.462
Freight on concentrate	0.002
Smelting, freight on copper, refining, marketing, etc	0.737
Amortization	0.120
Total	$1.960

To attain these costs, a daily production and treatment of 14,400 tons has been assumed. Other assumptions are as follows: 15 per cent. of orebody is to be extracted clean by development work and undercutting; 63.8 per cent. (75 per cent. of 85 per cent.) is to be extracted clean by caving, leaving 21.2 per cent. part of which is to be extracted by diluting with waste, and the remainder lost. Dilution and extraction are to be related as shown by the curve in Fig. 1. The copper tenor of the concentrate is to vary uniformly between 25.33 per cent. and 30.00 per cent. on 0.80 per cent. and 1.50 per cent. ore respectively. Mill extraction is to vary uniformly between 69 per cent. and 83 per cent. on 0.80 per cent.

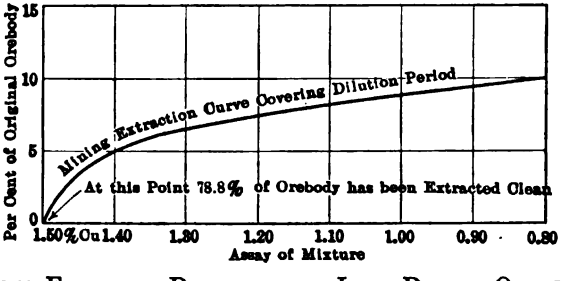

FIG. 1.—SHOWING EFFECT OF DILUTING THE LAST DRAWN ORE WITH CAPPING UPON COPPER ASSAY OF THE MIXTURE.

and 1.50 per cent. ore respectively. The selling price of copper is to average 14 c. per pound. The life of property will be 12 years. Smelting is to be performed under contract with an outside company.

In the above set of assumed costs the charge of $0.12 per ton of ore for amortization represents the outlay necessary to redeem in 12 years all expenditures for plant (not including smelter), lands, railroad, water supply, etc., other than the expenditures required in preparing the mine for the assumed daily production. This latter expense is covered by a charge of $0.20 per ton, which is included in the estimate of $0.60 per ton for mining. The amortization fund is considered as an annuity invested at 4 per cent.

It should be borne in mind that as long as the ore is undiluted with waste, the above total cost of $1.96 is all that is applied to the ton in place, and that the instant dilution starts the cost per ton in place increases because the added waste, until it disappears in the mill tailing, goes through all the operations to which the ore is subjected. Further-

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more, when the ore is diluted, the mill extraction, and consequently the smelter return, decreases, likewise the copper tenor of the concentrate decreases. These disadvantages, however, are to a certain extent offset by the fact that on dilution, the increasing concentration ratio and the decreasing copper content of the concentrate lower the smelting cost on the ton of ore in place.

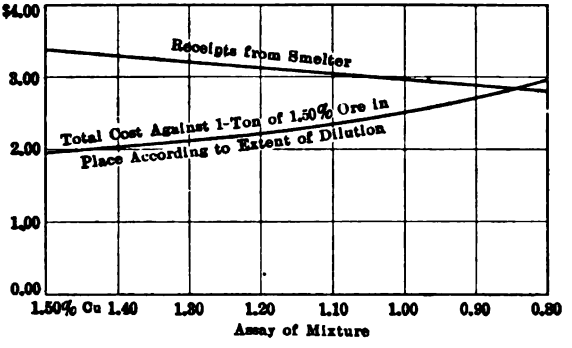

FIG. 2.—SHOWING DECREASING SMELTER RECEIPTS AND RISING COSTS WITH INCREASED DILUTION.

Taking into account all these variables, the curves shown in Fig. 2 were plotted, and then by using these in conjunction with the curve of Fig. 1, the curve of Fig. 3 was established, showing the extent to which dilution could be carried to secure the maximum profit from that ore remaining in place after the extraction of all available clean ore.

The conclusions resulting from this set of calculations are as follows:

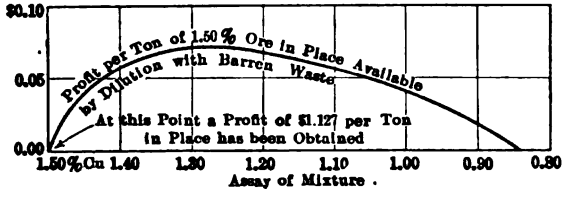

FIG. 3.—SHOWING MAXIMUM POSSIBLE PROFIT WITH ORE DILUTION INCREASED UNTIL ORE ASSAY IS REDUCED TO 1.28 PER CENT. OF COPPER.

1. To obtain a maximum profit the average assay of the dilution product will be 1.28 per cent. copper, the end point being 0.845 per cent., as shown in Fig. 3, at which point receipts from sales of copper just balance total costs.

2. The total ore recovered will be 85.5 per cent., as shown in Fig. 1, of which 78.8 per cent. will be extracted clean and 6.7 per cent. will be extracted in conjunction with 1.15 per cent. of waste.

3. From 100 tons of ore in place there should be recovered:

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78.8 tons clean, at a profit of $1.430	$112.68
6.7 tons diluted, at a profit of 1.063	7.12
85.5 tons at a total profit of	$119.80

Expressed otherwise, the above might be stated thus:

78.8 tons clean ore at a profit of $1.430	$112.68
6.7 tons clean ore at a profit of 1.273	8.53
Gross profit	$121.21
1.152 tons waste at a loss of 1.221	1.41
Net profit	$119.80

This is equivalent to $1.20 per ton in place.

To duplicate this performance by the employment of a mining system that would give, say a 95 per cent. extraction of clean ore (this representing the end of mining operations), a mining cost increased from $0.60 up to $0.77 per ton would have to be borne as indicated below:

	Per Ton
Receipts from smelter	$3.39
Costs other than mining	1.36
Difference	2.03
Cost of mining	0.77
Difference	1.26
Mining extraction, per cent	95
Profit per ton in place	$1.20

Further, it may be stated that on some particular ore, of higher copper content than 1.50 per cent., it would be equally profitable to apply either a low-cost, low-extraction mining method, or a high-cost, high-extraction method; and with further increase in the copper content of the ore the high-cost, high-extraction method would yield the greatest profit per ton in place.

By the application of similar reasoning to various mining problems such as that outlined above, it will be found that the resulting figures indicate plainly the mining method that will yield the greatest ultimate profit. Should the calculations show that there is little difference in the results of several methods, the logical decision would be to use that method in which the element of doubt is smallest regarding the realization of the estimated costs.

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Shaft Sinking through Soft Material

BY EDWARD A. SAYRE, E. M.,* DES MOINES, IOWA

(Arizona Meeting, September, 1916)

IN shaft sinking for coal mines, the cost item greatly influences the method adopted. This holds true especially when soft material must be traversed. The average life of a coal mine is short. This is due either to the limited area of the coal basin or to the great expense of maintenance and haulage underground when the entries are extended a considerable distance from the hoisting shaft. Therefore, the engineer in opening a coal mine is confronted with the fact that the cost must be kept within certain bounds else the venture will prove unprofitable. For this reason he may be prohibited from adopting certain safe, but expensive, methods that are used in sinking caissons for foundations, driving transportation tunnels, or other work of permanent construction.

A common method of sinking through difficult ground employs the aid of a steel shoe pushed ahead of the shaft timbers. Another is the drop-shaft method. These two methods were used in sinking the main shaft and the air shaft of the Eagle No. 3 mine, at Des Moines, Ia., and the following data show the relative success and cost of the two methods of sinking under the same conditions.

A drill log of the material to be penetrated was as follows (see Plate 1, Fig. 1):

	Thickness, Ft.	Thickness, In.	Total Depth, Ft.	Total Depth, In.
Drift	38		38	
Sand	2		40	
Drift	2	6	42	6
Jointed clay	5	6	48	
Drift, firm	10	6	58	6
Clay, with sand streaks	3		61	6
Sand	12		73	6
Shale, with two thin beds of coal and thin strata of rock	86	6	160	
Rock	5		165	
Coal	3	9	168	9

Main Shaft Sunk by Steel-Shoe Method

This record shows the first 73 ft. to be of drift material, the lower 12 or 15 ft. of which was sand. The remaining 92 ft. to the coal was shale and rock, and offered no trouble in sinking.

* General Manager, Eagle Coal & Mining Co.

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The general equipment, purchased from a company that had twice attempted to sink on this property and failed, consisted of one 80-hp. boiler, a hoisting engine, one No. 5 horizontal and one No. 6 vertical

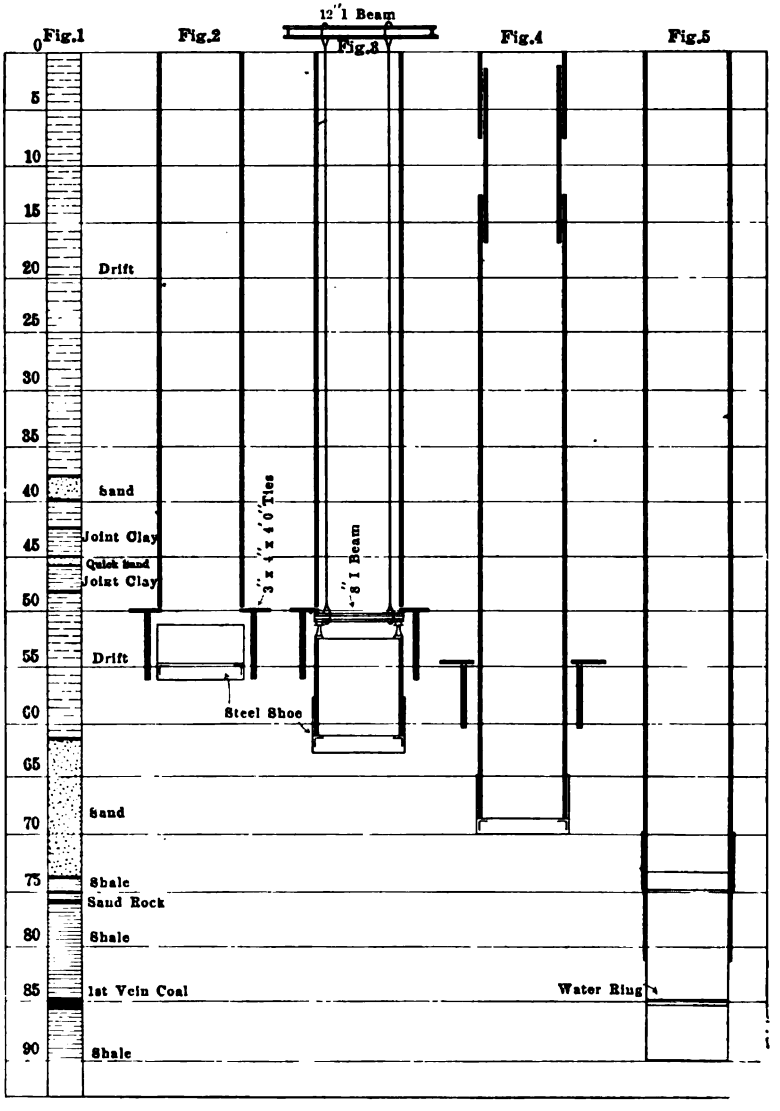

PLATE 1.

Cameron pump, a dump car, hoisting buckets, etc. The men employed were a day and a night engineer, one timberman, and three sinking crews, each consisting of three sinkers and one top man. Work was

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continuous 24 hr. a day until the sand was reached, when it was cut to two shifts, the remaining time being used in repacking pumps, etc.

The excavation was made 8 by 14 ft. The curbing for the first 50 ft. consisted of 2 by 4 and 2 by 6 timber laid flatwise, alternately with a quarter shaft bratticed off with 3 by 12 material, and the hoisting compartments divided by 4 by 6 buntons.

No trouble was experienced in sinking until the bed of sand at the 38-ft. level was encountered. This sand carried a small amount of water and, as a shaft protection, a concrete ring was put in behind the curb. The next trouble was experienced from the 42- to 48-ft. level. It had been the intention to assemble the shoe at this level, but what appeared to be a firm green clay proved to be jointed clay with a seam of quicksand through it and the exposed wall would not stand. Temporary 2 by 12 curbing was put in place and then regular curbing carried on down.

At the 50-ft. level, in order to safeguard the assembling of the steel shoe, 125 wooden mine ties, 3 by 4 in by 4 ft. long, were sledged into the clay just below the curb as shown in Plate 2. Steel angles, 6 by 4 in., were fastened with lag screws to the under side of the ties. An excavation 6 ft. deep was then made, undercutting the ties 16 in., and a false curb of 2 by 6 boards built. The shoe was then assembled at this point.

The shoe consisted of four sheets of $\frac{3}{8}$-in. steel, two 8 ft. long by 5 ft. high and two 14 ft. long by 5 ft. high, with 3-in. angles at the corners, and a 6-in. pressure angle placed 18 in. from the bottom of the shoe. The curbing was then built inside the shoe to a height permitting the jack screws to be placed against the curb timbers of the shaft, as shown in Plate 2.

The clay was then excavated, the shoe lowered into the sand, and pumping started. The sand carried about 60 gal. of water per minute, which would rise about 18 ft. above the sand when pumping was stopped.

In order to avoid the formation of cavities behind the curb, as little sand as possible was excavated. To prevent an inrush of sand under the shoe, at least $1\frac{1}{2}$ to 2 ft. of sand had to be left within the shoe. The method of sinking was to agitate the sand at the bottom of the shoe and force the shoe through it by means of the jack screws bearing against the shoe and the curb of the shaft. Two methods were used to agitate the sand. First, while the men could reach the bottom of the shoe, they stirred the sand with spades. By this method the shoe was lowered about 18 in. in the sand. Later the pumping system shown in Plate 2 was used.

In this second process the discharge of the No. 6 pump could, when desired, be sent through five $\frac{3}{4}$-in. pipes, as indicated in the drawing, and these five jets of water could be played upon the sand at the bottom of

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the shoe. It was found that these jets would agitate the sand sufficiently to permit the jack screws to push the shoe down, except when the sand was at too high a level inside the shoe, in which case sand would have to be excavated before the jet process could be resumed. In using this process the men would stir the sand with the jets for about 10 min., and then tighten the jack screws. When sufficient space was obtained between the timbers in the shoe and the curbing above, the jack

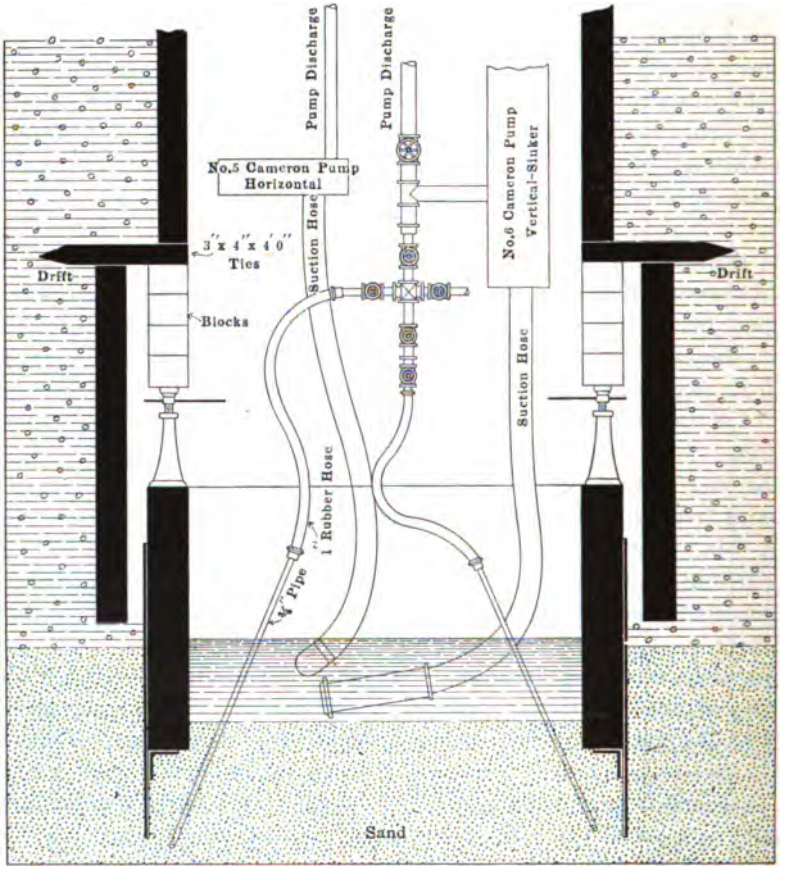

PLATE 2.—DETAILED SKETCH OF SINKING SHOE, JACK SCREWS AND PUMPS.

screws of one side were removed, and sets of timber put in place on top of the timbers in the shoe, and then the other jack screws were moved. An advance of 18 in. per day was exceptional by this method. More often it was less than a foot.

Necessarily some sand had to be excavated at times. This had a tendency to cave the dirt around the shaft, which in turn caused an excessive down pressure and broke the curbing apart a number of times.

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An attempt was made to overcome this trouble by supporting the curb with I-beams and cables from the surface as shown in Plate 1, Fig. 3. Ten 12-in. I-beams were supported on cribs at the surface, and ten 8-in. I-beams were swung under the 4 by 6 angle below the ties, these beams being connected by twenty 7/8-in. steel cables. When the sinking was resumed, the curbing continued to break, the I-beams bent, and two cables were broken. Since it appeared impossible to hold the curb, it was decided to timber the shaft solid from the 8-ft. level (where the most uniform break occurred) to the bottom of the shoe, and then drop this portion of the shaft through the remaining 5 ft. of sand.

To do this, the ties were driven back into the wall and solid timbering put in between the shoe and the upper curbing of the shaft. The entire shaft curbing from the 8-ft. level down was then tied together with 2 by 6-in. stringers. At the 8-ft. level, 2 by 4-in. by 16-ft. planks were spiked to the lower curb, the upper ends projecting above the break as shown in Plate 1, Fig. 4, preventing the loose material from falling down the shaft. The jet system, with occasional excavations of sand, was resumed, and the shoe, with 50 ft. of curbing, was lowered through the sand.

In landing the shaft on the solid, seven boulders from 1 to 2 ft. in diameter were encountered. Six of these were under the cutting edge of the shoe, and were removed only after being broken up by means of a long chisel and sledge.

While lowering the shaft through the sand, the upper timbers of the of shaft buckled 18 in. out of line. This necessitated retimbering of the shaft from the top of the sand to the surface, an expensive undertaking because the old timbers had to be cut out and replaced in sections.

On completion of the retimbering, sinking through the shale was commenced. Three shifts of four sinkers each were used with an average daily advance of 5 ft. The only problem involved in sinking through the shale was the elimination of the water, a considerable proportion of which was choked off when the solid was reached. This water was taken care of by placing a water ring at the 85-ft. level with a pump located at that point to elevate the water to the surface as shown in Plate 1, Fig. 5.

Air Shaft Sunk by Drop-Shaft Method

The air shaft was located 350 ft. from the main shaft, and its sinking conditions were similar except that the surface at this point was 10 ft. lower than at the main shaft, making the actual distance to be traversed to the solid 63 ft. 6 in.

The equipment was the same as that used at the main shaft. The air shaft followed a drill hole tapped by an entry from the main shaft, so that most of the water was drained through this drill hole and then

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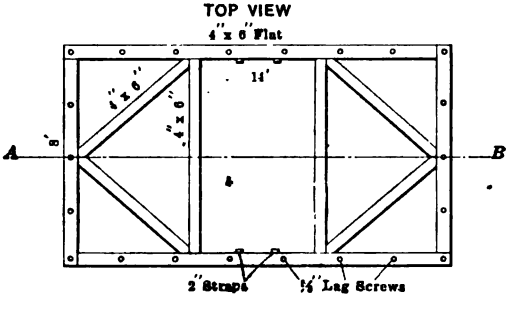
pumped to the surface. Two shifts of three sinkers, one top man, and an engineer were employed.

The size of the excavation was the same as before, 8 by 14 ft. The steel shoe was similar to the one in the main shaft, except that it was 10 ft. high instead of 5 ft. The timbering for the first 30 ft. above the shoe consisted of 4 by 6 laid flat, tied together by lag screws 5/8 in. diameter by 10 in. long, spaced 2 ft. apart. The shaft was divided into three equal compartments by 4 by 6 buntons as shown in Plate 3. The middle com-



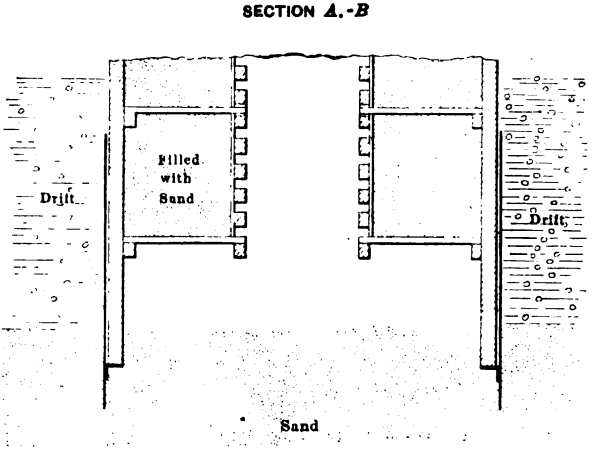


PLATE 3.—METHOD EMPLOYED IN SINKING AIR SHAFT.

partment was left free and was used for hoisting purposes. The end compartments were braced by 4 by 6 extending from the ends of the shaft to the sides. Four steel straps 3/8 in. by 2 in. tied the shaft timbers together from top to bottom.

An excavation 10 ft. deep was first made, the shoe assembled and lined with timbers. The sinking was continued until the shoe was hung up by the friction on the sides. Then a platform was built every 5 ft. in the end compartments. These 5-ft. chambers were filled with sand to give additional weight, and the sinking continued. The ground surrounding

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the shoe gradually broke in an oval shape. At one place it was necessary to fire a small charge of powder in order to loosen the ground sufficiently at one end of the shaft.

As the shoe was sunk, timbers were added at the top of the curbing. This method of building the curb at the top is decidedly better than that of adding timbers at the bottom since the timbers are placed much more accurately and expeditiously.

One difficulty experienced in the drop-shaft method was to keep the bottom of the shoe level. When one side of the shoe got lower than the other it kicked the opposite side outward. To right it, the lower side was blocked until the higher side caught up.

The progress through the drift material, until the sand was reached, was slow, much slower than at the main shaft. The drop-shaft went much faster, however, after reaching the sand. In fact, the difficulty at that time was to keep the bottom of the shaft from moving faster than the top. When within 10 in. of the bottom of the sand, the shaft broke apart 20 ft. up from the shoe. This was due to the fact that the movement of the shoe was faster than that of the top of the shaft, and to the insufficient strength of the straps connecting the top and bottom of the curb. At this point (20 ft. above the shoe) the curb separated from 6 to 8 in., and the upper part of the shaft kicked over 9 in. north and east of the lower part. A temporary platform of 8 by 8 timber was put in the end compartments of the shaft at this place, and time given for the upper part of the shaft to settle down before starting the excavation again. The sinking was then continued and the shoe landed on the solid without further difficulty, aside from hitting two small boulders at the bottom of the sand.

As the excavation was larger than necessary for an air shaft, it was decided to cement the shaft for a distance of 28 ft. from the bottom of the shoe, in order to shut off the water. A wall of cement 4 to 8 in. thick was accordingly then constructed.

After the cement was given time to set thoroughly, the excavation was again started in the shale and continued without difficulty to the coal. Sinking through the shale in the air shaft cost slightly more than in the main shaft because work in the mine prevented as careful supervision being given the sinkers in the air shaft.

One difficulty encountered in drop-shaft sinking was in keeping the position of the shaft vertical. At one time this shaft was 2 ft. out of plumb. By regulating the movement at the bottom of the shoe, the shaft partly righted itself, until at the finish, in a total depth of 63 ft. 6 in. to the shale, the bottom of the shaft was 16 in. to the south and 10 in. to the east of the top. Part of this variation was remedied in the cementing of the shaft.

A much larger amount of sand was removed in sinking the air shaft

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by the drop-shaft method than in sinking the main shaft. This could be done without danger of a cavity forming, because the surface dirt followed the air shaft down as it descended. When the sinking through the sand was completed, the surface directly surrounding the air shaft had caved to a depth of 15 to 16 ft. and for a distance of 20 ft. in all directions from the shaft. In fact, all the shale that was removed through the remaining 92 ft. to the coal did not fill this space at the surface.

A comparison of the costs of the two shafts is as follows:

	Main Shaft		Air Shaft	
Labor:				
Through drift material	$916.80		$789.08	
Through sand	1,941.80		541.62	
Through shale	1,065.68		1,213.30	
	$3,924.28		$2,544.00	
Superintendence	600.00		435.50	
Retimbering	1,343.23			
Cementing			207.68	
Total labor cost		$5,867.51		$3,187.18
Materials:				
Curbing	$1,878.62		$1,195.17	
Supplies	900.14		642.74	
Power, light, water, insurance, etc	1,248.50		649.71	
Total curbing, etc., cost		$4,027.26		$2,487.62
Total costs of shafts		$9,894.77		$5,674.80

Conclusions

In this particular work there was no question about the superiority of the drop-shaft method of sinking. It made a net saving of $4,300 in the total cost of the air shaft compared with the main shaft. A saving of $2,700 was effected in the labor cost, while in the cost of materials, power, etc., the saving was $1,600. A saving in time also resulted, 30 days being required to traverse the sand with the main shaft, while the air shaft was dropped through it in 17 days.

From the results obtained in these two shafts, and from the experience of others in the western interior coal field, we believe that the drop-shaft method of sinking is the safest, most economical, and most successful that can be adopted for sinking through soft material that lies within 100 ft. of the surface. At greater depths a variation of the method can be used by first sinking a larger shaft close to the soft material, and then telescoping a drop-shaft within it.

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Power Plant of the Burro Mountain Copper Co.

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(Arizona Meeting, September, 1916)

THE power plant of the Burro Mountain Copper Co. is located near Tyrone, N. M., at 5,950 ft. elevation. It is interesting because it uses the largest stationary Diesel engines in the United States.

The general layout of the plant is shown in Figs. 1 and 2. The building is of steel-frame construction with hollow-tile walls and corrugated-iron roof; the floor is concrete and all windows have steel frames, making a fireproof building.

The building is made to accommodate one more electric generator unit which is being installed; also two 4,000-cu. ft. air compressors direct-connected to three-cylinder engines with the same size cylinders as the electric units.

The power plant consists of two 815-kva., 60-cycle, three-phase, 6,600-volt, 180-r.p.m. generators, direct-connected to five-cylinder, two-cycle Diesel engines; the necessary circulating pumps for cooling water, cooling towers and oil storage tanks.

A portion of the power is used in the same building as the generating units to run one of two 200-kw. rotary converters delivering 260-volt direct current, and a 2,500-cu. ft. air compressor, delivering air at 95 lb. pressure, direct-connected to a 400-hp., 6,600-volt, three-phase synchronous motor.

The remainder of the power is used to run a 1,500-ton concentrating mill 3½ miles from the power house, various hoists, pumps and motors about the mines and the lighting and water-works of the town of Tyrone.

The direct current is used for electric traction in the mine, also to run hoists and motors that were installed before the new power plant was built.

Due to the large hoists that take starting peaks of 250 kw. and the 10-ton electric locomotive that takes at times 150 kw., the load is extremely variable, as shown in Fig. 3.

The two five-cylinder vertical Diesel engines, which are rated at 1,250 b.hp. at sea level, have cylinders of 525-mm. diameter (20.6 in.) and

* Consulting Engineer, Phelps, Dodge & Co.

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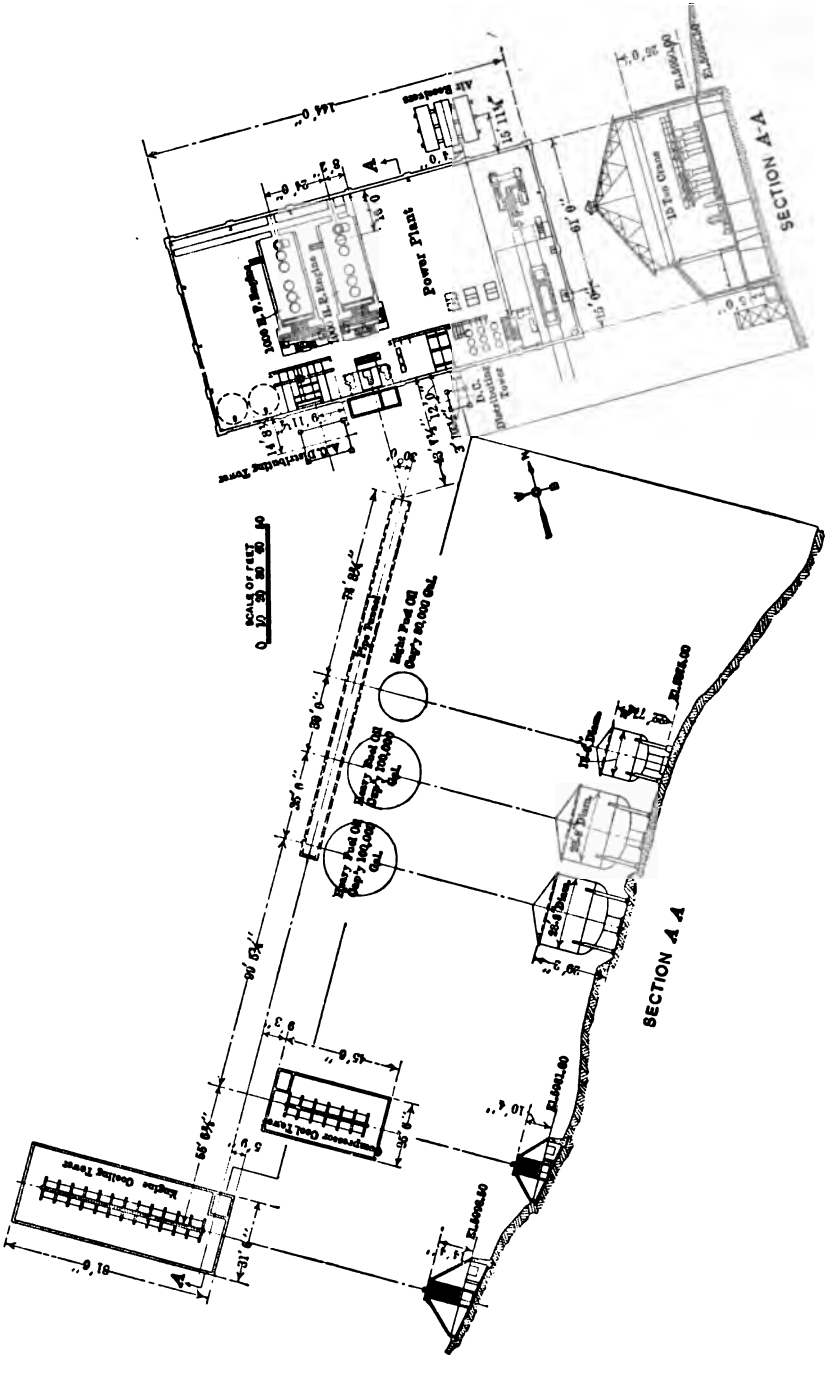

FIG. 1.—GENERAL ARRANGEMENT OF POWER PLANT EQUIPMENT, BURRO MT. COPPER CO., TYRONE, N. M.

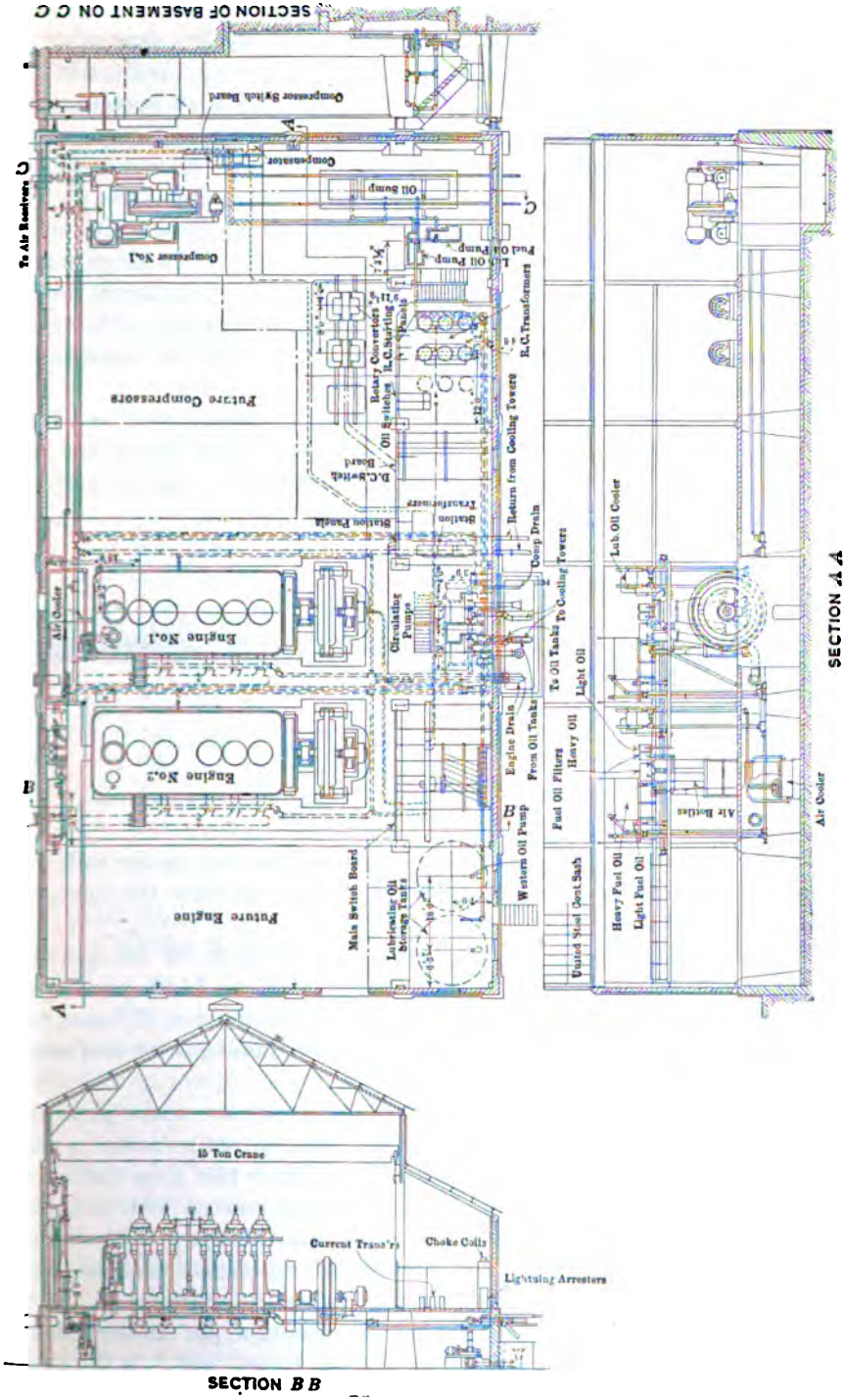

FIG. 2.—GENERAL ARRANGEMENT OF POWER PLANT OF BURRO MT. COPPER CO.

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660-mm. (26 in.) stroke; each engine has a scavenging cylinder of 1,050-mm. (41.25 in.) diameter and 600-mm. (23.6 in.) stroke; also a three-stage, four-cylinder high-pressure vertical compressor, both directly connected to the engine. This compressor delivers the air necessary for fuel injection and for starting the engine. The scavenging pump, which is larger than usual, delivers the air to blow off the products of combustion and fill the cylinders with fresh air at the beginning of the stroke This pump was increased in size to be able to fill the cylinders with air at 2½ lb. gage pressure at the beginning of the stroke. This gives nearly the same initial absolute pressure and allows the engine to generate nearly the same indicated horsepower as it would at sea level. The work done in the scavenging pump is, however, increased and the horsepower available is approximately 95 per cent. of sea-level output.

The fuel consumption per horsepower is increased over sea-level conditions because of the extra work of the scavenging pump and the resulting lower mechanical efficiency of the engine when operating at the elevation of this plant.

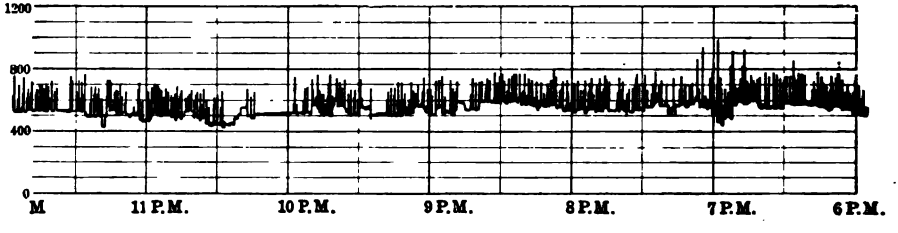

FIG. 3.—SECTION OF CHART SHOWING VARIATION IN POWER LOAD.

The plant has been in operation for 17 months, but as the mill was not ready until the middle of April, 1916, up to that time the load was carried on one engine.

The fuel used is California asphaltum base oil 14° to 18° Bé. gravity, averaging 16° Bé. and 18,360 B.t.u., costing $1.85 to $1.98 per barrel. The number of heat units in the oil was not determined at Tyrone, but at another plant which gets the oil from the same shipping point and under the same contract.

The pistons, cylinders, heads and exhaust pipes are water-jacketed; the jacket water of the exhaust pipes, which can be easily varied in temperature without danger, is used to heat the fuel oil to 120° F. as the heavy fuel oil cannot be used if cold. The engines are started with a lighter fuel oil of about 25° Bé. and run on this oil until the heavy oil is heated to the required temperature; when shutting off an engine, it is run for a few minutes on light oil to fill up the oil piping with light oil.

The curve in Fig. 4 shows the oil consumption per kilowatt-hour delivered at the switchboard when engines were tested new; it is the aver-

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age of several tests. It is interesting to compare the test figures with the actual consumption as shown on power reports of the company. Although individual months show variations from the curve, the yearly average for 1915 checks very closely with it, showing that the efficiency of the engines is well maintained and that the variable load has very little detrimental effect on the fuel efficiency.

By the courtesy of Phelps, Dodge & Co., owners of the Burro Mountain Copper Co., I am allowed to publish in the accompanying table the operating costs from Jan. 1, 1915, to June 1, 1916. These costs are as they stand on the books and are higher than is to be expected. They cover the running cost of the complete plant, exclusive of the air compressor. These costs do not include taxes, overhead charges or depreciation.

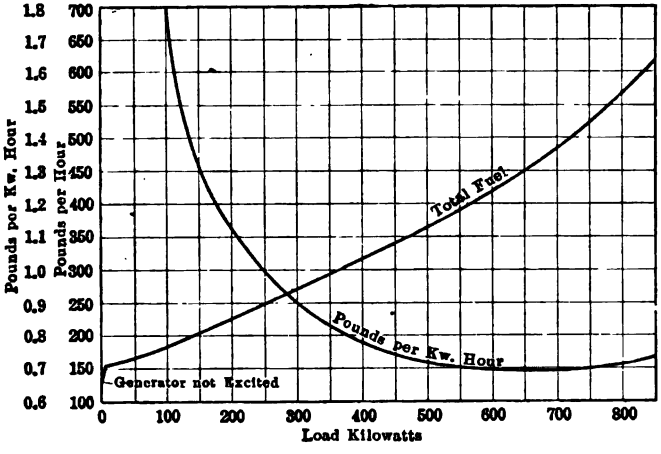

FIG. 4.—AVERAGE FUEL CONSUMPTION TESTS ON DIESEL ENGINE, BURRO MT. COPPER CO.

Diesel Engine: 5 cylinder, 1250 hp. (sea-level rating), 2 cy. ch.
Plant Elevation: 5,950 ft.
Fuel: 16° Bé California crude oil, 18,360 B. t. u. per pound.

tion. The cost per kilowatt-hour available is based on total power generated less that used for auxiliaries in power plant, such as lighting, circulating pumps for oil and water, etc. To simplify bookkeeping, the compressor is not charged with the cost of circulating its water or with its proportion of the lighting.

The labor cost is high because we have operated from the first with the full labor needed for operation when the power house is delivering the power necessary to run the mines and mill at the rate of 1,500 tons per day, a condition that has not yet been reached.

The repair cost is high, for soon after operations were started the construction account was closed and several items, such as labor for installing recording thermometers, measuring gages, some lubricating-

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TABLE 1.—*Burro Mountain Copper Co., Cost of Power*

1915	Labor	Fuel Oil	Lubricating Oil	Miscellaneous Supplies	Repairs	Total	Total, Kw.-hr.	Available Kw.-hr.	Cost per Kw.-hr. Available	Pounds Fuel Oil per Kw.-hr.	Load Factor	Average Load
Jan	$602.15	$754.79	$98.50	$34.50	$63.93	$1,553.87	101,315	93,322	$0.01665	1.29	0.18	136
Feb	796.60	742.50	72.60	33.31	198.44	1,843.45	77,400	69,512	0.02683	1.68	0.15	115
Mar	782.72	859.50	80.06	50.71	179.18	1,952.71	143,820	135,025	0.01446	1.02	0.26	192
Apr	731.83	919.71	71.45	85.67	291.32	2,099.98	134,473	125,795	0.01669	1.13	0.25	187
May	740.86	959.00	128.86	28.26	456.64	2,313.62	130,690	122,703	0.01880	1.24	0.24	176
June	705.83	882.92	142.96	29.47	393.57	2,154.75	152,646	145,347	0.01482	1.00	0.28	212
July	728.01	1,043.44	111.90	51.56	443.64	2,378.55	159,742	151,089	0.01574	1.16	0.30	228
Aug	762.68	1,037.96	110.99	34.08	477.01	2,422.72	155,800	149,397	0.01622	1.14	0.28	209
Sept	673.05	1,046.87	122.77	15.76	507.69	2,366.14	185,753	177,853	0.01330	1.00	0.34	253
Oct	705.01	1,284.25	122.66	17.97	689.58	2,819.47	275,836	267,174	0.01055	0.81	0.49	370
Nov	692.50	1,393.87	128.69	40.31	363.06	2,618.43	288,394	280,323	0.00934	0.84	0.53	400
Dec	702.45	1,492.40	132.50	- 9.81	316.58	2,653.74	335,702	326,580	0.00813	0.77	0.60	450
.....	8,623.69	12,417.21	1,324.48	431.41	4,380.64	27,177.43	2,141,571	2,044,125	0.01330	1.00	0.33	244
1916												
Jan	$712.15	$1,527.09	$124.88	$30.60	$285.79	$2,680.51	349,154	339,822	0.0078880	0.764	0.63	470
Feb	694.43	1,671.94	127.57	21.66	264.84	2,780.44	353,119	344,553	0.0080697	0.750	0.70	520
Mar	791.45	1,399.03	135.93	20.60	497.61	2,844.62	370,790	362,149	0.0078548	0.750	0.68	505
Apr	768.65	1,897.18	174.38	21.07	574.02	3,435.30	451,590	441,557	0.0077799	0.780	0.58	433
May	961.97	2,522.00	186.66	29.14	1,024.58	4,724.35	559,550	546,282	0.0086635	0.837	0.57	425
	3,928.65	9,017.24	749.42	123.07	2,646.84	16,465.22	2,084,203	2,034,363	0.0080935	0.780	0.62	464

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oil piping, etc., were charged to repairs. During the first year we also took apart the pistons, valves, etc., of the engine to satisfy ourselves that everything was going right and to determine if possible at what intervals this work would have to be done. This is shown by the relation of repair labor to material used in the repairs. The third engine is being furnished with scavenging valves having removable seats, and we decided to alter the valves of the two engines already erected while we needed only one engine in service.

The cost of repairs is shown in Table 2.

The repairs to circulating pumps and water piping are repairs outside of the engine water piping; the repairs on lubricating-oil system are also the repairs outside of the engine. The item of other repairs covers all repairs to building, switchboard, lighting, fuel-oil system and rotary converters. The latter should not be included under the cost of alternating-current power, but to separate them would have involved a great deal of work and the final result would not be appreciably different.

TABLE 2.—*Burro Mountain Copper Co. Power Plant Repairs*

1915	Engines		Circulating Pumps and Water System		Lubricating-Oil System		Other Repairs	
	Labor	Material	Labor	Material	Labor	Material	Labor	Material
Jan.	$46.40	$5.21	$1.50	$2.82	1.10	.....	6.90	
Feb.	142.95	2.61	36.10	2.49	.....	.....	11.70	2.59
Mar.	144.30	7.67	4.60	.....	.....	0.78	18.85	2.98
April	171.05	35.46	2.70	14.08	0.35	0.48	48.25	18.95
May	366.60	6.80	26.70	7.85	5.15	.....	42.30	1.24
June	297.50	3.52	53.80	20.71	2.58	2.58	14.50	0.96
July	234.60	3.51	79.60	58.56	.....	.....	50.10	17.27
Aug.	303.55	70.65	63.80	6.56	12.00	1.10	12.95	6.40
Sept.	223.75	7.67	70.30	40.47	33.95	11.79	95.95	23.81
Oct.	280.80	261.37	61.15	16.43	16.85	6.15	33.30	13.53
Nov.	299.67	5.92	7.95	3.04	2.75	0.15	18.70	24.88
Dec.	229.75	51.85	21.00	0.05	0.95	0.43	8.80	3.75
Jan.	169.00	0.34	19.70	0.10	59.45	9.54	12.20	15.46
Feb.	167.75	3.68	17.35	2.39	14.25	0.44	21.40	37.58
Mar.	284.95	152.50	.....	.....	9.40	2.56	36.05	12.15
Apr.	74.30	3.33	.....	.....	142.00	120.43	122.55	111.41
May	320.95	483.38	8.15	0.47	80.70	29.45	52.80	48.68

The water used in cooling the engines is mine water, which apparently has no bad effect on cast iron, but has a strong action on steel, so that all of our oil filters had to be rebuilt.

The circulating pumps come in for a large repair item due to breakage of check valves above the pumps which fell in and wrecked the pumps.

The two largest items of repairs on the engines were, one main bearing

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that was drained of its oil by a mistake of the oiler the first night we operated and one high-pressure air-compressor head that was wrecked by starting the engine without draining the air cylinder, after it had been idle some weeks. A small water leak had filled the cylinder with water and the head broke at the first revolution of the engine.

The greatest trouble we have had, not due to carelessness, is with the helical gears driving the camshaft. These gears have a tendency to get loose and one of them split from the keyway. We have not ascertained the exact cause of this trouble, which is now much reduced since iron bands were shrunk on the hub of the gears.

Two important items of running expense were doubtful when the engines were purchased: One was lubricating oil and the other the maintenance cost. The first one is now settled and we hope to reduce the cost from that shown in the table by the use of separate pumps to each point where the oil is delivered in the cylinders, insuring a definite quantity of oil. The present engines have one pump delivering to four points and it happened once that one of the oil pipes became plugged and gave slight trouble on one piston.

The maintenance is still in doubt, and cannot be determined until we reach and maintain regular operations for some time; but the writer thinks it will show a decrease from the figures shown.

The water-cooling connections to the pistons were expected to give some disagreeable troubles, but we have had practically none.

The operation of the engines in parallel is very satisfactory even when the load is light. In actual operation, with one engine in service, the peak load carried has been higher than was expected to be possible. Under test, 900 kw. was the limit with water-rheostat load, but in service, peaks up to 1,000 kw. have been carried for several seconds without seriously slowing down the engine.

On the whole, the operation of the plant has been satisfactory and the costs obtained are considerably better than with a steam plant of the same capacity run under the same conditions, amply justifying the extra investment for this type of engine where the cost of fuel oil is high.

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The Basic-Lined Converter in the Southwest

BY L. O. HOWARD, GLOBE, ARIZ.

(Arizona Meeting, September, 1916)

WHAT was perhaps the first attempt at basic converting in the Southwest was made by the late Charles F. Shelby at Cananea early in 1907, when he removed the acid lining from one of the 8 by 12-ft. barrel-type converters then in use at the reduction works of the Cananea Consolidated Copper Co. and substituted a lining of one course of magnesite brick. In this shell, he blew 21 taps (about 180 tons) of matte to white metal before the brick gave out along the tuyère line. In every case, the white metal so made was transferred and blown to copper in other converters. The cost of the lining was $700 and Mr. Shelby gave up the experiment as impracticable. No further serious effort to bessemerize copper mattes in a vessel having a more or less permanent lining of basic brick was made by any of the large copper smelters in the Southwest until some months after Messrs. Peirce and Smith had brought the question permanently before the public, and had proved it to be a metallurgical and financial success by their work at Baltimore and Garfield.

Early in 1911, the Cananea Consolidated Copper Co. had in operation at its reduction works at Cananea, Mex., several 8 by 12-ft. barrel-type converters lined with magnesite brick which had previously been operating with acid linings. In the 18 months following, these converters produced 50,000 tons of copper. In June, 1912, this company installed the first of its new equipment of six stands of electrically operated 12-ft. Great Falls type converters, and shortly afterward discontinued the use of the smaller shells.

In September, 1911, the Consolidated Kansas City Smelting & Refining Co. had installed at its El Paso plant two 10 by 26-ft. Peirce-Smith converters. Later, in March, 1913, one standard Great Falls type converter was added.

About the time the Peirce-Smith converters were going in at El Paso, the Calumet & Arizona Mining Co. was experimenting with basic linings in 7-ft. by 10-ft. 6-in. barrel converters at its plant in Douglas, Ariz. These efforts were sufficiently successful to enable the plant to handle the total output of copper in these small shells until the standard Great Falls converters were substituted in June, 1914.

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During the year 1911, the Anaconda Copper Mining Co., at its Great Falls plant, proved the worth of the so-called Great Falls converter. It was to this type, now being recognized as standard, that the Copper Queen Consolidated Mining Co. turned when it was decided to equip the reduction works at Douglas with basic-lined converters. In April, 1912, the first of these shells was put in operation. A second was added during the following month, and these two were quickly followed by six others.

In the meantime, the Consolidated Kansas City Smelting & Refining Co. had in operation, at its Hayden, Ariz., plant, two 10 by 26-ft. Peirce-Smith converters and later, in July, 1913, added one standard Great Falls converter.

The Arizona Copper Co. first used the basic-lined converter in October, 1911, at its old plant at Clifton, Ariz. The acid lining in some of the small converters was removed, and one of magnesite brick was substituted. The use of these small converters was continued until October, 1913, when the new plant with its equipment of three stands of electrically operated standard Great Falls converters was put in operation.

By January, 1913, the Old Dominion Copper Mining & Smelting Co. at Globe, Ariz., had installed a 12-ft. Great Falls type converter. It was the intention to handle in this shell the total output of copper, amounting to possibly 36,000,000 lb. yearly, but later, to insure sufficient converting capacity in case of accident to the larger shell, two of the old 7-ft. by 10-ft. 6-in. acid converters were lined with magnesite brick. It was not the intention, nor has it been the practice, to use these small converters unless absolutely necessary, and to date they have together turned out somewhat less than 1,500,000 lb. of copper.

The Detroit Copper Mining Co. at Morenci, Ariz., with a yearly copper output of about 11,000 tons, has never considered it advisable to adopt the large basic-lined converter. Instead, the 7-ft. by 10-ft. 6-in. acid-lined shells have been relined with magnesite brick and with these some remarkable work has been done, despite the serious difficulties that accompany the bessemerizing of copper mattes in such small basic-lined vessels. A record of over 3,100,000 lb. of copper converted with one lining, before any patching of the brick was necessary, has been made.

The Consolidated Arizona Smelting Co., at Humboldt, Ariz., likewise has continued in the very successful use of the small basic-lined converter, but contemplates the purchase of a large unit in the near future.

The International Smelting Co. at Inspiration, and the United Verde Copper Co. at Clarkdale, Ariz., have installed the standard Great Falls electrically operated converters in their new plants, both of which were put in operation during 1915. Unfortunately, owing to the newness of the latter plant and the fact that a proper working basis had not yet been

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TABLE 1.—*Operating Data of Basic-Lined Converters in Southwest*

	1	2	3	4	5	6	7	8	9	10
Blast pressure in pounds	13	13.2	14	12	12.4	12–15	10–14	14	11.3	13
Cu. ft. air per minute per converter	5,445	7,691	10,600	4,283	6,054	7,250	.....	8,181	4,033	4,117
Cu. ft. air per ton bullion produced	152,449	178,411	141,000	228,116	271,577	137,582	118,328	225,674	245,975	370,566
Cu. ft. air per ton iron and sulphur oxidized	96,145	142,224	156,000	98,847	139,361	112,950	118,328	119,475	240,937	123,583
Oxygen efficiency, taking into account total iron and sulphur oxidized	84.0	54.1	57.2	76.8	56.3	74.0	81.0	50.6	69.8	59.0
Total tons iron and sulphur oxidized per stand per month	2,075	1,610	.....	1,647	1,903	2,868	1,991	2,576	768	1,440
Punchers used per shell per shift	2	2	2	2	2	2	2	1	1.5	2
Tons bullion per puncher per month	221	204	522	119	478	392	255	137	129	240
Tons iron slagged per puncher per month	192	153	.....	169	533	251	171	156	147	
Average time of blow, hours and minutes	4–00	4–11	3–09	6–44	9–20	8–00	8–30	7–12	4–51	10–00
Average time to blow one ton bullion, minutes	28	23	14	54	45	19	28	28	51	90
Average tons bullion per blow	8.5	10.8	13.3	8.22	12.7	25.3	18.0	15.6	5.8	6.7
Average weight of matte per charge, tons	21.9	26.0	25.9	33.9	40.5	60.0	50.0	51.2	19.7	26.6
Tons ore charged per ton of matte	0.23	0.20	0.155	0.201	0.17	0.25	0.30	0.21	0.286	0.283
Tons iron slagged per ton of available silica	.....	2.44	.....	2.45	3.85	1.75	1.80	2.36	2.03	2.60
Size of converters	12 ft.	12 ft.	12 ft.	12 ft.	12 ft.	10 by 26 ft.	12 ft. Gt. Falls 10 by 26 ft. P.-S.	12 ft.	7 ft. by 10 ft. 6 in.	9 ft. by 7 ft. 6 in.
Number of tuyères	28	24	24	24	24	35	24 Gt. Falls. 35 P.-S.	22	12	12
Size of tuyères, inches	1¼	1¼	1½	1½	1¼	1¼	1¼ Gt. Falls. 1½ P.-S.	1½	1½	1¼
Thickness of brick used on tuyère line, inches	24	30	30	14	30	18	18	24	18	15
Average slag analysis:										
SiO_2	19.6	21.7	17.2	17.4	15.6	23.6	25.7	29.2*	22.2	23.8
FeO	68.2	67.9	69.1	70.7	67.3	66.2	60.0	62.1	63.4	61.5
CaO	0.8	1.1	0.6	0.8	0.4	.....	.....	1.6	0.9	
Al_2O_3	3.4	3.0	3.7	3.0	2.9	.....	.....	.....	2.8	
Average matte analysis:										
Cu	37.7	43.37	51.2	28.22	34.7	43.7	42.8	31.9	35.21	25.0
Fe	32.5	29.2	21.6	30.9	34.9	28.0	27.4	28.8	36.1	46.0
S	27.4	23.5	24.5	25.3	24.9	25.1	25.6	18.7?	25.0	25.0

1. Arizona Copper Co., Clifton, Ariz.
2. Old Dominion Copper Mining & Smelting Co., Globe, Ariz.
3. International Smelting Co., Inspiration, Ariz.
4. Calumet & Arizona Mining Co., Douglas, Ariz.
5. Copper Queen Mining Co., Douglas, Ariz.
* Insoluble.
6. Consolidated Kansas City Smelting & Refining Co., Hayden Plant, Ariz.
7. Consolidated Kansas City Smelting & Refining Co., El Paso Plant, Texas.
8. Cananea Consolidated Copper Co., Cananea, Sonora, Mex.
9. Detroit Copper Mining Co., Morenci, Ariz.
10. Consolidated Arizona Smelting Co., Humboldt, Ariz.

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arrived at, it was impossible to obtain any figures covering the converting operations.

There is no question about the superiority of basic over acid linings or, where the output of copper is sufficiently large to insure efficient operating, of the standard type over the smaller basic-lined converter. The inability properly to control the temperature of the charge in the small vessel, and the resulting destruction of the brick lining, are the most serious difficulties with which the operator using a small basic-lined converter has to contend. Throughout the Southwest, the preference has been for the standard upright rather than for the horizontal converter. The greater flexibility of the former, and the greater ease with which repairs may be made, strongly recommend it.

The operating data in Table 1 will give an idea of the working conditions and the results obtained at some of the large smelting plants in the Southwest. It is to be regretted that replies to questions pertaining to power consumption were at such variance that it was thought best to omit them entirely, as no reconciliation seemed possible.

TABLE 2.—*Old Dominion Copper Mining & Smelting Co.*

Campaign of basic-lined converter No. 2.	
Put in operation	June 27, 1913
Removed for initial patching	Dec. 7, 1915
Total hours blowing	13,734
Total number blows made	3,288
Average time of blow	4 hr. 11 min.
Total number taps matte	9,316
Total tons matte charged	85,578
Average weight of matte per charge	26 tons
Average copper content of matte	43.37 per cent.
Total tons bullion produced	35,431
Average time to blow one ton copper	23.2 min.
Average tons copper per blow	10.80
Average blast pressure	13.2 lb.
Average air used to blow one ton copper	178,411 cu. ft.
Average air used per minute	7,691 cu. ft.
Average air used per ton iron slagged	242,502 cu. ft.
Total ore fed	17,097 tons
Tons ore fed per ton matte blown	0.200
Magnesite brick used for repairs	None

Average Slag Analysis	Per Cent.
Cu	2.1
SiO_2	21.7
Fe	52.2
CaO	1.1
Al_2O_3	3.0
S	1.8

Average Matte Analysis	Per Cent.
Cu	43.37
Fe	29.2
S	23.5

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Separate operating data for the Peirce-Smith and Great Falls converters at El Paso could not be obtained, so the figures in column 7 are a combination of results obtained in these different types. The figures in columns 9 and 10 are for former acid-lined shells now operating with a lining of magnesite brick. All figures are for periods of 1 month or more. Those in column 5 are for the year 1915. The data in column 2 cover a period of 29 months' operation—the life of one original lining to the time of first patching. Although some of the figures in Table 2 have already been given, this more complete summary of the campaign may prove of interest.

I am greatly indebted to the managers and superintendents of the different properties who so graciously and promptly furnished me with the necessary information for this article, and take this opportunity to thank them.

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Mine-Fire Methods Employed by the United Verde Copper Co.

BY ROBERT E. TALLY,* B. S., JEROME, ARIZ.

(Arizona Meeting, September, 1916)

UNDERGROUND fires have been common in the mines of the United Verde Copper Co. for the past 22 years. The first fire started in the 300 Hampton stope in the fall of 1894, following a cave in that orebody. The soft and highly pyritic nature of the ores is responsible for most of these fires. Since ore characteristics, methods of mining and ventilation, have an important influence on mine fires, they will be described in this paper.

Orebodies

The ores are mostly chalcopyrite except on the upper levels where considerable chalcocite, black oxides and other secondary minerals occur. The orebodies are in the form of lenses and vary in size. Some are small, while others are several hundred feet in length by a hundred or more feet in width. These orebodies extend, in some cases, from the surface to the lowest levels of the mine and usually at a steep angle of inclination.

There are three different classes of ores, classified in accordance with the gangue rock. The copper in all cases, except as above specified, is in the form of chalcopyrite. The largest and most important of these reserves are in a pyrite gangue. Next in order of importance are the schist ores and lastly the quartz porphyries. The pyrite, locally termed iron ores, average from 30 to 42 per cent. sulphur. The schist and quartz porphyries vary from 3 to 25 per cent. sulphur.

Mining Methods

All the orebodies were worked by the overhand square-set method of mining prior to 1908; since then the cut-and-fill, shrinkage and mill-hole systems have been largely employed.

But 15 per cent. of the total tonnage now extracted is from timbered stopes. Most of this is from the upper levels where the ore is soft, heavy and more or less broken to the surface. Large reserves of these ores are being conserved for more efficient methods of mining.

* Superintendent of Mines, United Verde Copper Co.

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There was no choice in the past with reference to the methods of working the upper levels because the smelting plant and other surface buildings were directly over the orebodies. Only such places were worked as were considered necessary and by methods which caused as little settling as possible. In most of these places it was necessary to use spiling and to fill to the roof. Bulkheads were used on the levels to retain the gangways and were carried up through the stopes for ore chutes.

Top slicing could not be used on account of surface settling and because this method is unfavorable, in heavy sulphide orebodies, for the control of mine fires.

The smelting plant and other surface interferences have now been removed and it is planned to start steam-shovel work in the near future.

History of United Verde Fires

The first fire of 1894 was caused by spontaneous combustion, following a cave. It was not extinguished and the district was bulkheaded from the remainder of the mine. The soft and broken nature of the ground in this district was such that the smoke and gas worked its way over the orebody on the several different levels and cut off a large productive area of high-grade ore.

Later, another serious fire occurred in the Chrome or quartz-porphyry orebodies, which extended over a large area and cut off production from this district.

A third fire followed, cutting off the middle district which lies between the Chromes and Hamptons. Many smaller fires occurred in the meantime, which were extinguished. There has never been a serious fire in the schist orebodies which join the Hamptons on the south, yet there is always danger in that or any other heavily timbered district, where the ore carries any appreciable amount of sulphur.

Fire Stopes

Many attempts were made to extinguish these fires and recover the large reserves of high-grade ore. The districts were first flooded with water but, owing to the large area involved and the soft, broken-up condition of the ground, this method was a failure. Sufficient air always found its way through fractures to feed the fire.

The next attempt to extinguish the fire was by CO_2 gas. A gas-generating plant was installed, pipe lines were laid and large volumes of gas were forced into the district, but without effect. Steam was then tried and was likewise a failure. Water, steam or CO_2 gas will readily extinguish a fire if the district in which the fire occurs can be tightly sealed up and the extinguishing agent brought into contact with the fire in

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sufficient quantities. Fractures to the surface and underground workings outside the fire district made an impossible condition to seal up.

In 1905, the Plenum system was tried on this district with success. Actual combustion was extinguished and the latent heat reduced to a temperature of about 120°. This work was planned and directed by

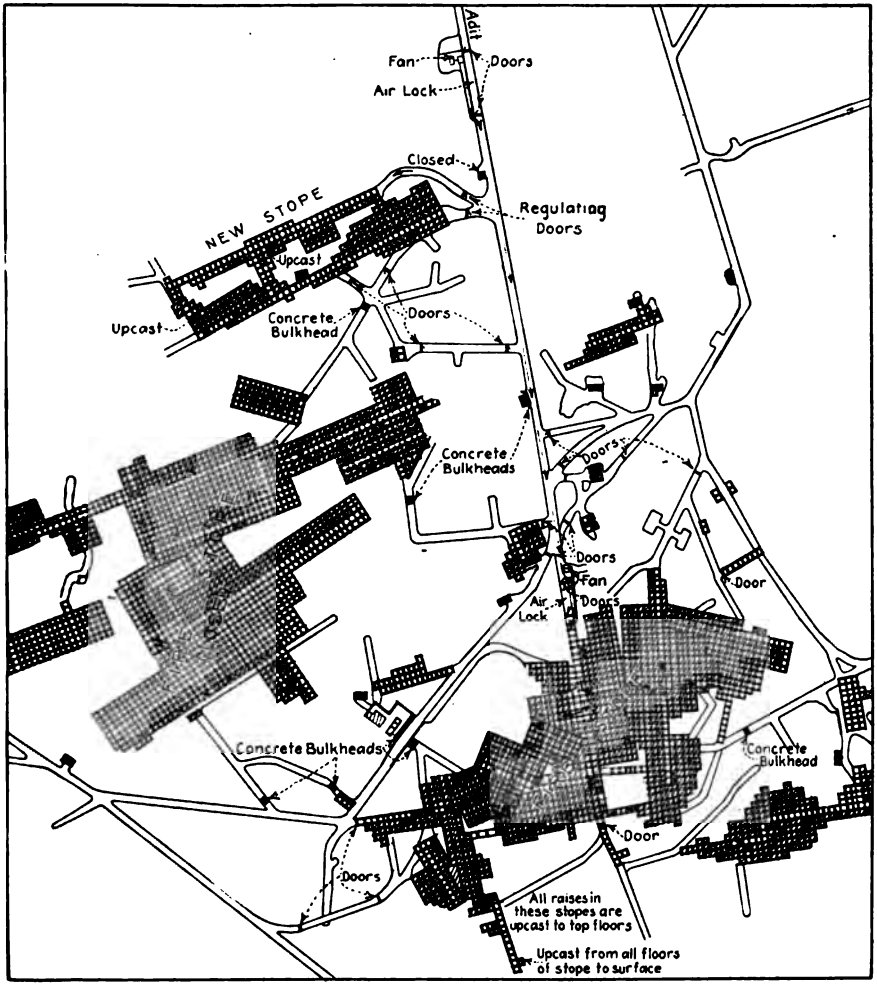

FIG. 1.—PLAN OF FIRE STOPES ON 300 LEVEL SHOWING METHOD OF VENTILATION.

Joseph J. Shaw, a mining engineer from the Mountain Copper Co. of Keswick, Cal., where this system was successfully used. It has been used here continually since its inception and consists in forcing the air under a pressure, varying from 2 to 5 lb., into the fire district. The air pressure varies in accordance with the gas pressure and must be sufficient

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to keep back the gas and to cool the ground, so that work can be accomplished.

When a bulkhead, in a connection leading to a fire stope, is opened there is invariably an outward pressure and rush of gas or smoke, which must be overcome. The Plenum system consists in forcing the air against this gas at a pressure slightly greater than that of the gas.

Fig. 1 shows a large fire stope on the 300 level, with doors for controlling the pressure, and the method of ventilation. The air is furnished from the adit and forced under pressure into the fire district. All chutes and manways are upcast, and the spent air and gases outlet to the air raise through connections on the different floors.

The installation used with this system consists of a No. 8 double-inlet Sirocco fan, direct-connected to a 50-hp. variable-speed General Electric motor; also the necessary airways, doors, etc. The air from the fan must be under perfect control in order to regulate the gas. This is accomplished by doors, as shown in Fig. 2.

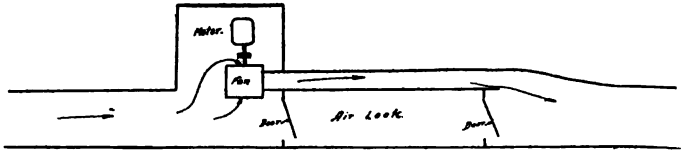

FIG. 2.—SKETCH OF FAN STATION.

An airway about 3 ft. square and from 50 to 200 ft. long is connected to the outlet of the fan. Doors are placed in the tunnel at both ends of this airway. These doors prevent the fan from drawing its own air, for when one door is open the other must be closed. All workings between the fan and the fire district must be closed off by doors; otherwise the air pressure will be reduced and trouble result from gas. Two doors are always used so that one will remain closed when the other is open. For electric haulage these doors should be from 100 to 200 ft. apart. The doors are made of 2-in. grooved lumber, and concrete frames. The airways are made of lumber or concrete.

With a variable-speed motor and the proper system of doors, it is an easy matter to increase or diminish the air pressure in order to meet any new conditions within the fire district.

When the bulkheads to this district were first opened, the ground was red hot. The roof and walls, where opened, were aglow. All timber was burned and the soft sulphides, where exposed, were burning. The continual blowing of air on this hot mass gradually cooled it, so that within a few weeks the actual extraction of ore was under way.

The timber often took fire before it was blocked, making it necessary to keep it well sprinkled with water. The air reduced the temperature

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from 1,200° F. to 120° F. in about 6 weeks. The conditions gradually improved until the average temperature in the fire stopes was 100° F. During recent years a system of ventilation has been introduced whereby the temperature has been reduced to about 75°, except in places that are above the line of ventilation.

Formerly the air was forced into these stopes and found an outlet through fractures into the other workings and to the surface, with the result that none of the upper levels were free from gas.

A system of air raises was driven in the foot wall to the surface and connections from these raises to the stopes were made at different elevations. Doors were placed in these connections and the air outlet thereby controlled. If an outlet is too free, it will not only release the pressure and cause gas, but, by suction, will also cause fire. These raises were driven in ground that did not require timbering. Cast-iron sets were used where necessary in the connections from the raises to the stopes. Timber should be avoided if possible in airways, since there is a constant danger of fire in these places.

These stopes have now been opened for 11 years and fire occurs on an average of about once a month in a rather extensive area of old timbered workings. It is therefore important to have an efficient fire-fighting organization and to have good control of the ventilation. This is an important factor in connection with mine fires. In the first place, every preventive measure must be used, and, in the second, the ventilation must be under absolute control in order to extinguish the fire.

An excess of air increases the flames in the same manner that an open draft increases the fire in a stove or furnace. Air must be furnished only in sufficient quantities to carry away the smoke so that the firemen can get at the fire. Where the smoke is thick it is impossible, even with helmets, to handle the fire satisfactorily. Many unsatisfactory results in connection with mine fires are due to these causes.

Causes of Mine Fires

Probably 90 per cent. of all underground fires in pyrite or heavy sulphide ores are caused by spontaneous combustion. In the fire district of this mine, many fires are due to red-hot dust dropping on the timbers, from fractures in ground above. Some fires are from friction due to caving, some from defective electric wires or cables; others from burning powder in contact with timber, and a certain number from incendiarism and carelessness with candles, lamps, etc.

Fires from spontaneous combustion usually occur in the interior of a gob or filled stope and are due to the oxidization of fine sulphides in contact with timber. Very small amounts of air are necessary for spontaneous combustion and this air works its way through the filling along the line of bulkheads, timber or walls, where it comes into contact with sul-

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phide dust. In the process of oxidization this produces sufficient heat to start a fire. This theory has been proven here by experiments. Heavy sulphides should not be used for filling in timbered stopes. Considerable fine sulphide ore is lost in the fire stopes or in any other heavy ground where the square-set method of mining is used and where the floors are kept filled to the roof.

Prevention of Mine Fires

Careful consideration should be given toward making heavy sulphide mines as fireproof as economy will permit.

Main hoisting shafts in such properties should be constructed of concrete or other fireproof material. Where timber is used, a sprinkling system should be installed.

Mine methods should be developed in which timber will be eliminated as far as possible.

Heavy sulphide material should not be used for filling in the timbered stopes. In these places fine waste should be used and sprinkled so that it will pack and make conditions unfavorable for fire.

In fire stopes or stopes adjacent to the fire district, the timber next to the walls or ends should be removed.

Careful attention should be directed toward ventilation so that the temperature will be unfavorable for fires to start.

In hot places the timber should be kept damp by sprinkling with water. Water should never be put on red-hot or burning ground, as this will invariable cause an explosion, one of the great dangers in handling mine fires.

Water and air lines, hose and hose connections and all tools used in connection with mine fires should be kept convenient and in good working condition where fire is liable to occur.

Helmets, pulmotors or lungmotors, electric lamps and supplies should be kept in stock in sufficient quantities and convenient for immediate use.

Carefully selected men should be trained in the use of the helmets and resuscitating machines, and in the methods of fighting fires. The average man, unless specially trained, is useless in fighting fires. The foremen, shift bosses, electricians, pumpmen, watchmen and selected miners working in different places which are subject to fires should familiarize themselves with all things appertaining to the ventilation, and should know just what to do if fire occurs in their respective places. Rules to this effect should be printed, distributed, and understood by all concerned.

Watchmen should inspect the air outlets often and regularly in order to detect smoke as soon as possible.

Iron fire doors should be erected in connections near all timbered stopes, so that in case a fire gets beyond control the doors can be closed and the district quickly sealed up, in order to protect the remainder of

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the mine. Similar doors should also be erected near timbered shafts, airways, etc., for protection against fires.

A fire signal should be in use at all mines and understood by all employees. A practice signal should be given occasionally. The following fire signal is in use here:

"In case of fire, ring nine bells on the electric cage call signal. The engineer on receiving these bells will flash all electric light throughout the mine, nine times. This flash will be repeated three times and followed by flashing the station signal on which level the fire is, three different times. Carmen and all others working where there are electric lights will notify those employed in stopes and other places where there are no lights. The trained firemen on the various levels will then take charge of the situation. Their first consideration will be for the safety of the men and then for the extinguishing of the fire and protection of property."

When a fire occurs in the fire district no signal is given, for the system there is such that the fire can be easily confined without danger from smoke or gas to the other parts of the mine.

When fire occurs in any other part of the mine, the signal is given and all men working where there is danger from suffocation are instructed to go to the surface or other places of safety. The next step is to get water on or over the fire as quickly as possible. When the fire is in the interior of the gob, it is sometimes necessary to drift from a chute or manway through the filling in order to get at it. Often, however, the fire can be extinguished by water from above. This is usually accomplished by driving holes in the filling with a pinch bar and turning water into these holes. For this work, hose nozzles made of 1-in. pipe with about a $\frac{3}{8}$-in. hole are used. By this method, water can be scattered all over the stope. It will keep the fire from spreading and usually extinguish it.

In case a fire gets beyond control, the fire doors should be immediately closed, after which concrete bulkheads should be erected. It is sometimes possible to extinguish a fire in a sealed-up stope by water, through a diamond or churn drill hole. In sulphide stopes, however, it is usually necessary to burn the timber before the stope can be recovered. A fire, when apparently extinguished, may start again when the bulkheads are opened, unless the timber is consumed.

At the present time one large fire district here is being experimented with along these lines and with apparent success. Air in small amounts is gradually forced into the stopes to burn the timber. This causes the filling to settle, and, to avoid caving, more waste is put in from above.

Ventilation

Ventilation has received considerable attention here during the past 10 years, not only in the fire district but throughout the entire mine. Sulphide mines are naturally warm and disagreeable unless ventilated. Natural ventilation is used where possible; otherwise, fans are used to

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blow the air where it is required. The ventilation of the fire stopes interferes with the remainder of the mine, since the air is taken from its natural source and forced into the fire district. Artificial ventilation is used for the middle and lower levels.

Fig. 3 is a plan of one level showing the location of the fan and the method of ventilation. All levels are connected to the air raise.

Fig. 4 is an ideal vertical section of the orebodies, showing the direction of the air currents, and the methods of ventilation.

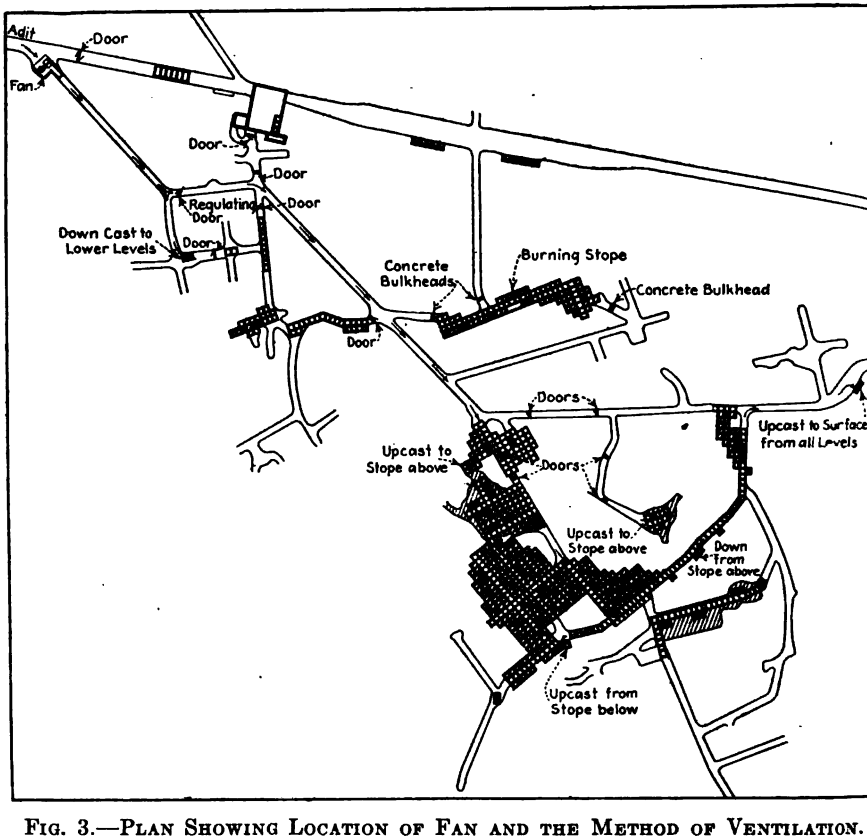

FIG. 3.—PLAN SHOWING LOCATION OF FAN AND THE METHOD OF VENTILATION.

Fans are installed in suitable places where they can draw air from the outside and force it into the different workings, thence to the air raises to the surface. The same airways and doors are used as for the fire district, except that no particular pressure is maintained, the air having as free an outlet as possible to the surface. The ventilation for any particular place or section is regulated and controlled by doors. For the proper ventilation of stopes, the air should have an inlet at one end and an outlet at the other. Most air is forced into the main working places and only a

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sufficient amount is sent to the old workings to preserve the timber and make conditions unfavorable for fire.

The maximum temperature in the middle and lower levels is 80°F. It varies from 60 to 80° F. and averages about 72°, with small amounts of humidity.

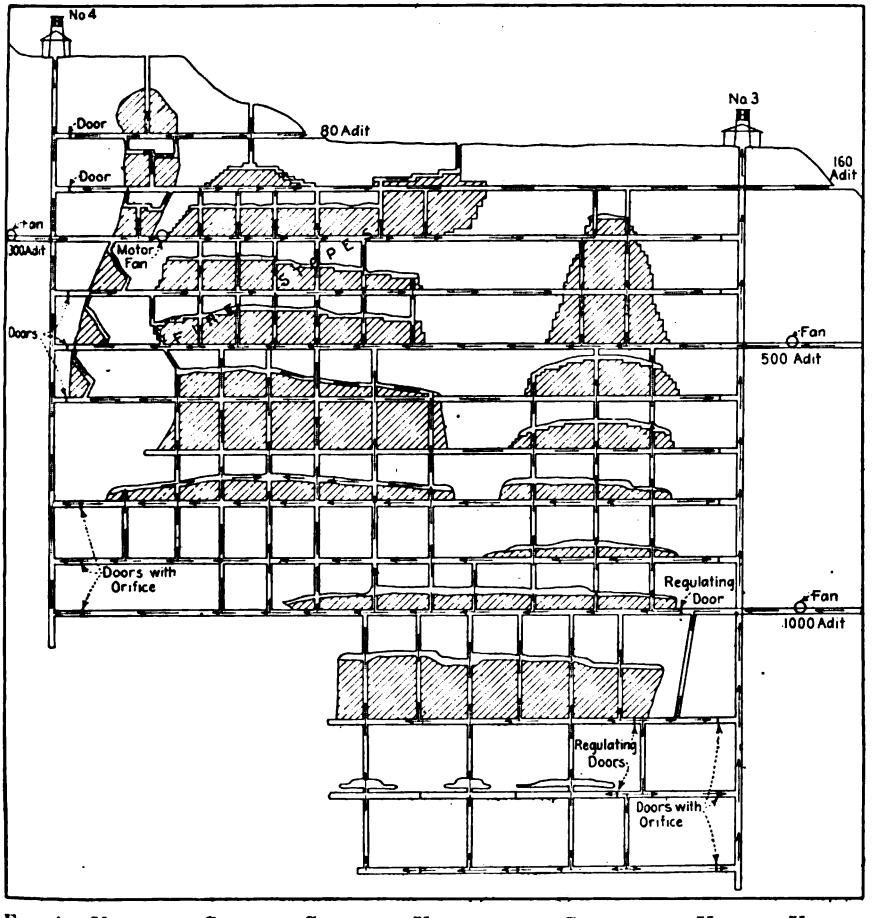

FIG. 4.—VERTICAL SECTION SHOWING VENTILATING SYSTEM OF UNITED VERDE MINE, JEROME, ARIZ.

Conclusions

For handling mine fires satisfactorily, the Plenum system as employed here has been a marked success, but the mining methods in connection therewith are necessarily expensive and inefficient.

Mine fires are the most disagreeable and dangerous conditions that are encountered in metal mining and every possible precaution should be exercised to prevent them. In addition to preventive measures, there should be carefully trained firemen who thoroughly understand how to control and extinguish fires.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Cyaniding Clayey Ore at the Buckhorn Gold Mine

BY PAUL R. COOK,* B. S., GUAYAQUIL, ECUADOR

(Arizona Meeting, September, 1916)

THE ore deposit of the Buckhorn Mines Co., Buckhorn, Nev., is peculiar in being a shallow kaolinized mass of material with basalt walls, and having apparently no direct connection with any of the usual gold-bearing rocks. The average ore contains 16 per cent. water of hydration, and the cyaniding of this hydrous clayey material offered unusual difficulties as compared with the typical gold quartz ores of Nevada.

The orebody was thoroughly developed; then the mill was built according to the latest cyanide practice, with such changes as were thought to be demanded by the peculiar nature of this ore.

Upon starting the mill, the ore proved more difficult to handle than had been anticipated. It is hoped that an account of how these difficulties were met may prove of interest to anyone having a clayey ore to handle and to the profession in general.

GEOLOGY

The Buckhorn orebody lies along a north and south fault plane of perhaps 1,000 ft. dislocation, that can be traced for miles; but the only other known mineralization consists of similar ore in the Murphy mine, a mile farther north.

The east or hanging wall is hard and smooth, being a typical fault plane. The best ore is along this wall, gradually grading down toward the west until at 30 to 60 ft. it is too low-grade to mine. The country rock on the west consists of alternating layers of hard and soft basalt and basalt scoria, pitching toward the mine.

One of these basalt layers on the hillside a little above the mine is marked, for 3 or 4 miles in length, by a line of springs which seep out along it. Perhaps the meeting of the surface drainage, passing down these basalt layers, with the fault-plane solutions explains the formation of the Buckhorn orebody.

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*With Buckhorn mines during the first year of mill operation (December, 1913, to December, 1914).

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Beneath the oxide ore a smaller body of almost pure marcasite occurs with about the same assay value ($8 per ton). Beneath the 250-ft. level there appears to be no further mineralization.

MINING

The first difficulty was to get the ore out of the mine. Since the orebody extended from the grass roots to a depth of 175 ft., with a width of 50 to 80 ft. and a length of 1,400 ft., the "glory-hole" system of mining was adopted. The ore was broken through 8-in. grizzlies into 18 chutes extending from the surface to an intermediate level, on which it was trammed by hand to three chutes leading to the 200-ft. level, and hauled by electric motors through a 1,000-ft. tunnel to the mill.

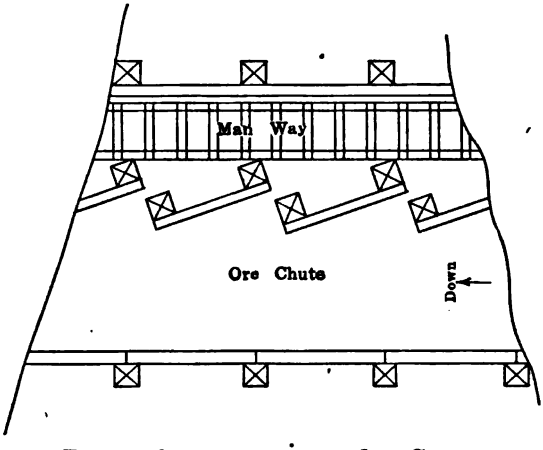

FIG. 1.—SECTION THROUGH ORE CHUTE.

In the summer little trouble was experienced; but the ordinary rain and snowfall of a Nevada winter made it almost impossible to keep the chutes open, as first built. After reconstructing, so as to provide an opening for barring at each set of timbers (see Fig. 1), it became possible to keep the mill supplied with ore during all kinds of weather. During wet weather all the cars had to be scraped with a shovel after dumping and unusual care was required to keep a sticky load from carrying car and all into ore bin or over the dump; 30 per cent. of the tonnage mined went to the waste dump.

MILLING

Ore Bin and Crusher

The next problem was to get the ore out of the mill bin and crushed. The bin was an ordinary circular steel bin, with natural earth bottom and

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side gate. This ore absolutely refused to run from the bin. The mill was built to treat 300 tons a day, but even with one man in the ore bin, and two at the crusher, it was impossible to get over 150 tons through in 24 hr. The large kaolin lumps gave the most trouble in crushing. They had to be practically chiseled to pieces and poked through the jaw crusher by hand.

The replacement of the jaw crusher with a high-speed toothed roll (see Fig. 2) gave the desired crushing capacity. This machine was developed at one of the Bingham Canyon (Utah) properties, and is manufactured by a Salt Lake firm. It is well adapted for sticky ores. The Buckhorn ore contains an occasional "nigger head" of very hard "malapi" or basalt. The mill crew was afraid one of these would break off the teeth of the crusher shell, and they were very carefully picked out at first,

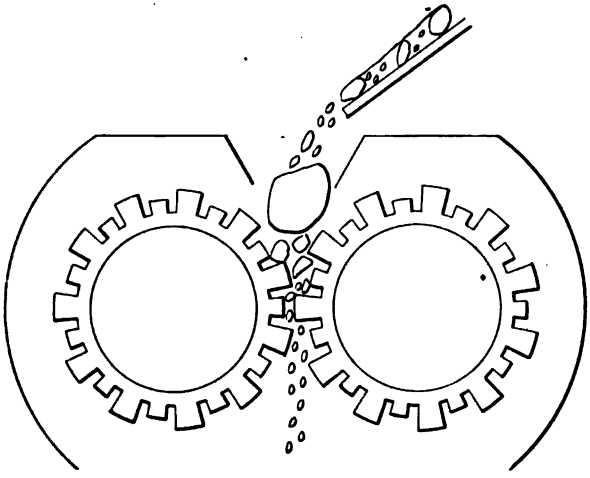

FIG. 2.—SECTION THROUGH WALL CRUSHER SHELLS.

but after a few had gone through by accident, and a few pieces of drill steel, etc., had been chopped up, no further concern was felt. All the boulders that had been sorted out were later put through as it was found that the easiest way to keep the teeth free of adhering clay, was occasionally to throw in a boulder of hard rock.

To do away with the necessity of a man on each shift to shovel the ore out of the bin, a 36-in. conveyor belt was installed to feed the crusher automatically. The opening in the bottom of the bin, over the belt, was about 2 ft. wide, and extended clear across the bin. It was closed by means of short pieces of mine rail that could be removed a few at a time, permitting the ore to be drawn from any point desired. The belt was driven from the crusher line shaft by means of ratchet and dogs. It could be started or stopped, from either floor of the crusher building, by means of a rope connected with the dogs. With these

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improvements, crushing required only a part of one man's time. Two men each shift attended to crusher, rolls, two ball mills, two classifiers and two tube mills.

Rolls

The 45 by 15-in. Anaconda-type rolls with smooth shells would clear themselves fairly well, if one of the shells had a channel about 1 in. wide by ½ in. deep machined in it, but it was troublesome to keep a groove in the shells as they wore down. A corrugated or toothed shell would have been better for this ore.

Ball Mills

One 6-ft. Hardinge ball mill was intended to handle the whole tonnage. After plastering the balls to the side of the mill with clay, a few times, the shiftmen learned to tell by the sound of the mill, when it was beginning to coat up. By shutting off the feed at this time, it took only a few minutes for the coating to be ground out.

The installation of a duplicate ball mill made it possible to keep the rest of the plant going while grinding out the ball mills, one at a time, and allowed the rolls to be set coarser on troublesome ore. With a good run of ore, 300 tons per day was sometimes put through one mill.

Classification and Tube Milling

This ball mill discharge was classified in two 36-in. Akins classifiers, the sand from which was fed to two 5 by 18-in. tube mills with Komata liners. The tube-mill discharge was classified in a home-made drag classifier. The small percentage of material requiring regrinding consisted largely of fragments of the basalt "nigger heads." This material was almost as hard as the pebbles themselves and of low assay value. Occasionally it accumulated in the circuit enough to be troublesome and was thrown away. A small amount of it was thought to help the grinding. About 80 per cent. of the product delivered to the treatment plant would pass a 150-mesh screen.

Agitation

About 80 per cent. of the mill-head value was dissolved in the crusher plant. Only a trifling additional extraction could be obtained either in the mill or experimentally. The real trouble was to remove the dissolved value from the clayey pulp. Accordingly the three 32 by 14-ft. Dorr agitators were changed to thickeners.

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THICKENING

These three "converted" agitators settled 300 tons per day of 1 to 10 pulp, as delivered from crusher plant, to a specific gravity of 1.15. The 8 sq. ft. of settling area provided per ton of this ore settled in 24 hr. would be sufficient to settle an average Nevada quartz ore to a specific gravity of 1.33. The overflow was precipitated, and the underflow mixed with the barren solution and fed to six 36 by 12-ft. Dorr thickeners, delivering a 1.23 specific gravity underflow to the filters. The 20 sq. ft. of settling area per ton settled in 24 hr. is three times the area required to settle an average Nevada quartz ore to a specific gravity of 1.33. Primary thickeners were held with 2 ft. of clear solution; the secondary thickeners with 6 in. It was impossible to settle the raw Buckhorn ore beyond a specific gravity of 1.26, either in the mill or experimentally.

Filtering

The maximum capacity of each of the four 14-ft. diameter by 12-ft. face Oliver filters was 50 tons per day, about one-half their capacity on a Nevada quartz ore. An additional filter, 14-ft. diameter by 24-ft. face, had to be installed to handle 300 tons per day.

Dehydration

A sample of Buckhorn ore carefully dried at a temperature below 110° C. had a specific gravity of 1.9. A higher temperature gave an additional loss of 16 per cent. in weight, and entirely changed the physical properties of the ore. The dehydrated sample had a specific gravity of 2.4, and settled and filtered almost as well as a quartz ore. Dehydrating also removed the sticky milling qualities. Both samples, however, gave the same extraction with cyanide. The temperature of a laboratory electric hot plate was sufficient to dehydrate a sample nicely. As CO_2, etc., would not be driven off at this temperature, this loss in weight must be due to water of hydration.

With a cheap fuel supply, dehydration before milling would be the best treatment for this class of material. The ore would mill and classify more easily; the thickeners and filters would have normal capacity; and dissolved values would be more completely removed. The temperature of a commercial drier would dehydrate the ore with about the same fuel consumption (100 lb. of coal per ton of ore) as in removing the 18 per cent. of H_2O if it existed in the form of moisture.

The high price of fuel delivered at Buckhorn prevented the adoption of dehydration at this mill. The ore was milled raw at the cost of $1.59 per ton (see Table 1). Power cost $8 per horsepower per month.

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The careful drying at a temperature below 110° C. of a large number of samples, with the equipment of an ordinary cyanide plant assay office, would be a rather tedious operation; so the regular moisture and assay samples at Buckhorn were dehydrated. All assay, moisture, tonnage, etc., figures are on this basis.

To compare with other ores the figures obtained by drying below 110° C. should be used. Both sets of figures are given in Table 2. Table 3 shows the mill flow sheet.

TABLE 1.—*Buckhorn Mining and Milling Costs*

Ore Milled, 10,000 Wet Tons, 8,100 Dry Tons, H_2O 19 Per Cent.

MINING

	Per Ton	
Ore Breaking:		
Labor	$0.259	
Supplies	0.098	$0.357
Tramming 100-ft. Level:		
Labor	0.044	
Supplies	0.002	0.046
Timbering	....	0.013
Electric Haulage:		
Labor	0.084	
Supplies	0.006	
Power	0.022	0.112
General Expense:		
Surface drainage	0.004	
Haulage tunnel repairs	0.039	
Assaying and sampling	0.027	
Surveyor	0.011	
Superintendence	0.067	
Incidentals	0.024	
Development	0.083	
Overburden and waste	0.174	0.429
Grand Total	....	$0.957

MILLING

	Per Ton	
Crusher and Rolls (Wall-toothed Roll and 45-in. Roll):		
Labor	$0.057	
Supplies	0.003	
Power	0.034	$0.094
Hardinge Ball Mills (Two 6-ft. Mills):		
Labor	0.050	
Supplies	0.026	
Power	0.076	0.152

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Elevating and Separating (Two 36-in. Akins Classifiers):			
Labor	0.006		
Supplies	0.001		
Power	0.002	0.009	
Tube Milling (Two 5-ft. by 18-ft.):			
Labor	0.011		
Supplies	0.049		
Power	0.092	0.152	
Agitation (Three 32-ft. by 4-ft. Dorrs):			
Labor	0.028		
Supplies	0.006		
Power	0.027	0.061	
Chemicals:			
Cyanide	0.218		
Lime	0.293		
Lead acetate	0.024	0.535	
Continuous Decantation (Six 35-ft. by 12-ft. Dorrs):			
Labor	0.004		
Power	0.017	0.021	
Filtering and Discharging (Six 14-ft. by 12-ft. Olivers):			
Labor	0.062		
Supplies	0.045		
Power	0.088	0.195	
Precipitation:			
Labor	0.025		
Supplies	0.067		
Power	0.019	0.111	
Return Pumping:			
Labor	0.009		
Power	0.012	0.021	
Refining:			
Labor	0.023		
Supplies	0.021		
Power	0.001	0.045	
Assaying and Sampling:			
Labor	0.018		
Supplies	0.008	0.026	
Superintendent and Foremen		0.090	
Experiments		0.010	
General Expense		0.043	
Water Supply		0.021	
Grand Total Milling			$1.586
Grand Total Mining			0.957
Grand Total Mining and Milling			$2.55

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TABLE 2.—*Milling Data*

	Figures Obtained by Dehydrating Samples		Figures Obtained by Drying Below 110° C.		Comparative Figure for Quarts, 2.7 Specific Gravity
	H_2O, Per Cent.	2.4 Specific Gravity, Dry Tons	Moisture, Per Cent.	1.9 Specific Gravity, Dry Tons	
Ore milled per month, 10,000 gross tons	19	8,100	4	9,600	2 per cent. H_2O
Ore milled per day, 333 gross tons	19	270	4	320	2 per cent. H_2O
Ball mill discharge, specific gravity, 1.439	48	270	36	320	
Tube mill discharge, specific gravity, 1.394	52	234	40	279	Specific gravity, 1 64; moisture, 38 per cent.
Akins classifier, sand product	38	175	25	208	Moisture, 24 per cent.
Slime to treatment plant, specific gravity, 1.045	1 to 11.5	270	1 to 10	320	1 to 7, specific gravity, 1.08
Primary thickeners underflow, specific gravity, 1.15; settling area, 8 sq. ft. per ton	1 to 3.5	270	1 to 2.57	320	1 to 2, specific gravity, 1.26 or better.
Secondary thickener underflow, specific gravity, 1.23; settling area, 20 sq. ft. per ton	1 to 2.1	270	1 to 1.5	320	1 to 1, specific gravity, 1.46 or better.
Filter cake	43	270	30	320	Moisture, 29 per cent.

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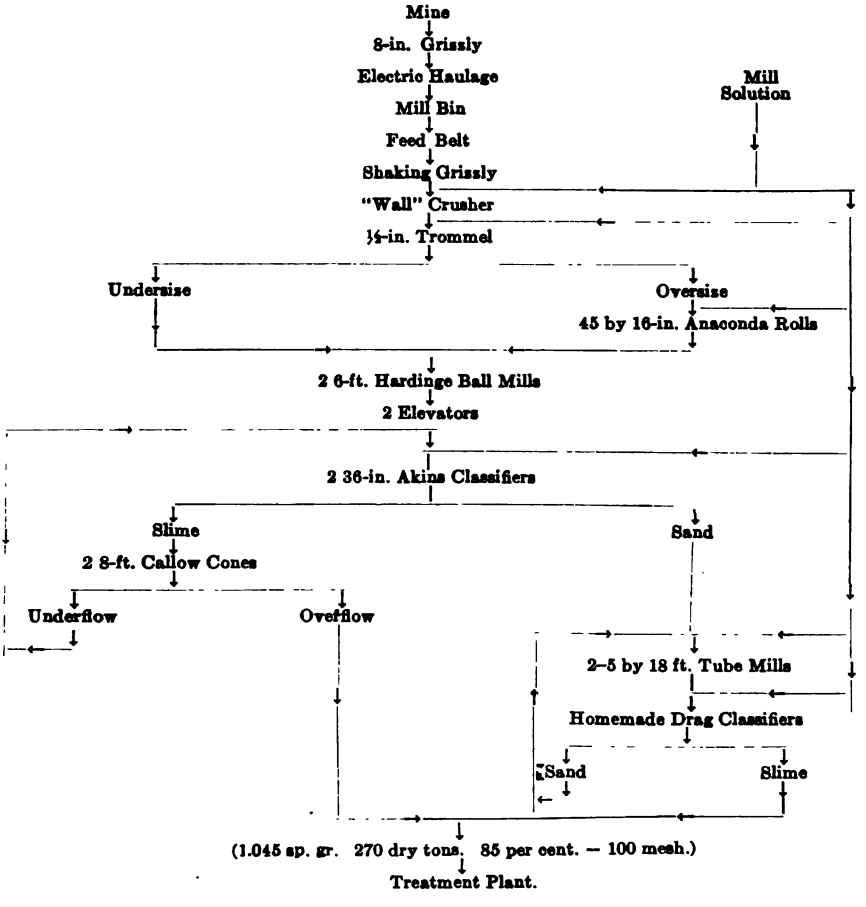
TABLE 3.—*Flow Sheet.*



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Possibilities in the Wet Treatment of Copper Concentrates

BY LAWRENCE ADDICKS,* B. S., NEW YORK, N. Y.

(Arizona Meeting, September, 1916)

AT the San Francisco meeting of the Institute last year, I presented, through the courtesy of Dr. James Douglas, some results of experiments on the roasting and leaching of concentrator tailings. After it became apparent that flotation rather than leaching was clearly the better method of handling the particular problem under consideration, before dismantling the experimental equipment, some data were secured on roasting and leaching the concentrate itself in competition with smelting. Some of the results obtained are given in the following paper and they are of particular interest at this time in view of the large quantities of flotation concentrates with their somewhat difficult smelting characteristics now being produced.

A complete wet process† consists of roasting and leaching the calcines in dilute sulphuric acid produced from the roaster gases, roasting the residue with salt and leaching with dilute tower liquors (the well-known Longmaid-Henderson process) and recovering the copper, silver and gold by cementation or electrolysis or a combination of both. It is evident, however, that the residue from the first leaching, carrying about 20 per cent. of the copper and all of the silver and gold, can be smelted if preferable. In considering the application of the scheme to individual cases, it must be remembered that freight plays a large part in any reduction process wherein smelting is not conducted at the mouth of the mine, and that it is not practicable today to build small smelting plants for individual operations.

The experiments may be grouped under four main heads: Roasting, leaching, chloridizing residue, and recovery of copper from solutions. The products of two concentrators were used: The Nacozari concentrates were the product of a large modern mill not using flotation, the copper mineral being largely chalcopyrite; and the Tyrone concentrates, the product of an experimental mill including flotation, the copper mineral being chiefly chalcocite. Typical analyses would be as follows:

* Consulting Engineer.

† Patent applied for.

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	Nacozari	Tyrone
Copper, per cent	14.0	14.0
Silver, ounces per ton	4.0	0.5
Gold, ounces per ton	0.01	Trace
Iron, per cent	31.0	28.0
Sulphur, per cent	34.0	30.0
Silica, per cent	13.0	
Alumina, per cent	3.0	
Lime, per cent	0.6	

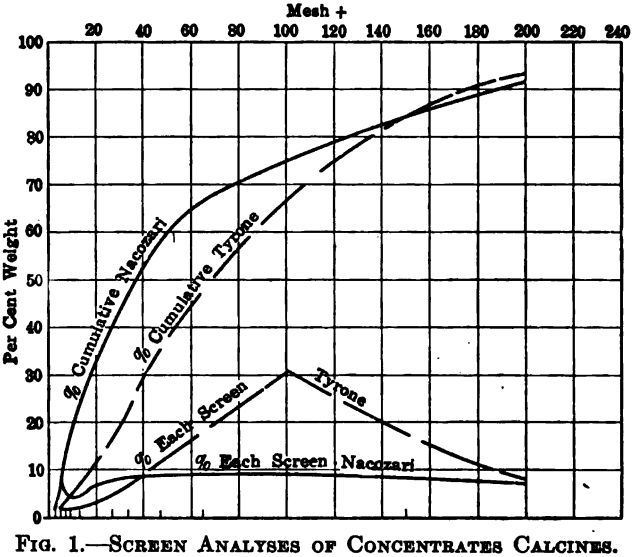

FIG. 1.—SCREEN ANALYSES OF CONCENTRATES CALCINES.

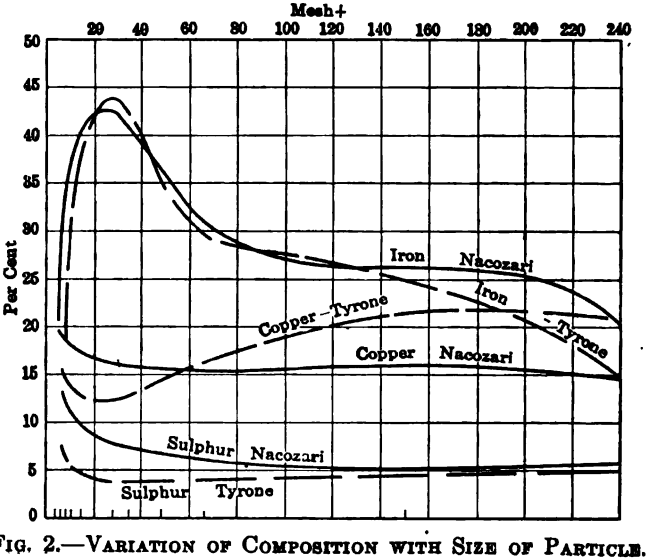

FIG. 2.—VARIATION OF COMPOSITION WITH SIZE OF PARTICLE.

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Fig. 1 shows screen analyses of calcines obtained by dead roasting the two concentrates as described later. Fig. 2 shows the analysis of the calcines for copper, sulphur and iron for each screen size. The Nacozari concentrates carry considerable coarse jig product while the Tyrone material comes from an ore where the values are finely disseminated and the quantity of 100-mesh is quite marked. The presence of the flotation concentrates in the Tyrone material brings up the copper contents of the fine sizes.

Roasting

The object in roasting is to make as much of the copper and as little of the iron as possible soluble in dilute sulphuric acid. The work is similar to roasting pyrites fines in sulphuric acid manufacture, except that this solubility ratio rather than the complete utilization of sulphur is the

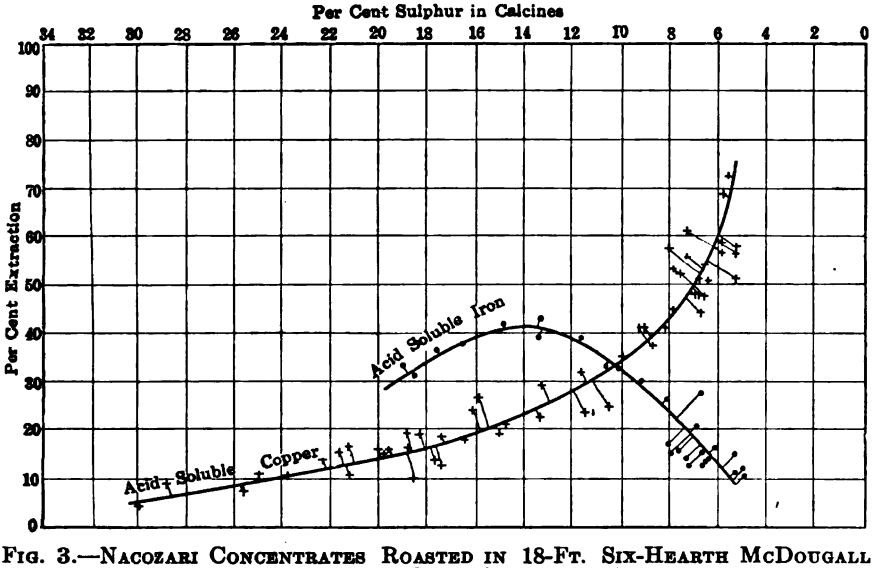

FIG. 3.—NACOZARI CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH MCDOUGALL FURNACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

controlling factor. Small-scale work is not very satisfactory as a guide to possible results as it is practically impossible to prevent overheating due to rapid oxidation of sulphur in a laboratory experiment. An 18-ft., water-cooled, six-hearth McDougall was used, the speed of rotation being gradually cut down until dead roasting conditions were obtained. Greater tonnages could doubtless have been obtained in a seven-hearth furnace.

Many samples were taken from various hearths and the acid-soluble copper and iron determined by leaching with 4 per cent. H_2SO_4 in the laboratory. The results of these tests are given in Figs. 3 and 4. The

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results of tests on a series of sixth-hearth samples to determine the relation between tonnage and sulphur elimination are plotted in Fig. 5. It is

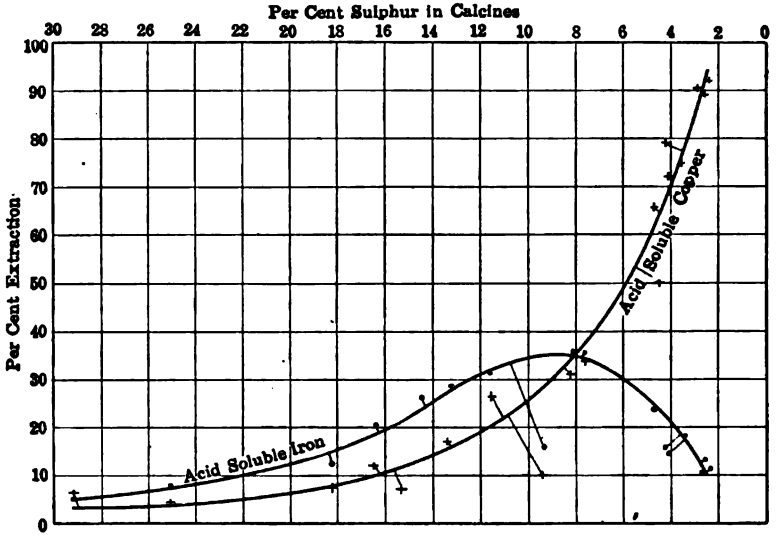

FIG. 4.—TYRONE CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH MCDOUGALL FURNACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

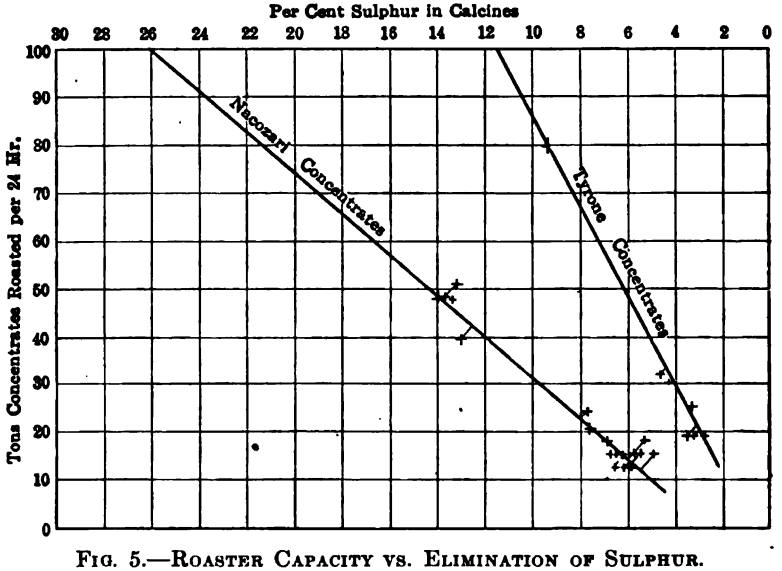

FIG. 5.—ROASTER CAPACITY VS. ELIMINATION OF SULPHUR.

evident that the chalcocite can be oxidized much more readily than the chalcopyrite, although size of particles has something to do with this. An investigation of the solubilities of the various sizes of particle was carried

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out by screening some of the calcines, as shown in Fig. 6. As would be expected, the finer particles are the more thoroughly oxidized; the jig product in the Nacozari concentrate is one reason for the poorer results obtained in the treatment of this material.

In general, these large-scale experiments indicate the possibility of reasonably obtaining the results desired—high copper and low iron solubility—but it is obvious that the residue after leaching will contain sufficient copper values to require retreatment, aside from the fact that any silver and gold will remain in this residue.

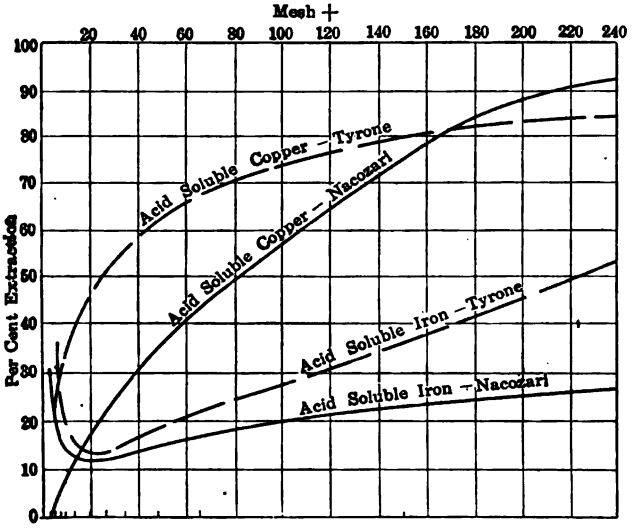

FIG. 6.—RELATIVE SOLUBILITIES OF VARIOUS SIZED PARTICLES OF CALCINE.

Leaching

As shown in the paper presented last year, such satisfactory results in the extraction of copper values from tailings were obtained by dumping the hot calcines from the furnace into a leaching trough, the few seconds' agitation thus obtained extracting almost as much as prolonged treatment in other apparatus, that the same idea was tried out with the concentrate calcines. It was not possible, for various reasons, to handle the output of the furnace directly, so the calcines were stored and then fed to a bucket elevator which in turn delivered into a V-trough in which the leaching liquor was flowing. The results were here disappointing, as although there was instant extraction of perhaps half of the soluble copper, a prolongation of the trough to give 60 sec. travel did not greatly increase this amount. It was definitely shown in the laboratory as well that prolonged agitation was necessary to extract all of the soluble copper. The

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leaching trough delivered into an acid-proof drag consisting of an endless belt with angle rakes, of the type commonly used in concentrators for dewatering. This acted more or less as a classifier, the very fine residues being carried over with the liquor, from which they were subsequently separated by settling. As this still gave insufficient agitation to the sands, a Parral tank was tried, but it was found that they were too heavy to yield readily to any sort of air-lift agitation. A Dorr classifier was then added to the apparatus and this did better. It was found, however, that it was necessary to pass the residues six or seven times through the leaching process in order to obtain an extraction equal to that shown by

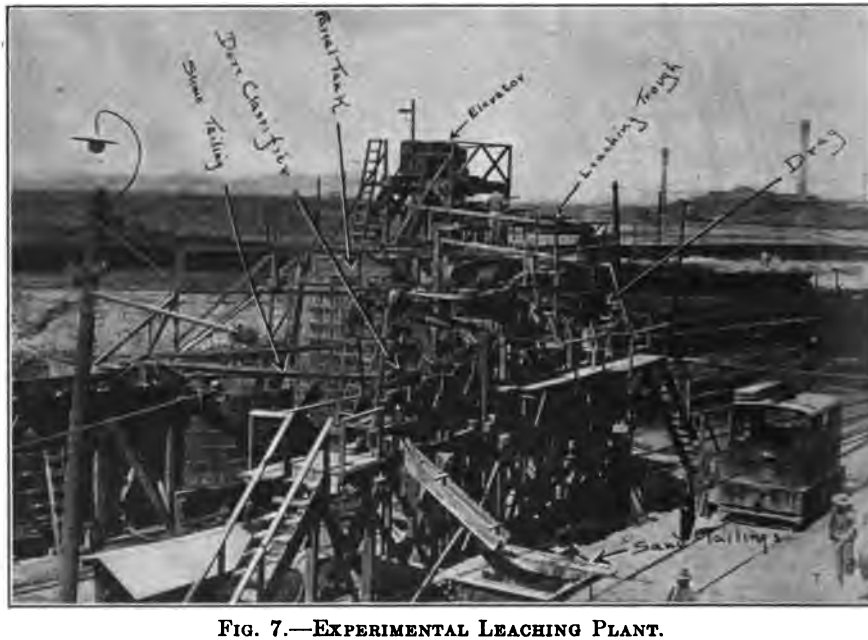

FIG. 7.—EXPERIMENTAL LEACHING PLANT.

laboratory tests on the calcines. Fig. 7 gives a general view of the leaching apparatus.

The large-scale leaching tests were confined to the Tyrone material, a lot of 30 tons of calcines from some of the roasting tests being used. The first runs on a lot of 17 tons of not quite dead-roasted material running 4 per cent. sulphur gave results that were satisfactory except in that too much iron was dissolved, causing a needless consumption of acid and embarrassing any electrolytic scheme of recovery. Later, better-roasted material was available and a careful record kept of the metal balance and acid consumption, with the results shown in Table 1.

These figures check reasonably close by except in the case of iron; but it must be remembered that various iron parts in the apparatus used were

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TABLE I.

Time Through	Per Cent. Cu in Tails	Trough Fed	Dorr Fed
1st	7.7	Acid	Water
2d	6.0	Acid	Water
3d	5.2	Acid	Water
4th	3.6	Acid	Acid
5th	3.3	Acid	Water

Extraction by Heads vs. Tails

	Weight, Pounds	Copper		Iron		Alumina	
		Per Cent.	Pounds	Per Cent.	Pounds	Per Cent.	Pounds
Heads	8,360	15.48	1,292	31.00	2,590	5.60	468
Tails	5,600	3.50	196	43.52	2,440	7.02	393
Extraction	.....	84.70	1,096	5.80	150	16.00	75
Extraction per lb. of Cu.	.....	...	1.00	...	0.14	...	0.07

Extraction by Analysis of Liquors

	Weight, Pounds	Copper		Iron		Alumina	
		Per Cent.	Pounds	Per Cent.	Pounds	Per Cent.	Pounds
Heads	51,538	0.46	238	0.20	104	0.47	246
Tails	86,910	1.46	1,271	0.76	657	0.45	395
Extraction	.....	80.00	1,033	21.30	553	31.90	149
Extraction per lb. of Cu.	.....	...	1.00	...	0.53	...	0.14

attacked by the liquor, which would artificially increase the iron taken into solution.

The acid consumption was 2,495 lb. of 100 per cent. H_2SO_4 for the run. This is equivalent to 2.28 lb. per pound of copper extracted. Laboratory tests on the same calcines indicated 2.0 lb. The leaching was done at about 125° F. with 5.6 per cent. free acid in the liquor entering the trough.

In general, when a 15 per cent. copper calcine is fed to the trough, the residue at the end of the trough will run about 8 per cent. Cu, the extraction representing the instantaneously soluble copper. This residue can be brought down to about 3.5 per cent. Cu by suitable agitating means, with a consumption of a little over 2 lb. of acid per pound of copper, and with the extraction of but little iron. The final residue weighs only about 60 per cent. of the original concentrate before roasting.

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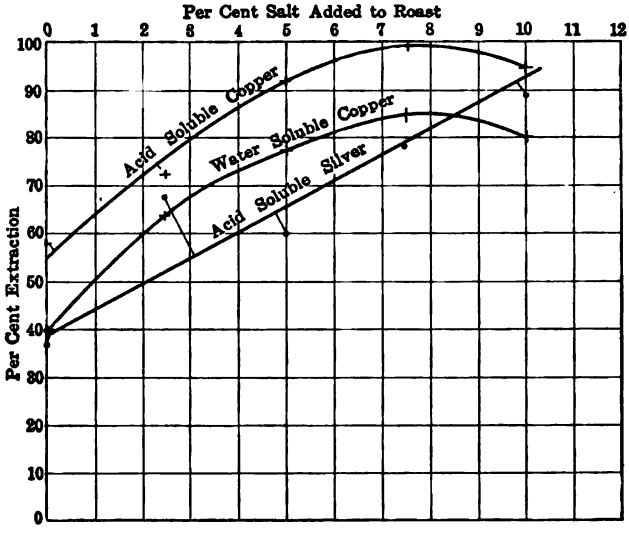

FIG. 8.—CHLORIDIZING LEACHED CONCENTRATES CALCINES.

Roasted 1½ Hr. at 975° F. with Addition of Salt and 10 Per Cent. Raw Concentrates
Calcines: 5.6 Per Cent. Copper, 1.9 Oz. Silver and 2.5 Per Cent. Sulphur.
Raw Concentrates: 14.4 Per Cent. Copper, 0.55 Oz. Silver and 34 Per Cent Sulphur.
Liquor: 5 Per Cent. Na_2SO_4, 5 Per Cent. Na Cl, 5 Per Cent. Fe Cl_2 and 0.5 Per Cent. $HCl + H_2SO_4$.

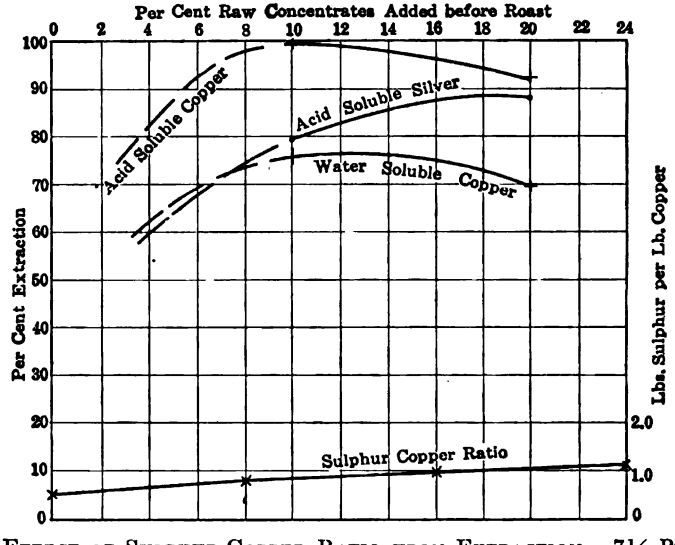

FIG. 9.—EFFECT OF SULPHUR-COPPER RATIO UPON EXTRACTION. 7½ PER CENT. SALT ADDED TO ROASTING "MIX".

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Chloridizing Residue

No large-scale work was done on the chloridizing of the residues from the first leaching. The analysis of these residues, however, differs from that of pyrites cinder, so long successfully treated by this process, only in the amount of silica present. Various small-scale experiments were tried and 50 lb. or so were sent for test to a plant where the Longmaid-Henderson process was in operation. Both sets of experiments were entirely satisfactory.

A small lot of leached residues was prepared for test. These contained 5.6 per cent. Cu, 1.9 oz. Ag, and 2.5 per cent. S. Raw concentrates for adjusting the sulphur-copper ratio were used, containing 14.4 per cent. Cu, 0.55 oz. Ag, and 34 per cent. S. Fig. 8 shows the extractions with varying percentages of common salt added to the "mix" after roasting in an electric muffle furnace 1½ hr. at 975° F. and leaching in a liquor carrying 5 per cent. Na_2SO_4, 5 per cent. NaCl, 5 per cent. $FeCl_2$, and 0.5 per cent. $HCl + H_2SO_4$. Fig. 9 shows the effect of varying the sulphur ratio. The results show a 99 per cent. copper and a 79 per cent. silver extraction. The report on the lot of residues sent away fully confirmed these results.

Recovery of Copper from Solutions

The liquor from the chloridizing plant would doubtless be reduced to argentiferous copper cement by iron. But 20 per cent. of the original copper is involved. The sulphate liquor from the first leach could be precipitated on iron if desired, or with certain limitations would be suitable for electrolytic deposition of the copper and regeneration of the acid. Sulphide concentrates carry from 1 to 2 lb. of sulphur per pound of copper, equivalent to from 3 to 6 lb. of 100 per cent. H_2SO_4, less process losses, if the roaster gas is oxidized to sulphuric acid. Since the leaching calls for but a little over 2 lb. of acid per pound of copper, plus tailings losses, it would seem possible, therefore, to figure on a simple cementation plant, considering electrolysis as a competitor on a basis of relative profit and not of necessity.

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Smelting at the Arizona Copper Co.'s Works

BY F. N. FLYNN,* CLIFTON, ARIZ.

(Arizona Meeting, September, 1916)

Introductory

IN 1882, The Arizona Copper Co. Ltd., acquired producing copper mines at Metcalf and Morenci (locally called Longfellow). Metcalf is situated a distance of 7 miles, and Morenci a distance of 6 miles from the general office of the company at Clifton, Ariz. A reduction works with five blast furnaces was built in the center of the town of Clifton in 1884 and operated until January, 1914.

Concentration of oxidized ores was begun in 1890–1891, in a mill built adjoining the smelter, and in October, 1893, a leaching process was started for treating tailings from the oxide mill. A sulphide concentrator with 300 tons capacity was built at Clifton and started treating chalcocite ores in July, 1896. This was the first mill to treat low-grade copper-sulphide ores successfully. Another concentration mill for sulphide ores was erected at Morenci in 1900. The sulphide mills are now being remodeled to treat ores by concentration and flotation, with a capacity of 4,000 tons at Morenci and 500 tons at Clifton. The leaching plant has not been running since September, 1914.

The average yearly production from this company's mines and reduction works from 1885 to date has been:

	Pounds
10 years ended 1894	6,864,902
10 years ended 1904	20,439,614
10 years ended 1914	32,665,905
Part of year 1915, to Sept. 12	30,206,106
4 months of 1916, to June	15,964,840
June, 1916	4,900,000

Construction on a new smelting plant was started in February, 1912, and finished in February, 1914. In the *Transactions*, vol. 49, E. Horton Jones published "Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd.," which fully describes the construction of this plant, with details of costs.

* Superintendent of Smelting Department, The Arizona Copper Co., Ltd.

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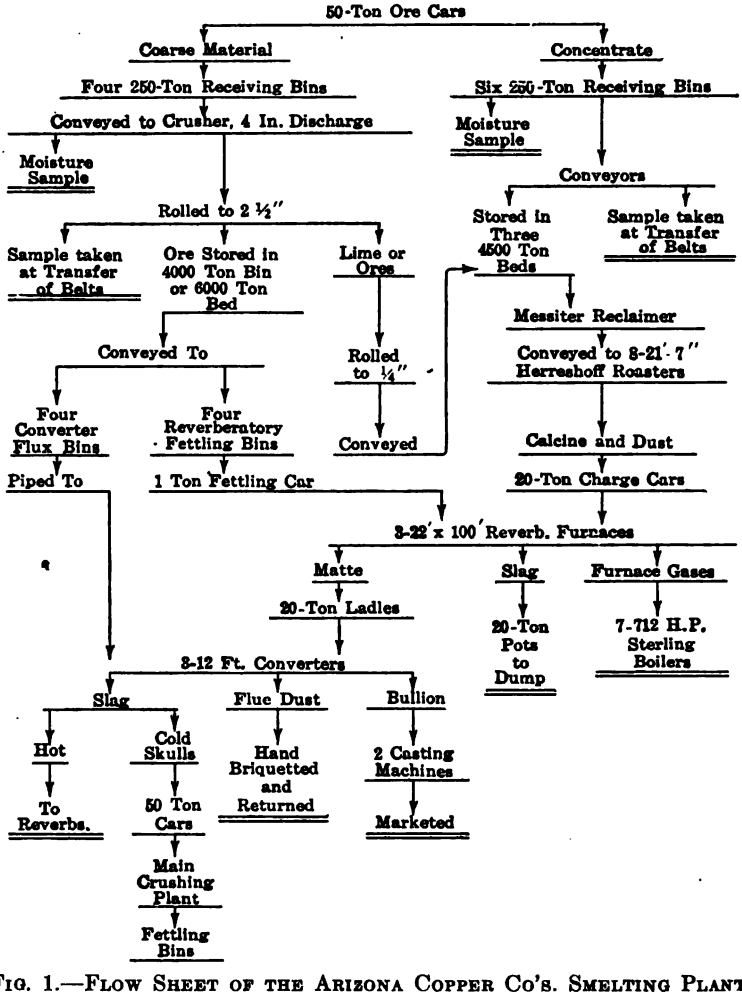
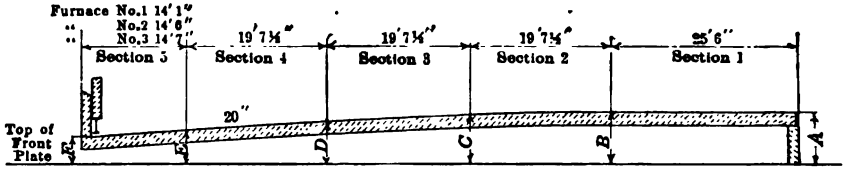


FIG. 1.—FLOW SHEET OF THE ARIZONA COPPER CO'S. SMELTING PLANT.



	Reverb. No.1	No.2	No.3		Reverb. No.1	No.2	No.3
A	7'8⅝"	6'10½"	7'10"	*D*	6'4½"	6'7¾"	6'0¾"
B	7'10⅜"	6'11½"	7'4"	*E*	5'4"	5'6⅜"	5'6"
C	7'0⅜"	6'9⅜"	6'10⅝"	*F*	4'0⅛"	5'0½"	4'8⅜"

FIG. 2.—SKETCH SHOWING POSITION OF REVERBERATORY ARCHES ON CENTER LINE

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The new smelter site is on gently sloping ground, 2 miles below the town of Clifton, and close to the San Francisco River. The Arizona & New Mexico standard-gage railroad and the Coronado narrow-gage railway serve the plant. Smelter yard standard-gage trackage consists of 2.38 miles for steam locomotives, and 1.72 miles for electric haulage.

Reference to the accompanying flow sheet will assist in following description of smelting practice at the new smelter.

Roaster Division

Roaster equipment consists of eight Herreshoff furnaces with steel shells 21 ft. 7 in. in outside diameter. Hearths and linings in the different furnaces are built of materials as shown in the accompanying table.

Hearth and Lining Materials in Herreshoff Roasting Furnaces

Furnace No.	Drier	Hearth Numbers From Top Downward						Shell
		1	2	3	4	5	6	
1	Red brick	Firebrick	Firebrick	Gravel-concrete	Gravel-concrete	Gravel-concrete	2-in. concrete	Red brick
2	Red brick	Firebrick shapes	Slag-concrete	Firebrick shapes	Firebrick shapes	Firebrick shapes	2-in. concrete	Red brick
3	Red brick	Firebrick	Slag-concrete	Firebrick	Slag-concrete	Brick-concrete	2-in. concrete	Red brick
4	Red brick	Firebrick	Firebrick	Gravel-concrete	Gravel-concrete	Gravel-concrete	2-in. concrete	Red brick
5	Red brick	Firebrick	Firebrick	Firebrick	Firebrick	Firebrick	2-in. concrete	Firebrick
6	Red brick	Firebrick	Firebrick	Firebrick	Slag-concrete	Slag-concrete	2-in. concrete	Red brick
7	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	2-in. concrete	Red brick
8	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	2-in. concrete	Red brick

Hearths designated "gravel concrete" were made of river gravel, 3-in. size and under, 2 parts; clean sand, 4 parts; and cement, 1 part. Slag-concrete mixture consisted of crushed slag (all the size that passed a ½-in. screen) 3 parts; clean sand, 3 parts; and cement, 1 part. Composition of this slag was: SiO_2, 47.0 per cent.; Al_2O_3, 8.5 per cent.; Fe, 27.5 per cent.; CaO, 3.3 per cent.; S, 0.5 per cent. In the fifth hearth of No. 3 furnace, crushed red brick was substituted for gravel in the concrete mixture.

The first nine concrete hearths built were reinforced with iron rods laid in top and bottom courses. Reinforcing in those with inner drop holes weighed 3,093 lb., and those with outer drops 2,148 lb. When fully heated the fourth and fifth hearths, with heavy reinforcement, rose 6 in. at the center of the arch, which necessitated chipping off some of the concrete from the center of hearths. In later construction ½-in. diameter reinforcing rods were used. This reduced the weight of iron to 1,172 lb. and 1,373 lb. respectively for inner and outer drop hearths.

Slag-concrete hearths are preferred, providing the slag is highly

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siliceous. All concrete hearths have given entire satisfaction. They have been in service since May 3, 1914, without any repairs.

Provision is made for cooling the central columns and rabble arms with air supplied from a motor-driven fan, but during the last 18 months cooling air has not been required.

The roaster charge has varied from ore, limerock and concentrate mixtures with 19.2 per cent. sulphur and 10.7 per cent. copper, to all concentrates with 30.7 per cent. sulphur and 15.8 per cent. copper, with fuel requirements varying from a maximum of 40 lb. of coal per ton of calcines to none when roasting all concentrate. The units of sulphur burned off have ranged from 7.5 to 17.4, the usual practice being to leave equal quantities of sulphur and copper in the calcine, because of the oxidized ore used in reverberatory fettling.

Normally, the material roasted requires only a little more heat than is derived from the sulphur burned. Heavy crude oil was first used as fuel, but with poor results, because with the low draft maintained burners could not be obtained of suitable size to burn continuously the small quantity of oil necessary to furnish this deficiency of heat, and distribute it properly over the hearth. The flame was admitted through one door on the third or fourth hearth from the top. A burner that would stay lighted in these comparatively cold hearths, produced an intense flame and if kept running long enough to ignite the charge to the temperature desired, the brick hearth over the burner became too hot and charge fused on the hearth directly in front of the burners. Calcine produced was not uniform, because the sulphur had to be controlled by burning oil intermittently, which resulted in first a hot furnace and low sulphur, then a cold furnace with high sulphur.

Coal-fired Dutch-ovens with 12 sq. ft. of grate area have been adopted for supplying additional fuel required for roasting. Two ovens for each furnace admit the flame on the fifth hearth from the top. With these the roast can be quickly and closely regulated, and the furnace repairs have been greatly reduced as compared with oil-burning conditions.

Ignition usually takes place on the fourth hearth from the top, and the temperature of the charge rises to 1,200 or 1,300° F. on the two lower hearths. With one reverberatory running, each roaster operated normally makes 65 tons of calcine per day. When two reverberatories are taking charge, each roaster has to supply 80 tons of calcine. Results show that while removing the same units of sulphur per ton, a furnace will consume less coal per ton when producing 60 than when producing 80 tons of calcine. Monthly averages of 113 tons of calcine produced per furnace day show a very high coal ratio. Furnace dampers are kept closed just to the point of smoking. Gases leaving the furnaces first enter a hopper-bottomed header-flue, through which they travel at a velocity of 6 or 7 ft. per second, at 325 to 425° F. tempera

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hen pass into a large dust-chamber. Dust recovered from er amounts to 0.8 per cent. of the dry charge to roasters. st-chamber roof, made of No. 11 plate steel, shows slight on at the laps. A sheet-copper cap laid over the joint at apex ed 2 years and has been replaced by concrete.

Reverberatory Division

g for production of matte is all done in reverberatories, either o furnaces being operated as required. Three furnaces set each other the long way, and 26 ft. apart, each with a hearth 100 ft., are available, one of these always being a spare. The No. 3 furnace is crushed silica fused in on a slag bottom. os. 1 and 2 have hearths of crushed silica fused in on bottoms quartz. Roofs were all originally built with 1 in. to the foot arch. When the roof of No. 1 furnace was rebuilt, the arch a rise of 1¼ in. to the foot. Six-inch holes, spaced 24 in. t in the roofs along the side walls and across the back, allow terial to be piped into the furnaces from trough-shaped bins cally over these holes. Five-inch diameter pipes on fettling ing changed to 8-in., to permit using coarser material without e pipe openings. Other dimensions of these reverberatories their operation are given in an accompanying table.

Reverberatory Dimensions

	Furnace Number		
	1	2	3
e tap hole to top of front plate...	1 ft. 10⅝ in.	.....	1 ft. 3⅞ in.
ole to skimming block, in use at	10⅞ in.	.....	1 ft. 1 in.
burners above front plate.....	1 ft. 11⅛ in.	.....	1 ft. 8⅜ in.
l burners above skim block at se.....	1 ft. 11⅛ in.	.....	1 ft. 5⅞ in.
at area when slag line is level with plate, square feet.....	17.4	26.2	22.0
at area, when slag line is raised top of front plate, square feet...	13.2	22.6	18.0
ng in verb shaft, square feet....	40.0	28.5	28.5
e opening to header flue, square	45.6	55.2	54.6
side walls, as rebuilt, above matte	30.0	30.0	12.0

The 12-in. side wall is more favorable for greater height of fettling.

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In furnaces Nos. 2 and 3 the weight of the shaft has been taken off of the verb arch, by means of a telescopic shaft supported on steelwork. Fig. 2 shows the relative positions of the reverberatory arches on the center line of roof. This height is satisfactory when burning small quantities of oil, but might be raised considerably in Sections No. 1 and 2 for larger quantities of oil.

The following log of the third campaign of No. 3 reverberatory furnace is representative of operations on a one-furnace schedule:

Furnace started May 20, 1914.

6.208 days lost in August, repairing side walls, verb shaft, and 24 ft. of new roof on Section 1.

0.769 days lost in September, October, November and December, 1914, making minor repairs.

1.122 days lost in January, 1915, patching roof and front end.

2.736 days lost in March, 1915. New roof on Section 1.

5.642 days lost in July, 1915. 50 ft. roof repairs on Sections 1 and 2, and new jambs under header-flue arch.

Furnace stopped Sept. 12, 1915, on account of labor strike, but was in good repair at that time.

16.477 days lost time in 16 months total campaign.

Dry Tons Solid Charge Smelted:

During total campaign	178,165.0
Daily average for total campaign	384.6
Daily average for one month, highest	512.4
Daily average for one month, lowest	312.2
Highest for one day, during highest average month	543.0
Lowest for one day, during highest average month	413.0

Barrels of Oil per Ton of Solid Charge:

Lowest month	0.653
Highest month	0.951
Average during campaign	0.744

By a campaign is meant the elapsed time between starting a furnace and tapping out the matte. New roofs on Sections 1 and 2 were built upon ore centers, without tapping the matte. The last side walls repaired were in November, 1914. Since adopting greater fettling height, no further wall repairs are anticipated.

Two parallel charge tracks cross the furnaces near the back in a direction at right angles to their length. Fifteen-ton charges of calcine, at a temperature of 1,050 to 1,150° F., are dropped through one water-jacketed charge hole in the center of the roof on either track, but 90 per cent. of all charges are dropped from the back track. This part of the charge dropped through the roof is designated "direct charge." Seventy-five per cent. of the total solid charge smelted is introduced in this way.

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nace conditions are always expected if such cold material as slag or limerock of any size, or anything coarse like siliceous erated with converter slag, either cold or warm, is dropped as ge without mixing it well with hot calcine. Ore, limerock or duces "charge floaters." Cold slag sinks and makes sticky toms. When it is necessary to add cold or coarse material ct charge, the best results are obtained when they are well hot calcines, in the calcine charge cars, but better still in the rnaces.

xtures to be roasted are made of the required composition to slag, when smelted, in which the ratio of oxygen in acids to ases will fall between 2.00 and 2.25. A vast difference, how- nd in the smelting properties of calcines made from these egardless of the similarity between their analyses or the the slags they produce. Calcine made from concentrate or es of ½ in. size smelts readily, whereas if the calcine is an arti- re made of siliceous oxidized ore crushed to this size, and mixed cessive proportion of limerock, the charges melt more slowly. from the two following mixtures will fairly illustrate this

Class of Material	Mix 103. Per Cent. of Total	Mix 120. Per Cent. of Total
entrate	100.0	70.7
low sulphur	.....	4.8
e ore	.....	12.9
	.....	11.6
	100.0	100.0

	Analysis						
	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Cu, Per Cent.	Oxygen Ratio
	18.3	4.6	28.2	0.7	29.0	14.10	1.43
	20.7	4.7	24.1	7.6	22.5	10.98	1.45

average of 365 tons of calcine per furnace was smelted during at mix 103 was on the charge. In comparison, 200 tons per was barely maintained while the charge consisted of calcine 20. The practice is to crush siliceous ore as fine as $\frac{3}{16}$-in. beds, and endeavor to keep the sulphide material as high as

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75 per cent. of the total bed mixture. Ore and limerock mixtures require a much higher oil ratio than calcine from straight concentrate.

In other words, the greatest percentage of oxidized material which it is permissible to smelt in the reverberatory furnace is found to be:

	Tons per Furnace Day	Per Cent.	
Direct charge	300	25	Oxidized ore and limerock.
Fettling	100	100	Oxidized ore.
Total	400	44	Oxidized ore equivalent.

Above 44 per cent. oxidized ore, the charge could undoubtedly be better handled in a blast furnace. The charges spread better when the furnace is nearly full of matte, and also when the copper and sulphur in the calcine are equal. Matte under 40 per cent. copper is favorable for higher tonnage.

Fettling.—Ore and byproducts crushed to a 2½-in. size for fettling constitute the remaining 25 per cent. of the solid charge smelted. It is customary after each skim to feed these materials from bridge-wall to shaft in the following proportions:

Furnace	Section No.	Per Cent.	Material
Back	1	25	Old smelter slag. Converter slag skulls.
Back	2	50	Less siliceous ore.
Middle	3	12	Less siliceous ore.
Middle	4	8	Less siliceous ore.
Front	5	4	More siliceous ore.
Shaft	..	1	Silica.

	Analysis							Oxygen Ratio
	SiO_2, Per Cent.	Al_2O_3 Per Cent.	Fe, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	S, Per Cent.	Cu, Per Cent.	
Converter slag skulls	19.6	3.9	47.2	0.9	0.0	3.6	9.00	0.89
Old smelter slag	37.2	8.9	22.0	10.3	5.2	0.3	3.56	2.12
Less siliceous ore	42.6	7.7	18.0	3.0	2.5	1.0	6.80	3.74
More siliceous ore	65.0	12.2	5.6	0.8	0.6	1.9	3.78	19.43
Silica	84.4	6.3	3.1	0.5	0.0	2.6	0.74	46.34
Approximate average	40.4	7.5	21.5	3.5	2.4	1.3	6.49	3.08

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July, 1914, the height of fettling was carried not to exceed
the slag line. This height was gradually increased until
14, when the practice of fettling to the holes in the roof was
and later successfully adopted after the size of fettling
s increased from ¾ in. to the present 2½-in. size. The coarse
better. It reposes at an angle of 45° above and 45° to 60°
ag line.

ılts achieved in performing fettling operations as practised
l upon the relative location and manner in which the material
as well as the amount of fines in it and its chemical and
nposition, as may be concluded from the following account of
vith different fettling materials.

ınd byproducts crushed fine work unsatisfactorily in any part
ce. However, if crushed coarse, results are good when such
charged in the back of the furnace.

siliceous fine ore is not good fettling in any place in the
does not stand, and requires care to keep it from running out
nace. Crushed coarse, it makes good fettling wherever used.

siliceous fine ore mixed with coarse byproduct make a bad
ı. The only place the more siliceous fine ore alone can be
ny degree of success is on the bridge-wall.

siliceous coarse ore is not so acceptable as less siliceous coarse
in the front half of the furnace, because it makes too much
m on top of the slag.

e raw concentrate is good in the back half of the furnace,
t stand so well nor so high as coarser material. It gives bad
e front half because it melts easily to a magnetic mush.

ng with hot calcine can only be done with any degree of
ection No. 1, and then only slightly above the slag line. At
on it runs out over the furnace bottom. An attempt was
tle the entire furnace with calcine, but was abandoned when
dropped to less than half capacity. Calcine gave poorer
any other material experimented with.

e in the front half melts easily to a magnetic mush. In the
hen mixed with siliceous oxide ore it works well.

deal fettling material was found to be less siliceous copper-
containing 5 to 10 per cent. sulphur.

words, basic or sulphide materials are charged near the
ecause of their lower fusing points. If charged in the front
ere the temperature is lower, they melt with sufficient ease
hick, mushy, semifused slag, and in the case of pyrite make
f magnetite. Siliceous ores with 60 per cent. silica produce
ch blankets the slag and causes higher oil ratio and lower
f the siliceous ores contain much copper in oxidized form,

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the scum will assay from 0.7 to 1.3 per cent. copper, whereas if the coppe is present as a sulphide the scum will assay very much lower.

The most important development as regards fettling has been alon the line of coarser crushing and greater height of material to protec the side walls without encroaching on the furnace bottom area. Fettlin "floaters" are unknown. The successful results of this practice are bes seen from the photograph of one of the furnaces (Fig. 3).

Fuel is 14 Bé., 18,000 B.t.u., California crude oil, pumped at 12 lb. pressure and delivered through steam heaters at a temperature o 170° F. to six burners and atomized with converter air. Two hundre

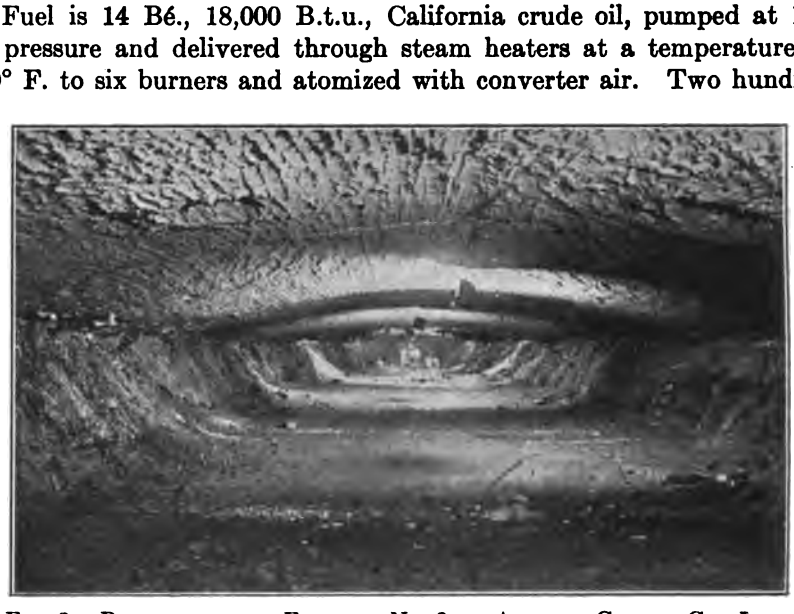

FIG. 3.—REVERBERATORY FURNACE NO. 3 OF ARIZONA COPPER CO., LTD.

Campaign No. 3. Started May 20, 1914, finished Sept. 12, 1915 (date of labo strike.)

Dry tons of solid charge smelted: total 178,165; highest daily average for on month, 512.4.

Fettled to the roof with "less siliceous" oxide ore and converter slag skull crushed to 1½ in. Slope of fettling, from 45 to 60°. Fettling material amounts to 2 per cent. of total solid charge.

and forty to 365 bbl. of oil is burned per furnace-day. This variatio covers a one-furnace schedule when hard firing for high tonnage is re quired, or two furnaces with light firing and correspondingly low tonnag smelted.

Furnace-throat draft (inside the furnace) is 0.12 to 0.16 in. of wate and in header-flue common to all furnaces, draft is 0.50 in. water. Gase leave the furnaces at temperatures from 1,800 to 2,000° F. Seven Stirlin boilers are available for reverberatory waste heat, each 712-hp. capacity and all connected in multiple through cross-over flues from the heade flue. Dampers are so arranged that gases can be sent to any one or a of the boilers. There is no bypass to the chimney. Four or five boile

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are used when running one furnace. Reverberatory dust caught in all the boilers does not exceed 5 tons monthly when running one reverberatory.

Furnace gases leaving the waste-heat boilers at temperatures varying between 471° and 588° F., are conducted through 6-ft. diameter steel pipes into a flue connecting with the main chimney. The temperature of gases in the flue ranges between 400° and 500° F. at different points, and at these temperatures the average velocity of gases is 10 ft. per second, when running one furnace. An average analysis of this gas is SO_2, 0.3 per cent.; CO_2, 5.9 per cent.; O, 12.9 per cent.; CO, 0.4 per cent.; N, 80.5 per cent.; SO_3, not determined. When the flue was cleaned recently, 100 tons of dust was removed that had accumulated during 2 years' furnace operations. The composition of this dust was:

	SiO_2	Al_2O_3	Total Fe	Ferrous Fe	CaO	S	Cu	Free H_2SO_4	H_2O
Reverberatory chamber dust	17.4	6.3	11.8	....	2.3	10.3	9.65	3.0	9.5
Soluble in water	....	1.8	3.7	2.6	2.1	9.1	6.60		
Per cent. of total, soluble in water	....	28.3	31.4	....	91.4	88.3	68.4		

The flue roof was originally made of No. 14 steel plate. This had to be repaired after 8 months' service. Holes were eaten in the plate and in spots the whole sheet was pitted and nearly eaten through. Moist flue dust covered the inside surface of the roof. This deposit consisted principally of sulphates of iron and copper, as is shown by the following composition calculated from analysis of dry sample:

Insoluble	3.0
$CuSO_4$	7.7
$Al_2(SO_4)_3$	8.0
$FeSO_4$	18.7
$Fe_2(SO_4)_3$	45.3
$ZnSO_4$	3.2
$CaSO_4$	2.9
Free H_2SO_4	12.3
Total	101.1

This would not jar loose from the sheet, and metallic copper had deposited on the iron. Concrete slabs reinforced with wire netting were laid on top of the steel plate and the joints between slabs filled with asphalt. The concrete covering has served 2 years. The corrosive action of gases on the iron was arrested for 10 months of operations after the slab roof

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was put on, but during an idleness of 5 months, due to labor trouble the plate had corroded completely.

Lime in the mortar used in laying tile walls of this flue was partly changed to sulphate by the furnace gases. While the flue was cold, the walls swelled far out of plumb. The tile broke and walls had to be rebuilt. New walls are brick laid in slime mortar.

Converter slag is poured through the back wall of furnace, at the center, through a launder discharging at the height of the oil burners. Magnetite builds up under the discharge of the launder, but the bridge wall on either side of it requires fettling.

Considerable has been written regarding the fluxing value of converter slag poured into reverberatory furnaces. The prevailing opinion seems to be that very little fluxing value is within one's control. It is argued that the basic converter slag does not act upon the charge resting on the slag surface, does not mix with the trisilicate slag, and passes through the furnace like a submarine as a monosilicate, emerging near the skimming end. The technical reason advanced for these arguments is the relatively high specific gravity of the basic converter slag. The writer has proved to his own satisfaction that practically all of the iron present in liquid converter slag, other than iron in the magnetic state, is effective as flux. The violent boiling and the reactions that take place when calcine charge is dropped through the central charge hopper are effective in mixing any slag that may have separated into layers in the furnace at the back. If it is true that the charge dropped in the back of the furnace partly melts to a trisilicate slag, underlaid by the bisilicate, which in turn rests upon the monosilicate converter slag, it would be a difficult task to explain why the surface or trisilicate slag is so active in its consumption of fettling material at the slag line. On the other hand, it is not difficult to imagine the monosilicate slag attacking the fettling material along the side walls, from the matte line up. How it would fail to do so would be difficult to explain.

The effect of liquid converter slag was vividly shown on one occasion when a furnace had been tamped with moist quartz from floor to roof along the walls, and when fused was largely filled with hot converter slag and little regular charge. Within 24 hours there was little quartz remaining.

On several occasions, two furnaces running on identical charges have been used for illustrative purposes. Into one, all of the converter slag was poured. In the other, without converter slag, the full quantity of limerock equivalent to the total fluxing value of the converter slag was required.

There are others who contend that even magnetite in converter slag is acted upon by the furnace matte and becomes effective as flux. Our own experience is that whenever magnetite accumulates in the furnace

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e pulled out with the rabble as mush, the necessary limerock the charge, and plenty of sharp drills made ready at the matte

bsence of an accurate method for the determination of magnetic ron in furnace byproducts is one of the most unfortunate and e conditions with which the metallurgist is confronted, and he elucidation of this obscure subject difficult. Without some quantitatively determining this substance, accurate data as to value can not be obtained from balance sheets even if materials ucts are all weighed, because it is not certain from an analysis nagnetic iron has been changed in passing through the furnace.

Converter Division

roduction of magnetite in the converters is, within reasonable ntrolled by the converting practice. The converter slag poured everberatory furnace should not contain any magnetite visible . The safest practice is to reject the last 2 cu. ft. of slag when The rejected slag with the ladle skull will eventually enter the s fettling material, where its detrimental effect to the furnace arily turned to some good, because of its refractory properties. ncy to stick to the sides when mixed with cold ore is quite when the fettling bank is removed from the walls when the out of service. By this practice, magnetite is made to perform at similar service in the reverberatory furnace to that which s in the protective coating in a basic converter. Unfortunately all portion of the total magnetite production can be so used. metallurgists prefer to make a small quantity of magnetite on erter charge, contending that in that way the coating remains onstant thickness. They accomplish this by blowing a few n the start or finish, either or both, but usually at the start, with e of silica. The writer is of the opinion that by intentionally agnetite with every charge, the quantity made must be very ompared with the small amount which can be made to stick to by so haphazard a practice going on continuously. Our practice the shells when the lines of brickwork can be seen through the ng. And, when a shell is being coated, no attempt is made thing with the matte other than form the coating. When e magnetite and copper mush is dumped on the floor under the r it is charged in small quantities at intervals in other charges. ated, a shell is never blown without a large excess of silica. siliceous ore is used for flux, 350 lb. of 80 per cent. silica is used ate the final slag on the copper finish, to insure the removal of aces of iron from the copper with the more active silica. Since

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this quartz, with any magnetite which it has collected, remains in the converter for the next charge, the magnetite is in contact with the quartz particles from the start of the blow.

The converting division is equipped with five 12-ft. Great Falls-type shells lined with magnesite brick; three converter stands, each operated by an independent motor (the fourth stand is under construction); two 40-ton cranes; two straight-line casting machines, each with 39 copper molds. Two stands are regularly used, for which air at 13 lb. pressure is supplied from power house by two of three Nordberg blowing engines.

When converting matte of over 40 per cent. copper, only a small amount of cold matte shells and cleanings from the floor can be used; but with lower-grade matte, all matte shells, converter flue dust, and any converter byproducts made except converter-slag skulls can be handled in this division.

Molten copper is transferred from converters to casting machines in cast-steel ladles lined with fines screened from ores regularly used as converter fluxes. Bars are cast in molds made of converter copper. A 1¼-in. cast-iron splash plate covers half of the bottom area, and an average of 73 tons of bullion is cast per mold. Bars weigh 240 lb. each and 35 min. is required for casting a charge weighing 7 tons. Considerable chipping of bullion bars is necessary to remove edges and fine shot due to blowing to gas finish of 99.60 per cent. copper.

Comparison of One-Month Periods with One and Two Reverberatory Furnaces in Operation

Roaster Division	One Reverberatory	Two Reverberatories
Total tons dry charge roasted	14,257.0	17,235.0
Dry tons charge per furnace-day running time	78.4	73.9
Analysis of charge:		
SiO_2, per cent	17.3	19.8
Al_2O_3, per cent	4.7	4.5
Fe, per cent	28.2	24.3
CaO, per cent	1.6	6.4
S, per cent	28.8	24.5
Cu, per cent	13.61	12.08
Oxygen ratio	1.33	1.44
Analysis of calcine:		
SiO_2, per cent	20.0	22.0
S, per cent	13.8	13.8
Cu, per cent	15.70	13.44
Weight of calcine produced as per cent. of dry charge	86.7	89.9
Units sulphur eliminated	15.0	10.7
Pounds coal used per ton of roaster charge	4.2	14.2
Waste-gas temperature	428.0	396.0

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Reverberatory Division	One Reverberatory	Two Reverberatories
ect charge per furnace-day	402.5	262.7
tling per furnace-day (ores)	84.9	65.7
tling per furnace-day (byproduct)	25.0	38.7
al solid charge smelted per furnace-day	512.4	367.1
fettling to total solid charge	21.45	28.3
direct charge:		
cent	19.8	21.7
cent	5.4	4.9
nt	32.3	26.6
cent	2.3	7.7
t	13.7	13.6
ent	15.55	13.24
tio	1.32	1.41
ettling (ores and byproducts):		
cent	38.8	39.5
cent	7.6	8.0
nt	23.4	25.5
cent	1.7	4.0
t	9.1	3.0
ent	6.65	5.52
tio	3.36	2.93
ot converter slag to reverberatories	2,419.0	2,862.0
everberatory slag:		
cent	35.6	38.0
cent	9.7	9.0
cent	49.7	39.4
cent	2.4	8.4
cent	0.6	1.5
ent	0.51	0.45
tio	2.04	2.06
matte, per cent. Cu	37.43	37.70
tte fall	31.68	27.1
phur volatilized in furnace	24.14	25.7
ed per ton solid charge smelted	0.653	0.778
ed chargeable to steam	0.315	0.349
ed account smelting	0.338	0.429

companying 2 months' results were selected for comparison, ey show the extreme variation in the material treated in this t. Limited storage capacity at the smelter required two fur- e operated a part of the time to handle receipts. Additional ded for the month that two reverberatories were operated was of siliceous oxidized ores and limerock, and represents a tem- dition. The concentrator additions, now almost finished, will ire supply sufficient concentrates to make the proportions of

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total material treated fall somewhere within the percentage limits given in the examples.

Converter Division	One Reverberatory	Two Reverberatories
Tons reverberatory net matte treated	5,293.0	5,903.0
Per cent. Cu in reverberatory net matte treated	37.43	37.70
Tons siliceous ore used for flux	820.0	1,356.0
Analysis of siliceous ore used:		
SiO_2, per cent	64.5	65.3
Al_2O_3, per cent	10.6	11.6
Fe, per cent	5.2	5.7
CaO, per cent	0.2	0.5
S, per cent	0.5	1.7
Cu, per cent	6.45	4.29
Oxygen ratio	25.30	22.65
Analysis of converter slag produced, per cent. SiO_2	19.6	19.6
Tons copper bullion produced	1,800.0	2,450.0
Cu in bullion, per cent	99.60	99.55
Tons copper produced per converter-day running time	43.9	51.4
Minutes blowing per ton copper	33.0	28.0
Average tons bullion per charge	6.429	8.524
Average time of blowing a charge	3 hr. 54 min.	3 hr. 59 min.
Tons cold new material treated per ton bullion	0.462	0.592
Tons cold byproduct re-treated at converters per ton bullion	0.849	0.577
Blast, pounds pressure	13.0	13.0
Cu. ft. air per converter per minute	6,738.0	5,445.0
Cu. ft. air per ton bullion produced	220,949.0	152,499.0
Cu. ft. air per ton iron and sulphur eliminated	135,043.0	96,145.0
Oxygen efficiency, calculated from blower displacement and requirements of Fe and S eliminated	60.63 per cent.	84.09 per cent.
Total Material Treated—All Divisions:		
Dry tons concentrates	13,227.0	13,360.0
Dry tons ores	4,069.0	7,303.0
Dry tons limerock	391.0	2,130.0
Dry tons new material, total	17,687.0	22,793.0
Dry tons byproducts treated at reverberatories	924.0	*2,317.0
Concentrates, per cent. of total new material treated	74.78	58.62
Ores, per cent. of total new material treated	23.01	32.04
Limerock, per cent. of total new material treated	2.21	9.34

* Including reverb. slag skulls.

Boiler Plant

There are 10 Stirling boilers, three 384-hp. direct-fired, and seven 712-hp. waste-heat. Water is measured in total and also separately to the two sets of boilers. Oil is measured to the direct-fired boilers.

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All boilers contain Foster superheaters, and deliver steam to the mains at 175 lb. pressure. The temperature of steam delivered at the power house is about 475° F. The temperature of the waste gases leaving the boilers varies between 471° to 588° F., depending upon the furnace practice and the number of boilers in service. The gases from one furnace practice are passed through five boilers. When two furnaces are in commission, six or seven boilers are used.

When smelting in one furnace, three direct-fired boilers are in continuous service to supply the power requirements, but the night load is light, therefore the boiler efficiency is low. The pounds of water evaporated per pound of oil averaged 11.98 over a period of 8 months. The oil averages around 15° Bé. and 18,200 B.t.u.

With two furnaces in operation, there is sufficient waste heat to generate all steam requirements for the night load, but oil must be used to carry the day load. Under these conditions, rather than keep the oil-fired boilers under steam over night, the three oil-fired boilers are not used. Instead, oil is burned under the waste-heat boilers. The oil so burned is measured and credited with evaporating 11.98 lb. of water, in the same manner as under oil-fired boilers on a one-furnace basis.

The net steam delivered to the power-house engines is credited to the boiler plant. Steam used in heating and atomizing oil and for other miscellaneous purposes, as well as steam wasted, is absorbed in the furnace-oil costs.

Under one-furnace conditions, the waste-heat boilers averaged over a period of 8 months, 7.71 lb. of water evaporated per pound of oil burned in the furnace.

The furnace-boiler division receives credit from the power house varying from 36 to 56 per cent. of the oil burned in the smelting furnace.

The indicated horsepower from the waste-heat boilers, per ton of solid charge smelted on a one-furnace basis over a period of 9 months averaged 98.71 i.hp.-hr. whereas on a two-furnace basis, over a period of 7 months, the average was 114.34 i.hp.-hr. from waste heat only.

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Smelter Power

Divisions	Kilowatt-hours		Kilowatt-hours per Ton New Material	
	One Furnace	Two Furnaces	One Furnace	Two Furnaces
Sampling:				
Receiving, high-pressure air converted to kw.-hr	1,813	2,590	0.10	0.11
Crushing, electric	10,501	21,548	0.59	0.95
Sampling, electric	1,146	2,351	0.06	0.10
Bedding, electric	3,819	7,836	0.22	0.34
Reclaiming, electric	22,576	22,843	1.28	1.00
Total	39,855	57,168	2.25	2.50
Roasting:				
Roaster furnaces, electric	7,692	7,495	0.43	0.33
Calcine cars, electric	3,140	5,093	0.18	0.22
Total	10,832	12,588	0.61	0.55
Reverberatories:				
Reverb. furnaces, electric	3,013	3,303	0.17	0.14
Reverb. furnaces, high-pressure air converted to kw.-hr	453	647	0.03	0.03
Reverb. furnaces, low-pressure air converted to kw.-hr	37,339	72,774	2.11	3.19
Slag railway, electric	4,420	10,451	0.25	0.46
Boilers, high-pressure air converted to kw.-hr	1,813	2,590	0.10	0.11
Boiler feed-water pumps, electric	31,220	20,553	1.77	0.90
Oil pumps, electric	4,030	2,009	0.23	0.09
Total	82,288	112,327	4.66	4.92
Converting:				
Converters, electric	5,263	5,674	0.30	0.25
Converters, low-pressure air converted to kw.-hr	218,235	206,802	12.34	9.07
Cranes, electric	22,790	19,635	1.29	0.86
Bullion casting machines, electric	2,340	2,920	0.13	0.13
Total	248,628	235,031	14.06	10.31
General Works:				
River well, electric	6,760	8,001	0.38	0.35
Boiler and machine shops, electric	2,900	2,730	0.16	0.12
Boiler and machine shops, high-pressure air converted to kw.-hr.	4,986	7,122	0.28	0.31
Yards, electric	1,313	1,439	0.07	0.06
Laboratory, electric	2,807	3,077	0.16	0.13
Total	18,766	22,369	1.05	0.97
Total, all divisions	400,369	439,483	22.63	19.25
Dry tons new material treated	17,687	22,793		

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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Leaching Tests at New Cornelia

BY H. W. MORSE, PH. D., LOS ANGELES, CAL.,* AND H. A. TOBELMANN,† AJO, ARIZONA

(Arizona Meeting, September, 1916)

INTRODUCTION

THE experimental work on the oxidized copper ore at the New Cornelia mine at Ajo, Ariz., ended on Jan. 12, 1916. On that date final decision was made on the general nature of the process to be used in the 5,000-ton leaching plant, and on many of the details, as far as experience on a40-ton scale could decide them.

With the approval of the Board of Directors and the General Manager, John C. Greenway, we have compiled what seem to be the most interesting data on the results obtained during the experimental period. The most important part of the data resulted from the operation of a 1-ton and a 40-ton plant at Ajo.

Experimental work on Cornelia oxidized ore dates back to April, 1912, and has been going on nearly continuously since that time. A good many variations from the original idea have been tried, but the process finally decided upon is in principle and in all of its details a simple one.

The history of the leaching work on this ore has been brought down to about a year ago in papers presented to this Institute by Stuart Croasdale[1] and Dr. L. D. Ricketts.[2] The present paper will, therefore, deal principally with the results obtained during the last year of the work.

PRELIMINARY TESTS BY MR. CROASDALE

Preliminary tests were begun by Stuart Croasdale in July, 1912. A number of important points were definitely decided by his work. Among these were:

1. New Cornelia oxidized ore, not very finely crushed (to about 2- to 3-mesh) showed good extraction with 5 per cent. sulphuric acid. It appeared that better than 80 per cent. extraction could be expected.

2. The consumption of acid for this extraction was not prohibitive. It was apparently 3.5 to 4 lb. gross per pound of copper recovered.

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[1] Leaching Experiments on the Ajo Ores, *Trans.*, vol. 49, p. 610.

[2] Some Problems in Copper Leaching, *Trans.*, vol. 52, p. 737.

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3. It was proposed to make cement copper. The iron consumption per pound of copper recovered was a little over 1 lb.

4. A column 12 ft. deep of ore crushed to this fineness could be successfully percolated.

During Mr. Croasdale's work the plan in view was a very simple one indeed. The ore was to be leached with sulphuric acid. Cement copper was to be produced by the use of metallic iron. As Dr. Ricketts explained to the Institute nearly 2 years ago, the oxidized Cornelia ore could apparently be leached and cement copper produced at a profit, if sulphuric acid and scrap iron were bought in the open market and shipped to Ajo, and cement copper shipped out, all solutions being thrown away after removal of the copper.

Mr. Croasdale showed also that the amount of substances other than copper, taken into solution under his conditions of crushing and leaching, is small. This naturally suggested the possibility of a closed leaching cycle with electrolytic deposition and at least partial regeneration of the acid required for solution of the copper. Early experiments showed that iron, which is dissolved from the ore to a certain extent, and oxidized at the anode during electrolysis, was a troublesome factor.

There seemed to be three possible ways of meeting this difficulty: (1) The use of a diaphragm; (2) the use of a depolarizer; (3) purification from iron before electrolysis.

Antisell Process

About this time there came to the notice of the manager good results that were being obtained with a special anode, the invention of F. L. Antisell. This anode was in the form of a long, deep, narrow box with wood veneer sides, containing graphite electrodes and having the space between graphite and wood filled with selected coke.

A cyclic process was tried with these anodes. Ore crushed to 4-mesh was leached 3 days by upward percolation, with a practically constant solution containing 3 per cent. H_2SO_4, 2 per cent. ferrous iron, 0.5 per cent. ferric iron and 2 per cent. Al_2O_3. Sulphur dioxide gas was used to keep down the ferric iron concentration.

The process gave good results. The anode was, however, rather cumbersome and the absorption of the sulphur dioxide was not satisfactory, presumably owing to the rather high acid content of the solution.

The following is a summary of average results obtained by the Antisell Process: Tons of ore leached, 75; heads, 1.68 per cent. Cu; extraction, 77.6 per cent.; current density, 6.54 amp. per square foot; voltage, 1.23; production of copper, 1.72 lb. per kilowatt-hour; H_2SO_4 in solution, 2.73; sulphur per pound of copper, 1.06 lb.; total days run, 40.2; total cathodes produced, 803 lb.

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Process of Pope and Hahn

While these tests were being carried on, another process was being developed at Raritan by F. A. Pope and A. W. Hahn. The Antisell idea was in a sense a combination of diaphragm and depolarizer, without any attempt to remove the iron. Pope and Hahn planned to remove the iron (and most of the other impurities as well) before sending solution to be electrolyzed.

Their process was cyclic. Ore crushed to 4-mesh was leached for 3 days with dilute sulphuric acid, the counter-current leach being so distributed as to permit of drawing off two classes of solution—one high in iron, the other low. The high iron solution was heated to about 200° F. with the addition of the requisite amount of finely ground (90 per cent. through 200-mesh) copper oxide in the form of roasted ore. The mixture of solution and oxides was agitated for 3 or 4 hr. and then put through a filter press. The result was the removal of nearly 90 per cent. of the total iron and 70 to 80 per cent. of the alumina from solution. An electrolyte high in copper and low in all impurities was produced and the precipitate, as filter cake, was granular and easy to wash and handle.

The other portion of the leaching solution, low in iron, was electrolyzed and returned to the cycle until its iron content was high enough to require purification.

Results of the Pope-Hahn process were very satisfactory, but the plant had hardly started when the war broke out and all experimental work was stopped. The following summary will indicate the results obtained: Ore leached, 329 tons, 10 charges; average copper heads, 1.12 per cent.; copper tails, 0.30 per cent.; water, tails, 9 per cent.; copper per kilowatt-hour, 1.0 lb. About 40 lb. of roasted Miami concentrate per ton of ore handled took care of impurities; 92 per cent. of the copper in the concentrate was recovered.

Working the Pope-Hahn process would have required the installation of roasting plant, fine-crushing plant, agitators and filter presses and the purchase of high-grade sulphide ore or concentrates. It offered decided advantages, among them a practically pure solution for electrolysis, together with the regeneration of a high percentage of the acid necessary for leaching.

Greenway Process

Before the study of the effect of impurities had progressed very far, an interesting fact was noticed. If neutral solution from the leaching system was circulated through fresh ore, a large part of the iron in solution was precipitated, presumably as a basic sulphate. This basic sulphate was apparently not readily soluble in a solution containing more

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acid. A possible method of purification from iron seemed to be indicated and the results of small-scale tests were so encouraging that a 1-ton plant was built. In the first tests, ore crushed to about ½ in. was leached for 6 days. The oldest charge of ore was in contact with fresh leach from the electrolytic cells. This leach carried about 3 per cent. of free H_2SO_4. Solution passed in counter-current through six tanks containing ore, and the acid of the leach was neutralized by the time it reached the fourth or fifth tank. In the last two tanks a neutral solution was in contact with fresh ore and here the iron was deposited.

The results obtained will be evident from the data of Table 2. The ferric iron was under good control for more than 100 complete cycles.

TABLE 1.—*Analysis of Average Cornelia Oxidized Ore*

	Per Cent.
SiO_2	65.7
Al_2O_3	15.3
Fe total	4.5
CaO	0.6
MgO	1.6
MnO	0.14
Cu	1.50
	Oz. per Ton
Au	0.005
Ag	0.20
	Per Cent.
NaO	3.6
K_2O	4.6
P_2O_5	0.3
Cl	trace
CO_2	1.3
	99.04

At this time it was already practically settled:

1. That a good extraction could be obtained with dilute sulphuric acid.

2. That the acid consumption was not prohibitive.

3. That electrolytic copper could be produced successfully.

There remained, however, some important problems to be solved. Among these were:

1. Crushing; from two points of view: (1) for percolation in a deep bed; (2) for extraction.

2. Further study of the fouling of the solution, including the general accumulation of soluble salts, but especially those which might affect extraction or power efficiency in electrolysis.

3. Circulation and general handling on a small commercial scale.

4. Any factors other than those under (2), which might affect electrolysis itself or the quality of copper produced.

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Charge No. (Inclusive) Date	1–19 8–8 to 8–31	20–74 9–1 to 10–19	75–100 10–31 to 11–25	101–116 11–26 to 12–10	117–217 12–11 to 3–27	218–259 4–20 to 5–31	260–275 6–1 to 6–16	276–299 6–17 to 7–11	300–330 7–12 to 8–11	341–413 8–18 to 11–3
Average:										
Heads—per cent copper	1.70	1.22	1.55	1.39	1.29	1.30	1.36	1.29	1.22	1.33
Tails—per cent copper	0.64	0.29	0.43	0.32	0.24	0.24	0.19	0.17	0.24	0.27
Extraction—per cent	62.8	75.9	72.2	77.1	81.4	81.5	85.8	86.8	80.3	79.7
Per cent. ore on 4-mesh	29.0	47.0	43.4	28.0	33.0	25.0	26.0	25.1	35.1	39.0
Days leached	6.0	6.0	6.0	6.8	7.6	8.0	7.5	7.5	7.5	8.0
Circulation—1-1 gal. per ton per min	2.0	1.8	3.3	3.3	3.0	2.0	2.0	2.0	2.0	2.0
Advance—1-1 gal. per ton per min	0.42	0.40	0.17	0.20	0.20	0.16	0.16	0.20	0.16	0.29
Electrolytic data:										
Current density, amp. per sq. ft.	9.8	12.0	7.0	7.3	7.1	6.9	9.2	6.6	5.3	8.5
Volts—drop in tank	2.11	2.36	2.00	2.15	2.12	2.29	2.30	2.08	2.03	0.97
Lb. copper per kw.-hr.	0.98	0.83	1.08	1.01	0.99	0.77	0.76	1.07	0.95	2.0
Ave. free H_2SO_4 in cells	4.2	3.0	2.8	2.7	2.9	2.9	3.3	3.4	2.8	3.8
Ave. circulation in cells	3.8	4.6	4.4	4.2	5.8	7.0	8.2	7.5	7.8	5.0
Ampere efficiency	83.7	76.8	83.5	85.0	83.1	67.2	67.3	75.0	74.0	69.5
Acid consumption:										
Total lb. 100 per cent. H_2SO_4 used	1,217.0	1,160.0	729.0	467.0	3,428.0	1,107.0	492.0	1,002.0	1,113.0	4,240.0
Acid per lb. cathode Cu	5.33	1.76	1.16	1.62	1.66	1.46	1.30	2.01	2.74	6.65
Acid per lb. Cu leached	3.02	0.94	1.06	1.36	1.61	1.24	1.31	1.86	1.83	2.30

NOTES.—Charges Nos. 1 to 19 had no crushing plant, so screened rejects from sampling oxide pits were used. Ore had excess fines and contained some sulphides. No filter bottoms in tanks. Segregation, started with new solution. Leaching started Aug. 8, electrolytic end started Aug. 17 with grid lead anodes.

Charges Nos. 20 to 74, on Oct. 6, tried reduction with liquid SO_2 using filtros bottom to distribute gas. Test stopped Oct. 14.

Charge No. 64.—All other charges normal. On Oct. 19, decided to discard one-half of solution volume, dilute to normal volume with well No. 1 water and make special run

Charges Nos. 75 to 100.—No SO_2 used and water from well No. 1 (high Cl) used on these charges.

Charges Nos. 101 to 116.—Same solution, longer leach, finer crushing variable conditions—No SO_2.

Charges Nos. 117 to 217.—On Dec. 14, SO_2 reduction tests were begun under the direction of F. L. Antisell. Lead-coke anodes were used. Tests discontinued Dec. 26—between charges 117 to 127. Leaching varied from 7 to 13 days. Charges Nos. 127 to 217 were run under normal conditions.

Charges Nos. 162, 163, 164 were high-grade ore—"k-10"-av. 2.7 per cent. Cu. These have not been included in the above averages. On Feb. 27, the average current density of 7 amp. per square foot was reduced to 4 amp. per square foot.

Charge No. 218.—April 19 was started with 40 tons high ferric iron solution, other conditions normal.

Charge No. 275.—June 17, arrangements made for another SO_2 run with lead anodes; same solution used. Test continued until charge 300.

Charges 300 to 330 run without SO_2 and under normal conditions.

Charge Nos. 341 to 413.—New solution, carbon anodes, SO_2 reduction, air agitation.

Ore used on charges Nos. 20 to 413 was from open cut and averaged 1.307 per cent. Cu.

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The 1-ton plant was therefore continued in operation and a plant t handle 40 tons of ore per day was built.

It is fortunate that this larger unit was built, for a good many thing that had seemed easy to do in the 1-ton plant refused to work at all i the 40-ton. Fouling by iron and aluminum salts kept on increasing, th ferric-iron content of the solution rose to a point where the ampere eff ciency in electrolysis was poor, and various undesirable phenomen appeared.

So the 40-ton plant was kept in continuous operation for nearly year and all the factors that had been studied in the 1-ton plant wer taken up, one at a time, and worked over again until finally it seemed tha they could all be brought under complete control.

Final Leaching Process

The final process decided upon was as follows: Ore crushed to abou 4-mesh; leached 8 full days by counter-current; washed with thre counter-current wash waters (or possibly four); solution, practicall neutral during last 2 days of contact with ore, is sent through reductio towers, where it meets sulphur dioxide in counter-current. The ferri iron is thus reduced to below 0.4 per cent. Thence it goes through revolving tumbler in contact with cement copper. There the ferric iro is still further reduced and a corresponding amount of cement coppe passes into solution. From the cement-copper tumbler the solutio passes to a settling pond and then into the electrolytic cells, where a pa of the copper is removed; then back into the leaching system, passin first through the oldest ore which has already been leaching for 7 day and so on to begin the cycle again.

Important Factors

In a cyclic process such as this, no single factor or step can be said t be the most important. We can, however, consider a list of those facto that were studied with especial attention and can offer definite figur on the results to be expected.

Extraction

Both in the 1-ton and the 40-ton plants, it was easy to obtain 80 p cent. extraction, using about 3 per cent. sulphuric acid and 8 days counter-current leaching by upward percolation on 4-mesh materia An extraction of 83 per cent. was reached over considerable periods time. The continuous attainment of this figure was limited by oth factors than the nature of the ore or the size to which it was crushed, su

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as the fouling of solution, with the result that salts carrying copper actually crystallized in the ore in the leaching tanks and were not completely dissolved by washing.

Incomplete washing was, of course, another reason for low extraction under otherwise normal conditions, and the possibility of improvement in this point is clearly shown by the wash-water data of Table 6.

Keeping well within the time limit for washing set by the fixed cycle, it should apparently be possible to get an extraction of 82 per cent. or better.

Acid Consumption

The 1-ton plant, over its whole life of 413 cycles and under all sorts of favorable and unfavorable conditions, showed a net consumption of 1.65 lb. of 100 per cent. H_2SO_4 per pound of copper leached. This was somewhat lower than the figures given by the preliminary tests on the ore.

The 40-ton plant apparently did not do so well. During its 301 cycles, the average net consumption was 2.8 lb. of acid per pound of copper leached. The difference in favor of the 1-ton plant is probably due to the large leakage and general losses in the 40-ton plant.

In designing the plant which is now being built at Douglas to furnish acid for Cornelia, 3 lb. has been taken as a conservative figure for acid consumption.

Power Consumption for Electrolysis

In the 1-ton plant, with lead anodes, the average was 0.934 lb. copper per kilowatt-hour at 8 amp. per square foot of cathode surface. This was for all the conditions, good and bad, under which the plant was operated. With graphite anodes, also under all sorts of conditions, the figure was 2 lb. copper per kilowatt-hour at 8.5 amp. per square foot.

In the 40-ton plant, with lead anodes, at an average current density of 6.9 amp. per square foot, the average production under all conditions was 0.87 lb. copper per kilowatt-hour. With graphite anodes, the figure was 1.65 lb. It should be said, however, that the ferric iron was never under proper control during these tests with graphite in the 40-ton plant. During series VI of the 40-ton plant, with current density 6.4 amp. per square foot, the copper yield was 1.04 lb. per kilowatt-hour. This run was with lead anodes and under fair conditions, so far as ferric iron was concerned. In the final run with lead anodes, and with both ferric iron and general fouling under still better control, an even higher figure was obtained.

Lead anodes will be used in the big plant. The reasons for this choice will be given later.

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TABLE 3.—*Summary of All 40-Ton Tests with Lead Anodes. Feb. 1 to Nov. 6, 1915*

Charge No. (Inclusive) Date	1 to 81 Feb. 1–May 21	91 to 130 June 1–July 12	131 to 154 July 13–Aug. 4	155 to 170 Aug. 5–Aug. 22	171 to 201 Aug. 23–Sept. 25	202 to 238 Sept. 26–Nov. 7	1 to 238 Feb. 1–Nov. 7
Average:							
Heads—per cent copper	1.248	1.314	1.225	1.274	1.286	1.361	1.281
Tails—per cent copper	0.294	0.226	0.242	0.265	0.346	0.223	0.275
Extraction—per cent	76.4	82.8	80.2	79.2	73.1	83.6	78.5
Per cent. ore on 4-mesh	25.0	25.3	35.0	32.0	40.7	37.9	36.0
Days leaching	9.3	8.0	8.0	8.0	8.0	8.0	8.3
Circulation—gal. per ton per min.	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Advance—gal. per ton per min.	0.20	0.35	0.35	0.35	0.35	0.38	0.30
Electrolytic data:							
Current density—amp. per sq. ft.	7.4	7.1	7.1	7.1	6.0	6.4	6.9
Volts—drop in tank	2.43	2.20	2.18	2.04	2.18	2.07	2.24
Lb. copper per kw.-hr.	0.72	0.99	0.85	0.98	0.86	1.04	0.87
Ampere efficiency	66.8	83.3	70.8	76.5	71.7	82.3	74.0
Lb. cathode copper	49,775.0	30,652.0	17,431.0	10,521.0	18,273.0	23,717.0	150,349.0
Acid consumption:							
Total lb. used—100 per cent.	188,484.0	98,442.0	47,484.0	37,197.0	85,173.0	102,402.0	559,182.0
Lb. H_2SO_4 per lb. cathode Cu	3.82	3.21	2.72	3.53	4.66	4.31	3.71
Lb. H_2SO_4 per lb. Cu. in solution	2.88	2.69	2.39	2.58	3.67	2.98	2.89
Sulphur:							
Total lb. used	None	None	4,693.0	5,434.0	18,767.0	26,697.0	55,861.0
Lb. S per lb. cathode Cu	None	None	0.27	0.52	1.02	1.12	0.80
Lb. S per lb. iron reduced	None	None	1.50	0.56	1.20	0.99	1.08
	Old solution— No SO_2 first run	New solution— No SO_2 new run	SO_2 small tower acid electrolyte	SO_2 small tower neut. electrolyte	SO_2 large tower solution saturated	SO_2 large tower solution diluted	Av. of all 40-ton tests with Pb anodes

NOTE.—Forty-ton plant started Feb. 1, 1915, with Pope-Hahn solution, tank house started Feb. 26. Control of ferric iron lost on Mar. 12

Charge No. 26, various attempts made to change iron ratio. Low tank-house efficiency caused copper content of leaching solution to increase to such an extent as to interfere with the extractions. Troubles with pumps, talcy ore, linings, etc. Solution of these first 81 charges discarded on May 21 and a new series of tests started, May 22, 1915. Charge No. 91, new solution was made up for this run. Electrolytic plant started June 1. Iron control lost on charge No. 117. June 27, reduction of the ferric iron decided upon and on July 13 the acid electrolyte was circulated through small reducing tower. As reduction did not appear to gain on the oxidation in the tank house it was decided to circulate neutral solution through the tower. Results showed reduction of ferric iron in neutral solution to be less difficult than the ferric iron in acid solution. Large tower built, and completed Aug. 23. Acid electrolyte reduction continued through large tower. Solution becomes saturated with salts and on sudden changes in temperature deposits crystals on ore. Extraction decreases and it was found necessary to dilute the solution by intermittently "bleeding" the "high iron" tank and making up the

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Control of Ferric Iron

Quite early in the study of the conditions for good electrolysis it became evident that the ferric-iron content of solutions must be kept within a low limit. The 1-ton plant took care of itself over a long period of time, as far as the ferric-iron content and the general fouling were concerned, without the use of sulphur dioxide or any other outside reagent. Iron and alumina were apparently precipitated on the ore.

The 40-ton plant started off in the same way. But before long the ferric iron began to go up and the ampere efficiency to go down. This is evident from the table showing general results on the 40-ton plant. Series II began with fresh acid and lasted 40 days. During this period 0.99 lb. Cu per kilowatt-hour was the electrolytic yield. The current efficiency during this period of low ferric iron and clear solution was 83.3 per cent. During period III the same solution was continued in use, but the ferric iron was not under control, and kept increasing. The analysis of the solution on July 22, in the middle of this run, is shown in column III of Table 4. The ferric iron content had risen rapidly from 0.180 per cent. on June 20 to 1.222 per cent. on July 22. The result was that during period III, the current efficiency was only 70.8 per cent. and the yield was 0.85 lb. copper per kilowatt-hour. From this time until the

TABLE 4.—*Solution Analyses*

Second Period of 40-Ton Plant Operation Beginning with Charge 91 in 1915

	I 6–1, Per Cent.	II 6–20, Per Cent.	III 7–22, Per Cent.	IV 8–22, Per Cent.	V 9–23, Per Cent.	VI 11–5, Per Cent.
Cu	1.11	2.26	2.50	3.39	3.47	2.95
H_2SO_4 free	4.03	2.88	2.56	3.15	2.40	3.05
Fe^{++}	0.09	0.176	0.720	1.49	2.14	2.52
$FeSO^{+++}$	0.164	0.180	1.220	1.07	0.80	0.239
Al_2O_3	0.145	1.250	2.730	3.69	4.40	2.840
CaO	0.092	0.090	0.080	0.050	0.120	0.018
MgO	0.132	0.122	0.120	0.150	0.049	0.054
MnO	0.023	0.030	0.030	0.050	0.046	0.029
Cl	0.08	0.08	.....	.....	0.130	0.090
SiO_2	0.075	0.084	0.110	0.040	0.960	0.096
Specific gravity	1.100	1.150	1.250	1.370	1.376	1.330

Column I gives solution analysis at beginning of a run, where fresh acid has been used, and before electrolysis has been started.

Column III is the analysis of solution when reduction of ferric iron was begun.

Column V shows solution at maximum concentration. Cold nights caused the separation of a large amount of $FeSO_4$, $CuSO_4$, $24H_2O$.

Column VI is an analysis of solution under normal conditions, after "bleeding" regularly for about 40 days.

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end of the experimental work, sulphur dioxide gas was used regularly to control the ferric iron. With lead anodes a low content of ferric iron is not necessary. The solution may contain 0.3 or 0.4 per cent. without any marked effect on the current efficiency.

Sulphur Consumption

To maintain satisfactory control of the ferric iron was, at this point in the work, the greatest apparent difficulty. To follow the course of the tests it will be necessary to keep in mind both Tables 3 and 4. The first of these shows the conditions under which the 40-ton plant was operating; the latter, the analysis of the solution at various times.

During period III, the acid electrolyte was circulated through a tower about 2 ft. square, in contact with sulphur dioxide. The acid electrolyte was taken from the electrolytic system and returned to the same system. The weight of sulphur that could be burnt and brought in contact with solution as SO_2 was too small, and the ferric iron kept on increasing. Under these conditions, we were burning 1.5 lb. sulphur per pound of ferric iron reduced.

Series IV began with only a single change, but this proved to be a very important one. The *neutral* advance from the leaching system was circulated through the SO_2 tower before entering the electrolytic system. Under these conditions, only 0.56 lb. of sulphur was required per pound of ferric iron reduced.

Although the small tower was not of sufficient capacity to make control easy, by Aug. 22 the ferric iron was on the down grade—1.07 per cent. as against 1.22 on July 22.

From this time on, as long as lead anodes were in use, the ferric iron did not get out of control, and on Nov. 5 the solution showed only 0.24 per cent. of this troublesome substance. The results with graphite anodes will be considered later.

Fouling of Solution

In the 40-ton plant the entire solution began to foul badly after about 100 charges had been leached. The cooler weather was beginning at this time and a bulky double sulphate of iron and copper crystallized out through the ore in the leaching tanks and plugged all the pipes and launders. To remedy this, a portion of nearly neutral solution was drawn off continuously from the leaching system and passed over scrap iron, making cement-copper. It was found that treating 1 per cent. of the total bulk of solution each day would maintain the fouling at a constant low point where it could cause no trouble. Columns V and VI of Table 4 show the effect of this "bleeding" of solution.

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nt-copper so produced was returned to a wooden tumbler
e on the way to the cells passed through it. Ferric iron is
rous in this way and copper passes back into solution.
se conditions only electrolytic copper is produced and at
the control of the ferric iron is aided considerably. In the
it was apparently necessary to send about 12 per cent. of
per through this side cycle and back into the main one, to
etely the fouling materials introduced from the ore during

Grade of Copper Produced

copper produced direct from the ore is the equal of any
pper produced in this country. Fortunately the orebody
minute amounts of either arsenic, antimony or bismuth.
and the physical tests on Cornelia cathodes will be found in

Table 5.—Analysis of Cornelia Copper Cathode

	Top, Per Cent.	Bottom, Per Cent.		Top, Per Cent.	Bottom, Per Cent.
...	99.900	99.868	S.....	0.0369	0.0655
...	0.0186	0.0291	As.....	0.0013	0.0017
...	0.0012	0.0019	Sb.....	0.00095	0.0014

Physical Tests on Wire Bars

est.....	64,500
gation.....	1.9
s.....	36
ts.....	43
ivity.....	100.7

Size of Material. Percolation

at Ajo was all done on the product from a Symons fine
out any screening (except for tests on sized material). It
y a 4-mesh product but carried an average of about 30 per
size. Extraction on this is satisfactory and while it could
by finer crushing, a large percentage of the copper minerals
re planes, and no very great profit would result.
luct is especially well suited to upward percolation in a bed
more. It does not segregate nor channel badly, drains well
o wash.

Circulation in the Leaching System

-ton plant, each of the eight active tanks had a closed circu-
rd through the ore, of about 80 gal. per minute. This rate

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is probably much higher than necessary, but for the experimental work i was made ample and retained as a fixed factor. Beside this closed circuit which runs continuously, a portion of the solution from each tank i continuously advanced to the next tank in the series.

Fresh electrolyte from the cells, low in copper and of maximum aci content, leaves the electrolytic system and passes into the leaching tan containing the oldest ore—that which has been leaching for 7 day Practically neutral solution, high in copper, leaves the leaching syste at the tank containing new ore. Between these two points, and rigidl linked to them, is the advance from tank to tank.

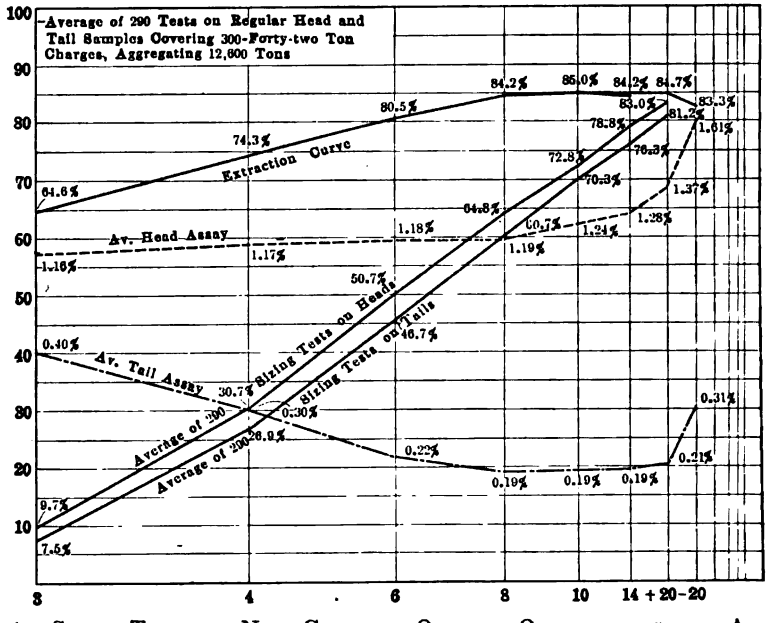

FIG. 1.—SIZING TESTS ON NEW CORNELIA OXIDIZED ORE SHOWING ALSO AVERA ASSAYS ON HEADS AND TAILS AND AVERAGE EXTRACTIONS ON SIZED MATERIAL

It is of great importance to have a practically neutral solution lea ing the last leaching tank and starting back toward the electroly system.

At the same time, we want as high an extraction as possible, wh means as many tanks as possible with a fairly high acid conte To attain both these objects at the same time the advance from ta to tank is carefully regulated so that about six tanks of the eight act ones are acid practically all the time. The two others are neutral a the point of neutrality of the system just balances, varying possi one tank in either direction during 24 hr., but returning to its origi position.

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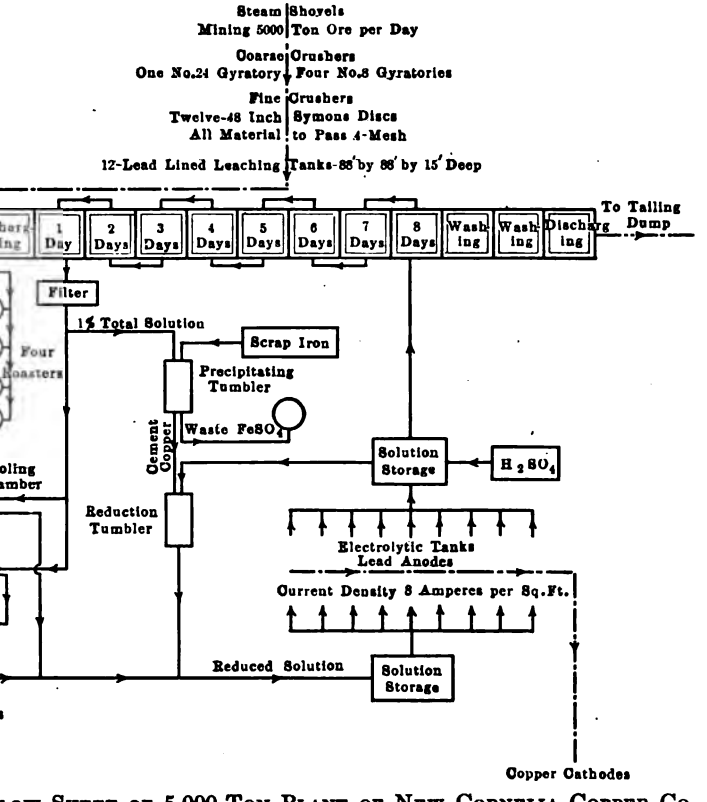
Circulation in the Electrolytic System

trolytic cells, together with tanks and sumps for storage, form sed circuit from which solution is bypassed continuously into system at the same rate as the advance from cell to cell. To od, coherent cathodes, the circulation through the cells must eater in volume than the advance from the leaching system, is rigidly fixed by the necessity for maintaining neutral leach- While the advance in the 40-ton plant was about 12 gal.



LOW SHEET OF 5,000-TON PLANT OF NEW CORNELIA COPPER CO.

it was advantageous to circulate in the tank house at rates al. per minute.

culation gives satisfactory results with lead anodes, but ite is used, rapid circulation alone is not enough to give good on at the anode. Violent agitation in the electrolytic tanks to attain the lowest voltage when graphite anodes are in use, s accomplished by blowing air through small holes in lead ong the bottom of the cell.

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Lead vs. Graphite Anodes

In making final decision on the process to be used, there were mar mechanical questions to be settled. Crushing, method of percolatio circulation, pumps, structural materials, the handling of ore and tailin and many other points came up for decision. Besides these, there we some questions which bear more directly on fundamental principle One of these was whether we should use lead or graphite as material f anodes.

Lead was in service for over a year and its behavior was well unde stood. But graphite promised much better results as far as power ef ciency is concerned, and also as a means of regenerating a much larg part of the necessary acid for leaching.

The 1-ton plant gave most encouraging results with graphite over period of continuous operation of nearly 3 months. With a curre density of 8.5 amp. per square foot, over 2 lb. of copper per kilowatt-ho was produced, and during part of this time conditions were not goo The ferric iron was not under complete control. When all conditio were right, we made 2.25 to 2.40 lb. copper per kilowatt-hour; on son days as high as 2.75 lb. This was during the warm weather at Ajo, wh electrolyte temperatures held well above 105° F. The 40-ton plant w using the large tower for reduction and the 1-ton plant used the sm one. The SO_2 used in controlling ferric iron was bypassed from t main supply.

The 1-ton electrolytic plant was out of doors and while the odor SO_2 was strong in the immediate neighborhood of the cell, it was not b enough to prevent working over it.

By the time the graphite anodes were installed in the 40-ton pla the average electrolyte temperature was only about 75° F. We cou now determine accurately the efficiency of absorption and reduction SO_2 (which we could not do in the 1-ton plant) and we could also exami carefully the circulation conditions necessary to control ferric iron.

The problem now began to look harder. As measured by the prod tion of ferric iron from ferrous in the cell, the anodic efficiency of graphi with rapid air agitation, is well over 100 per cent. With lead it is 30 35 per cent. We had therefore to combat about three times as mu ferric iron per pound of copper with graphite as with lead.

Under the existing temperature conditions and with the rapid circu tion that was necessary in the electrolytic circuit, the reduction efficie of the SO_2 was only about 20 per cent. We could not control the fe iron without either allowing the electrolyte to stand for a good ma hours between the SO_2 towers and the cells, or else heating it to ab 150° F. during its passage from towers to cells.

The absorption of SO_2 in the towers was perfectly satisfactory.

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used to better advantage in the reduction of the iron in the
vas impossible to live in the tank house without either very
lation or a diver's outfit. With the violent air agitation
essary for high power efficiency, most of the SO_2 which had
d in the towers was blown out in the tank house.
ficulties are by no means insuperable. Of more fundamental
low current efficiency obtained with graphite and its extreme
toward ferric iron. When ferric iron is practically com-
ved by careful reduction with SO_2, graphite can be made to
rrent efficiencies. But the very slightest rise in ferric-iron
n results in an immediate and considerable lowering in ampere
Even in our good runs with graphite in the 1-ton plant, the
ency was not much above 70 per cent. With lead anodes,
any particular attention to the ferric iron, which was
r 0.4 per cent., we often obtained current efficiencies of 90
better over considerable periods.
e of interest to state some of the points that enter into a
ween the two kinds of anodes:
rst cost of an installation is about the same.
fe of lead is known with some accuracy. The life of graphite
e is not known.
alvage value of lead is high. With graphite it is practically

robable power saving to be expected (graphite over lead) if
e successfully operated, is about 0.6 kw.-hr. per pound of

than twice as much acid is regenerated with graphite as with

ation must be more rapid with graphite.
ion during electrolysis is absolutely necessary with graphite.
olution must stand for some time (probably 16 to 24 hr.)
to 150° F. or higher, to obtain even reasonable SO_2 efficiency
e. This is not necessary with lead (see note under 10).
larger towers are needed when graphite is used.
volume to be pumped to the top of the towers is larger with

st two statements contain the fact that acid electrolyte must
gh the towers. The iron content of the solution in the cells
e above 0.2 per cent., if good efficiency is to be reached with
With lead anodes it may go to 0.4 or 0.5 per cent. without
effect on power consumption. With lead it is sufficient to
utral advance to the towers. With graphite not only this
also a much larger volume from the tank-house circulation
be kept low in ferric iron must be sent.)
tank house for graphite must be carefully ventilated.

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12. The tank house for graphite must be larger (about as 9 to 7 on account of the lower current efficiency.

13. Operation with graphite requires the closest attention to every detail of reduction, circulation and agitation. Operation with lead practically fool-proof.

This last point is probably the most important of all. There are enough so-called "minor" troubles about a leaching plant.

Control of Ferric Iron in Acid and Neutral Solution

During the periods when acid electrolyte was being sent to the towers for the reduction of ferric iron by sulphur dioxide, the sulphur efficiency was low. It proved to be about 20 to 25 per cent. Two causes at least combine to produce this result: First, the low solubility of SO_2 in acid solution, and second, the slowness of the reduction reaction in acid solution. Neutral solution was sent to the towers in later runs and on this the sulphur efficiency reached 75 to 80 per cent. That is to say, of the actual weight of sulphur burned to SO_2, 75 per cent. is utilized in the reduction of ferric iron to ferrous.

Washing the Ore

Each charge was given three washes. The first was the second wash on the preceding charge; the second, the third wash of the preceding charge, the third, fresh water. The wash which had been used three times was run into the electrolytic system. One complete fresh wash water was just sufficient to balance the loss from the system in the tailings and by evaporation.

The summarized wash-water data are given in Table 6 for the solution which was started in the 40-ton plant May 22, 1915. The most significant thing in this table is the effect of fouling as shown in the second column. While the solution was fresh, washing in the prescribed manner the extraction was 81.1 per cent. and soluble copper left in the tailings was 0.045 per cent. When the solution became foul, even though a little more fresh water was used, extraction dropped to 73.5 per cent. and the soluble copper in the tailings was 0.103 per cent. Later, when fouling was being reduced by "bleeding" and additional wash water could be used in the same cycle, the extraction rose to 83 per cent. and the soluble copper in the tails dropped to 0.033 per cent.

It is of course possible to wash tailings with extra water, passing this over scrap iron and making cement-copper, and this should naturally be done until the expense of operation balances the value of the recovered copper. The experiments at Cornelia were carried no further than shown in this table.

All the experimental work in the 40-ton plant was carried out during continuous operation, so that the result should correspond as closely

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commercial conditions. The regular tonnage was crushed charged into the leaching tank. Tailings were removed, and dvance, washing, electrolysis and reduction carried on with- Any change in operation was carried on for at least a week, or 10 days. If the change promised to give improved results, ixed part of the regular operating routine and the next point study.

TABLE 6.—*Wash-Water Data*

rges Covered by Period	No. 91 to No. 170, May 22 to Aug. 20	No. 171 to No. 205, Aug. 21 to Sept. 25	No. 206 to No. 301, Sept. 26 to Jan. 10
ated	3,376	1,435	4,051
er in the heads	1.278	1.282	1.331
er in the tailings	0.240	0.342	0.237
ction	81.1	73.5	83.0
er in washed tailings	0.195	0.239	0.204
raction if tailings had been			
	84.7	81.3	85.3
ire in tailings	10.52	9.57	9.65
er of washes	3	3	3
new water used per ton of ore	40.6	42.2	59.1
ned in tailings per ton tailings	24.2	21.2	21.8
unted for per ton of ore leached, on and leakage, etc	16.4	21.0	12.1
tion bled per ton ore	None	None	25.2
of 8th day solution	1.100—1.290	1.290—1.400	1.400—1.300
of last wash	1.050—1.060	1.060—1.150	1.150—1.050
nt. copper in first wash	1.66	2.96	2.25
nt. copper in second wash	1.17	2.17	1.45
nt. copper in third wash	0.61	1.25	0.67
nt. H_2SO_4 in third wash	0.35	0.35	0.35
copper per ton tailings	0.9	2.1	0.7
tion if one more wash had been of this soluble copper had been			
er cent	83.3	79.0	84.2

ble run:

. copper in heads	1.338
. copper in tailings	0.256
. extraction	80.7
. extraction—according to washed tails	84.1
. extraction—if 4th wash had been used	83.0
. moisture in tailings	10.0
gallons of solution entrained per ton of tails	22.6

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Using this method, when we were through, we had a definite set conditions under which known results could be produced for any leng of time. There were no odds and ends to go back and pick up. N "minor" problems, which might turn out on investigation to be fund mental, remained to be solved. Mr. Greenway saw that this plan w strictly carried out.

During the experimental work, something over 180,000 lb. of perfe cathodes were produced, besides a good deal of cement-copper.

Summary

So far as the Ajo tests can show what will happen in a 5,000-ton pla the following may be expected:

Extraction on 1.4 per cent. ore using closed wash cycle, 82 per ce

Extraction on 1.4 per cent. ore using extra wash water, 83 per cent

(The extraction figure used in making estimates has always be 80 per cent.).

Acid consumption (100 per cent. H_2SO_4), 3 lb. per pound of copp "Bleeding" to prevent fouling of solution.

Power efficiency in electrolysis with lead anodes, 1 kw.-hr. pound of cathode copper.

Sulphur consumption, 0.5 lb. per pound of cathode copper.

Neutral electrolyte only will need to pass through the reducti towers.

About 1 per cent. per day of the total solution volume must be s over iron to control fouling.

About 12 per cent. of the total copper produced will pass through cement-copper side cycle.

The circulation used in the test-plant leaching tanks (2 gal. minute per ton of ore) is undoubtedly much higher than necessary. It probable that a small part of this volume would give equally go extraction results.

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N OF THIS PAPER IS INVITED. It should preferably be presented in person at th
, September, 1916, when an abstract of the paper will be read. If this is impossible,
n writing may be sent to the Editor, American Institute of Mining Engineers, 29 West
York, N. Y., for presentation by the Secretary or other representative of its author.
rangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion
r should preferably be in the form of a new paper.

ent of Flotation in the Clifton-Morenci District, Arizona

BY DAVID COLE,* EL PASO, TEXAS

(Arizona Meeting, September, 1916)

time flotation appeared upon the metallurgical horizon in e writer, under the direction of Dr. Ricketts, was engaged in and enlarging the No. 6 Concentration Plant of the Arizona at Morenci, and the work had been in progress nearly a year nspiration experiments with flotation had disclosed the revolu- centration that was at that time impending.

plified flowsheet worked out for the remodeling of the Morenci been based upon the removal of the freed metal in a minimum stages by treatment upon tables equipped with Butchart latter being adapted to accomplish both classification of feed l of the metal at one operation, substantially as described in s paper.[1]

heme of treatment for the slime was based upon the well- that after copper sulphides, such as chalcocite and chalcopy- re reduced to a certain extremely fine state of comminution finitely beyond the reach of separation upon any of the con- devices then known.

g drag-belt classifiers (which served as conveyors as well as , the overflow would be of the usual "slimes" class. Experi- shown that when these drag-belt overflows were properly t is, to about 5 to 7 per cent. solids in the feed under treatment) e sand, and especially the very fine but still granular sulphide would, if given a short distance to fall, quickly settle out in ition to yield an excellent recovery on vanners; a further ature would be that the very fine non-separable final overflow ight be discharged direct to tailing, thus conserving space ts so as to permit within the old building, without embarrass- ncrease of capacity required.

ons that are obvious, this kind of feed preparation for the slime e successfully accomplished in any form of pointed boxes or

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ing Engineer.

ment of the Butchart Riffle System at Morenci, *Trans.*, vol. 51, p. 405

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spitzkasten device, therefore a further elaboration of the drag-belt ide was worked out as the best method for accomplishing the separatio This machine was known as a colloid separator and is shown in Figs. and 2. It works on the premise that nearly all of what may be calle

FIG. 1.—DRAG-BELT SEPARATORS IN MILL OF ARIZONA COPPER CO.

FIG. 2.—ANOTHER VIEW OF DRAG-BELT SEPARATORS.

ponderable material in the thinned pulp, falling but 2 in., will lodge up the belts and be quickly removed, while the flocculent slime material w remain in suspension and go away with the overflow. In this way a fe is prepared for the vanners containing a maximum amount of the ve fine but granular sulphides and a minimum of colloidal material; at t

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same time the drag-belt overflow product contains a minimum of sulphide particles and a maximum of flocculent slime or colloidal pulp. This final overflow was found in practice to be approximately two-thirds of the total tonnage handled by the belts and treatment upon vanners was, as far as copper recovery was concerned, devoid of beneficial results. The copper escaping in this overflow was in the form of extremely fine chalcocite, bornite with a little pyrite and chalcopyrite, together with oxidized, and water-soluble, copper salts. Taken together, these gave the overflow a copper tenor of from 1 to 1.3 per cent. in a ton of dry material, thereby

FIG. 3.—DORR THICKENER OF 130-FT. DIAMETER AT ARIZONA COPPER CO.'S PLANT.

accounting for the larger part of the tailing losses. However, this overflow, with its ultrafine and otherwise handicapped copper-bearing material, was practically beyond the reach of any concentrating machine at that time known, and could therefore go to tailing. A Dorr thickener[2] 130 ft. in diameter, the first one of such large size, was devised to recover the water from this overflow before it was allowed to go to waste. (See Fig. 3.)

By this plan, the treatable portion of the slime could be handled by the complement of vanners already installed in the company's mill. This arrangement, in conjunction with the saving in floor space, resulting from the introduction of the Butchart riffle, made it easily possible to double the capacity of the plant under practically the original roof. While it did not promise to recover a larger percentage than usual of the

[2] Described in *Engineering and Mining Journal*, Vol. 100, No. 4, p. 131 (July 24, 1915).

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truly slimed copper, it did promise to give the best possible results i one-half the space.

By the time this flow sheet had been put into practical operation, th Inspiration experiments were attracting widespread attention and flota tion was beginning to be taken seriously as a concentration agent for th handling of copper ores. It was, however, still regarded as an auxiliar process and was thought to be inapplicable to ores carrying an exces of talc or clay like the Clifton-Morenci ores. Some experiments with th Elmore process on these ores in former years had been unsuccessful an laboratory work which we did later on a very small scale with the meage information available seemed to corroborate this theory. Space wa reserved, however, in a proper place in the mill building, for the installa tion of flotation equipment in case further development should prove it adaptability—at least for some appreciable portion of the tonnage.

Meanwhile, the Inspiration company had built the 600-ton "pilot" mill in which the new process was rapidly graduating from an auxiliar into the main method of separation, in the manner so fully and inter estingly described by Dr. Gahl.[3] Starting with semi-mysterious com pounds, Inspiration had soon found that simple flotation reagents wer equally efficacious. Through the kindness of Mr. Mills, I secured a drun of cresylic acid and some pine oil with which to try a few experiments a Morenci.

The tailing from the No. 6 Concentrator was at that time discharge into Morenci Canyon and cascaded along for about 1 mile before bein taken into a flume to be carried to the impounding dams. The creek be was rough and steep, inducing great agitation of the pulp and resultin in the production of large amounts of white froth which floated down th stream. This froth carried no concentrations of copper minerals, but thought it might be possible to change its character and produce a miner froth by the use of flotation reagents introduced where the tailing le the mill, and thus possibly secure from the natural situation afforded som benefit at little cost.

A small can of the cresylic acid was arranged to drip into the tailing launder at a point where the tailings made the initial plunge into the cree bed. The results were instantaneous and very gratifying. Black frot began to collect in eddies and float downstream for a few yards to a secon plunge where we were greatly surprised to find that it became white agai on account of the instant dropping of the metallic load. Feeding th reagent into the stream immediately above the second plunge would n cause a mineral froth to rise as in the first plunge, and the reason w finally located as being the effect of a town sewer which was dischargi under the surface into the creek between the two pools; the sewage effec ively killed the metal-carrying capacity of the froth.

[3] *History of the Flotation Process at Inspiration*, in this *Bulletin*.

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Cresylic acid was then added to the feed of a regrinding Hardinge mill which was discharging into a long drag-belt classifier. The results were again most encouraging; black mineral froth began immediately to appear and to collect in large volume upon the relatively still water in the drag-belt trough. This rough froth concentrate was found to assay over 45 per cent. copper; the product contained 35 per cent. insolubles, mostly in the form of coarse sand, mechanically suspended in the froth and easily separated by screening. It was found that 1 per cent. of the copper in the froth concentrate was in oxidized form; 22 per cent. of the concentrate was too coarse to pass a 100-mesh screen, and this portion carried only 0.87 per cent. copper, while the minus 100-mesh material carried 46 per cent. copper, 1.32 per cent. of which was oxidized, and but 16 per cent. of insolubles. The high grade of the froth concentrate was astonishing, showing that chalcocite and bornite predominated in it.

A few days later I made a bank of tube grates, consisting of six parallel 1-in. pipes made up with return bends. The pipes were drilled full of small holes and were wrapped with cotton blanket tied with spirally wound wire. This tube-grate air filter was put into the drag-belt trough as deeply as possible, without touching the belt. Coarse sand could pass through between the grates and be removed by the belt underneath. The pipes were supplied with compressed air for the purpose of creating additional froth, and it was found that the product made, without further treatment, assayed 40 per cent. copper, 1.14 per cent. of which was oxidized, and that it carried but 20.4 per cent. insolubles.

Plans for a small frothing machine of the mechanical-agitation type were immediately made, and on July 20 the apparatus was tried with the colloid separator fines as feed, with the following remarkable result: Feed, 2.32 per cent. copper, of which 0.62 per cent. was oxidized; the concentrates produced assayed 20.4 per cent. copper, of which 1.18 per cent. was oxidized; the tailing carried 0.52 per cent. copper, of which 0.38 per cent. was oxidized, leaving only 0.14 per cent. sulphide copper as the rejection of the machine. This was an extraction of more than 79 per cent. of the total copper, and more than 92 per cent. of the available (sulphide) copper. This showed clearly that much could be expected in the application of the new process to Morenci ores.

The Cananea Consolidated Copper Co., in Sonora, Mexico, had been experimenting with the use of some Flinn-Towne pneumatic flotation units in its concentrating department. The plant had been temporarily shut down on account of revolutionary troubles, and arrangements were made by Dr. Ricketts for the removal of one of these units to the No. 6 Concentrator at Morenci. The apparatus was installed under the direction of the Flinn-Towne people in the space reserved for flotation and was operated for several weeks with very gratifying results as to recoveries. These experiments demonstrated clearly that the flotation process

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would be suited to the recovery of slimed copper sulphides in the Morenci ores. But the Flinn-Towne units were not of a size suitable for use in the equipment of a large plant, or for the handling of large tonnages, except by using a great number of them. It was thought that their capacity could not be enlarged to advantage because of the difficulty with the air-emitting medium used, which was in circular-disk form with central discharge. These disks could not be made larger in diameter

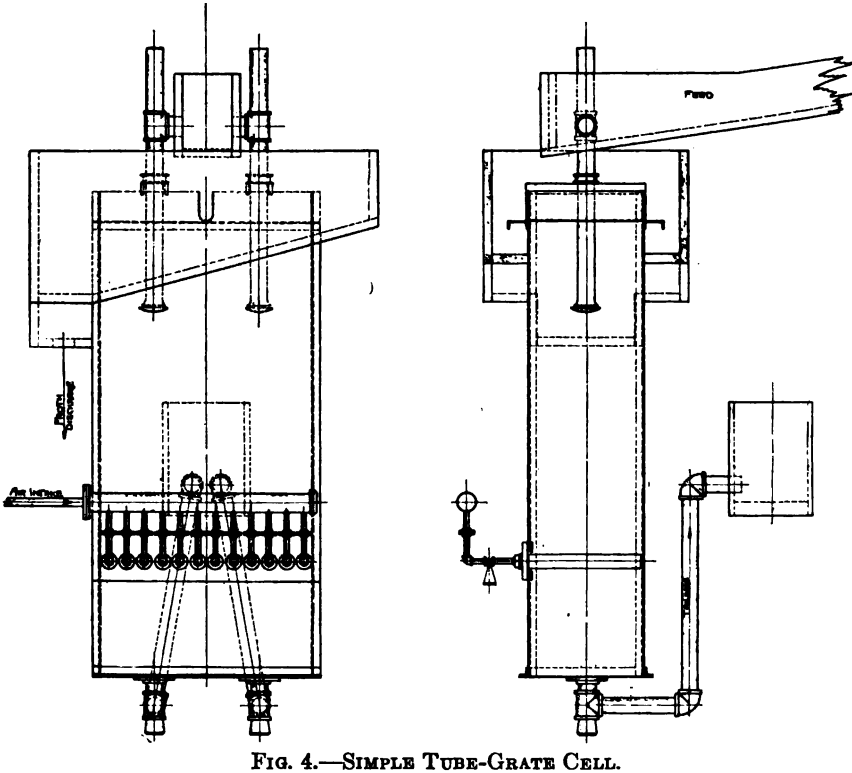

FIG. 4.—SIMPLE TUBE-GRATE CELL.

without increasing the difficulty coming from "blinding" of the air-emitting surfaces, through the lodging of coarse particles upon them, and from the formation of vortices by the larger volumes discharged through the single opening in the center which entrained froth with the reject.

The tube-grate idea previously tried in the drag-belt tank seemed to be a better way to admit the air because nothing could lodge upon the air-emitting elements to blind them, and constriction of the passage for the pulp and water would be avoided. This tube-grate idea therefore formed a basis on which to design units of large capacity for practica

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mill work and especially to solve the limited-space problem at No. 6 Concentrator. Accordingly, a tube-grate cell was installed, in January, 1915. This simple cell is shown in Fig. 4. It was used for some time to demonstrate the tube-grate idea and served as a "cleaner" in the subsequent work done with a full-sized machine.

The demonstration of the new tube-grate cell was such a success that a three-stage machine, to have a capacity of 400 tons per day, called the "C-B" machine, was designed and made. Another one of the same kind and size was made concurrently for the Inspiration Consolidated Copper Co., and both of them were started in operation early in March. The Inspiration machine, which is illustrated in Dr. Gahl's paper, gave good results, proving the design to be substantially correct. The Morenci machine was working in corrosive water, which resulted in the formation of rust on the steel tubes and the gradual closing of the openings. The air supply was found to be contaminated with grease and oil from the blower bearings, entrained muddy water in the air, etc., which closed the pores of the air filter from the inside. Some delay was experienced in overcoming these difficulties. That they were of minor importance was proved by the almost uninterrupted success and approximately perfect operation of the duplicate machine operating concurrently at Inspiration.

The C-B machine at Morenci made large volumes of very rich froth and had become immediately profitable by reason of its being able to handle a large tonnage and save copper which would be otherwise beyond the reach of concentration. It was therefore kept in operation as a profit maker, even though working under the handicap of partially clogged tubes, blower troubles, etc. The daily tonnage handled during the month of April, 1915, was from 125 to 390 tons per day, with an average of 209; the recovery made by the machine was from 35 to 79 per cent. of the sulphide copper present in the feed, with an average for the period of 65 per cent.

The blower used was an old one borrowed from the mining department, where it had been used in ventilation. It was designed for not more than 3½ lb. pressure, and the developing of 6 lb. in it deflected the shafts and caused the impellers to rub upon the sides of the machine, which had to be water-jacketed to keep down the heat developed. It was much larger than necessary, and a great excess of air was blown off from open valves. Therefore, no record of the amount of air used or power required, could be even approximately obtained.

In spite of these minor difficulties it was proved: That flotation could be applied to these ores with great advantage; that the copper in the mill tailing could be reduced to 0.50 per cent. (of which 0.25 to 0.30 per cent. was oxidized and beyond the reach even of flotation); that this result could be improved by finer grinding in the Hardinge mills; that the

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TABLE 1.—*Detailed Results of Tests on C-B and Callow Flotation Machines for Month of July*, 1915.

SCREEN ANALYSIS C-B MACHINE

Mesh	Machine Tails							Machine Concentrates							Vanner Tails						
	Solids		Assay			Copper		Solids		Assay			Copper		Solids		Assay			Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidised Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidised Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidised Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total
+ 20	.....	.....	.....	....	.....	.....	.....	.....	.....	.....	....	.....	.....	.....	.....	.....	.....	....	.....	.....	.....
+ 28	1.1	1.1	0.77	....	0.85	1.41	1.41	.....	.....	.....	....	.....	.....	.....	1.4	1.4	0.66	....	0.92	2.11	2.11
+ 35	3.8	4.9	0.65	....	2.47	5.51	5.51	.....	.....	.....	....	.....	.....	.....	4.4	5.8	0.60	....	2.64	6.04	8.15
+ 48	8.0	12.9	0.79	....	6.32	10.49	16.00	0.5	0.5	2.22	....	1.11	0.04	0.04	9.9	15.7	0.60	....	5.94	13.61	21.76
+ 65	9.9	22.8	0.80	....	7.92	13.14	29.14	0.6	1.1	14.72	....	8.83	0.32	0.36	11.0	26.7	0.51	....	5.61	12.85	34.61
+100	14.7	37.5	0.87	....	12.79	21.23	50.37	1.6	2.7	15.82	....	25.31	0.90	1.26	16.6	43.3	0.48	....	7.97	18.25	52.86
+150	11.2	48.7	0.79	....	8.85	14.69	65.06	7.9	10.6	20.16	....	159.26	5.64	6.95	10.9	54.2	0.37	....	4.03	9.23	62.09
+200	6.9	55.6	0.67	....	4.62	7.67	72.73	10.6	21.2	23.99	....	254.29	9.08	16.03	6.3	60.5	0.37	....	2.33	5.34	67.43
−200	44.4	100.0	0.37	....	16.43	27.27	100.00	78.8	100.00	29.83	....	2,350.60	83.97	100.00	39.5	100.0	0.36	....	14.22	32.57	100.00
Total...	60.0	.....	.....	....	60.25	100.00	.....	100.00	.....	.....	....	2,799.40	100.00	.....	100.0	.....	.....	....	43.66	100.00	.....
												Fe	Ins.								
Check..	.....	.....	0.61	0.24	.....	.....	.....	.....	.....	27.84	1.44	19.8	18.8	.....	.....	.....	0.41	0.25	.....	.....	.....

Mesh	Vanner Concentrates							Table Tails							Table Concentrates						
	Solids		Assay			Copper		Solids		Assay			Copper		Solids		Assay			Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidised Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidised Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum Per Cent. Wt.	Per Cent. Total Cu.	Per Cent. Oxidised Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total
+ 20	0.2	0.2	2.02	....	0.40	0.04	0.04	.....	.....	.....	....	.....	.....	.....	0.7	0.7	1.53	....	1.07	0.11	0.11
+ 28	0.6	0.8	3.63	....	2.18	0.24	0.28	0.9	0.9	0.65	....	0.59	1.26	1.26	1.0	1.7	4.23	....	4.23	0.45	0.56
+ 35	1.6	2.4	7.26	....	11.64	1.28	1.56	3.1	4.0	0.66	....	2.05	4.36	5.62	2.0	3.7	7.75	....	15.50	1.66	2.22
+ 48	4.7	7.1	9.08	....	42.68	4.70	6.26	7.2	11.7	0.60	....	4.32	9.20	14.82	6.3	10.0	9.79	....	61.68	6.59	8.81
+ 65	8.2	15.3	10.90	....	89.38	9.84	16.10	9.8	21.0	0.58	....	5.68	12.09	26.91	14.3	24.3	10.20	....	145.86	15.58	24.39
+100	17.4	32.7	11.60	....	201.84	22.23	38.33	14.8	35.8	0.54	....	7.99	17.01	43.92	36.8	61.1	8.77	....	322.74	34.48	58.87
+150	17.6	50.3	11.60	....	204.16	22.49	60.82	11.7	47.5	0.44	....	5.15	10.96	54.88	24.0	85.1	9.79	....	234.96	25.10	83.97
+200	13.4	63.7	8.26	....	110.68	12.19	73.01	7.1	54.6	0.49	....	3.48	7.41	62.29	8.9	94.0	10.81	....	96.21	10.28	94.25
−200	36.3	100.0	6.75	....	245.02	26.99	100.0	45.4	100.0	0.39	....	17.71	37.71	100.0	6.0	100.0	8.97	....	53.82	5.75	100.00
Total...	100.0	.....	.....	....	907.96	100.0	.....	100.0	.....	.....	....	46.97	100.0	.....	100.0	.....	.....	....	936.07	100.0	.....
																			Ins.		
Check	.....	.....	7.06	1.91	.....	.....	.....	.....	.....	0.41	0.24	.....	.....	.....	.....	.....	8.96	1.40	43.6	.....	.....

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Mesh	Vanner Concentrates						Table Tails						Table Concentrates					
	Solids		Assay		Copper		Solids		Assay		Copper		Solids		Assay		Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total
20	0.4	0.4	2.42	....	0.15	0.15	0.5	0.5	0.88	....	0.44	1.10	0.8	0.8	3.67	....	2.94	0.34
28	0.6	1.0	1.81	....	0.17	0.32	1.3	1.8	0.64	....	0.83	3.18	0.5	1.3	5.91	....	2.96	0.69
35	1.4	2.4	3.12	....	0.66	0.98	4.2	6.0	0.57	....	2.39	9.16	5.2	6.5	9.59	....	49.87	6.51
48	3.3	5.7	6.35	....	3.18	4.16	10.4	16.4	0.47	....	4.89	21.41	17.7	24.2	9.59	....	169.74	26.31
65	6.0	11.7	10.40	....	9.48	13.64	32.4	44.3	0.44	....	7.04	39.04	40.0	64.2	8.06	....	322.40	63.92
100	16.0	27.7	12.81	....	31.13	44.77	11.9	100.0	0.47	....	5.59	53.04	21.8	86.0	8.57	....	186.83	85.72
150	15.0	42.7	12.10	....	27.57	72.34	7.8	52.1	0.44	....	3.43	61.63	8.2	94.2	8.87	....	72.73	94.20
+200	12.1	54.8	6.76	....	12.42	84.76	47.9	100.0	0.32	....	15.32	100.0	5.8	100.0	8.57	....	49.71	5.80
−200	45.2	100.0	2.22	....	15.24	100.0	....	....	....	....	39.93	....	100.0	....	....	....	857.18	100.0
Total...	100.0	....	....	....	100.0	....	100.0	....	....	....	....	....	....	....	....	....	....	....
Check...	....	....	6.40	2.13	....	....	....	....	0.45	0.25	....	....	....	....	7.84	1.44	Fe 21.0	....

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TABLE 2.—*Comparative Results Obtained in Operation of C-B and Callow Flotation Machines at Concentrator No. 6, Arizona Copper Company, Ltd., Morenci, Ariz.*

		C-B Per Cent.	Callow, Per Cent.
May, 1915			
Flotation machine tails	Total copper	0.72	0.81
	Oxidized copper	0.38	0.30
	Sulphide copper	0.34	0.51
Flotation machine concentrates	Total copper	38.19	24.82
	Insolubles	25.00	27.00
Vanner tails	Total copper	0.53	0.46
	Oxidized copper	*	0.18
	Sulphide copper	*	0.28
Vanner concentrates	Total copper	8.52	8.89
	Insolubles	*	30.60
June, 1915			
Flotation machine tails	Total copper	0.71	0.69
	Oxidized copper	0.30	0.28
	Sulphide copper	0.41	0.41
Flotation machine concentrates	Total copper	35.24	25.41
	Insolubles	23.80	26.00
Vanner tails	Total copper	0.41	0.42
	Oxidized copper	0.23	0.22
	Sulphide copper	0.18	0.20
Vanner concentrates	Total copper	9.35	9.51
	Insolubles	*	* ....

	C-B, Per Cent.	Callow
July, 1915		
Flotation machine tails—Total copper	0.61	0.66
Oxidized copper	0.24	0.23
Sulphide copper	0.37	0.43
Flotation machine concentrates—Total copper	27.84	24.80
Insolubles	18.80	24.60
Vanner tails—Total copper	0.41	0.37
Oxidized copper	0.25	0.15
Sulphide copper	0.16	0.22
Vanner concentrates—Total copper	7.96	6.40
Insolubles	*	* ....
Table tails—Total copper	0.41	0.45
Oxidized copper	0.24	0.25
Sulphide copper	0.17	0.20
Table concentrates—Total copper	8.96	7.84
Insolubles	43.60	* ....

* No assay.

	C-B	Callow
Daily tonnage rate, average for the month of July, 1915	479	139

SEE NOTE, p. 1621.

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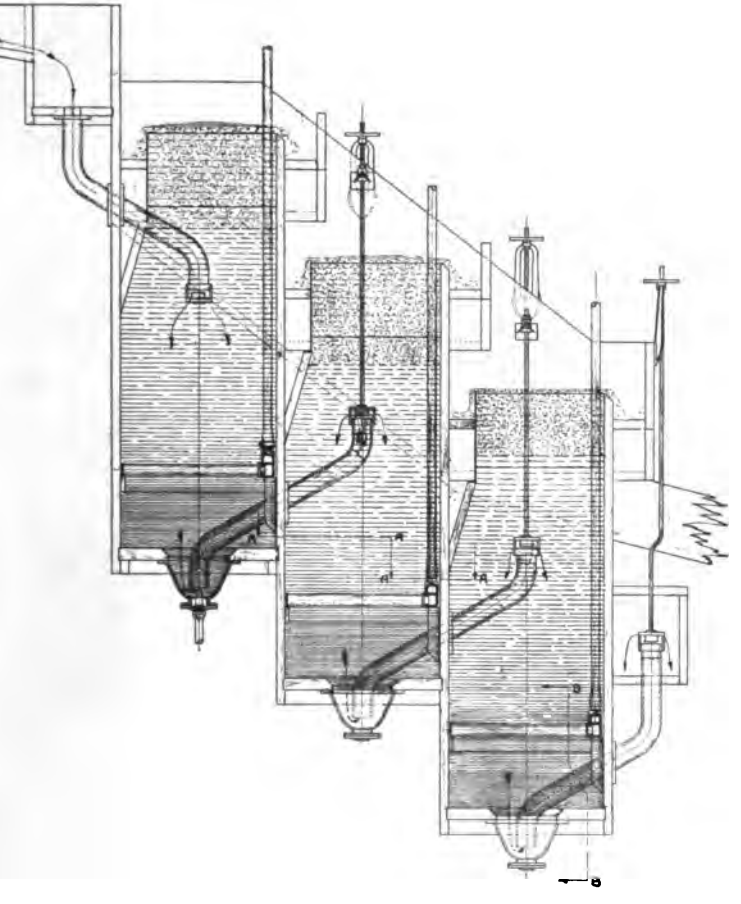
e reagents used elsewhere would apply. Further, a new type ic flotation machine, well adapted to the conditions at No. 6 or, had been successfully developed and its performance ed on a full-sized unit. The new type machine would not ay embarrassed by the oversize coming from Hardinge mills.



. 5.—Longitudinal Section of C-B Flotation Machine. Three Cells in Series.

drag-belt overflow being handled by the colloid separator a few of these new machines where all of the rich slimed sul- r would be taken out. Or the whole tonnage of reground

Table 2.—Without protection against "blinding" of the air-emitting l or "oversize" the C-B machine handled considerably more than three nage handled by the Callow in July, and did this without detriment rgical work.

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material produced in the Hardinge mills could go directly to the new typ frother in which the slimed copper sulphides would be removed. Th thoroughly frothed sands could then be treated on tables and vanne for the removal of the remaining sulphide particles which were to coarse to be separated by flotation. Since there would then be no en barassing losses in the slime part of the feed, these machines would wor efficiently, and a maximum mill recovery would result.

After it became evident that flotation would apply to Morenci or and before the value of the tube-grate idea was fully demonstrated, was decided to install and experiment with a standard Callow flotatio unit of 200 tons capacity, consisting of four rougher cells and one cleane This equipment was not received until after the full-sized C-B unit of 40 tons daily capacity had been installed.

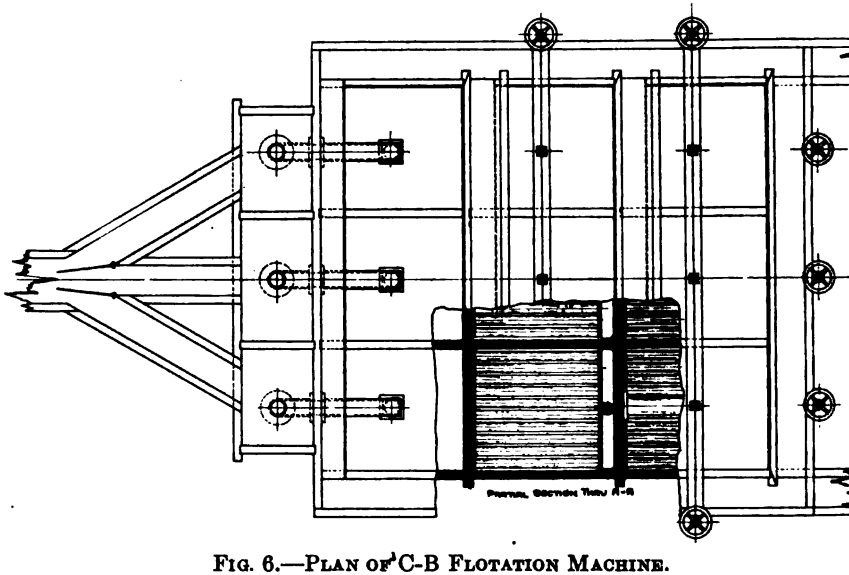

FIG. 6.—PLAN OF C-B FLOTATION MACHINE.

The Callow equipment was started in operation May 24, 1915, an competitive operation was carried on for about 3 months. The recoveri proved to be very much alike although the feed was not identical. Th Callow apparatus is not adapted to handle coarse particles of feed oversize, and had to be protected by a screen or spitzkasten. It w handle about one-half the normal tonnage of the C-B machine, occup ing the same mill space. A summary of the results obtained for th months of May, June, and July, 1915, also details showing the work f the month of July, are given in Tables 1 and 2. The performance shown to be substantially parallel as to quality of work done, but qui different as to quantity of tonnage handled.

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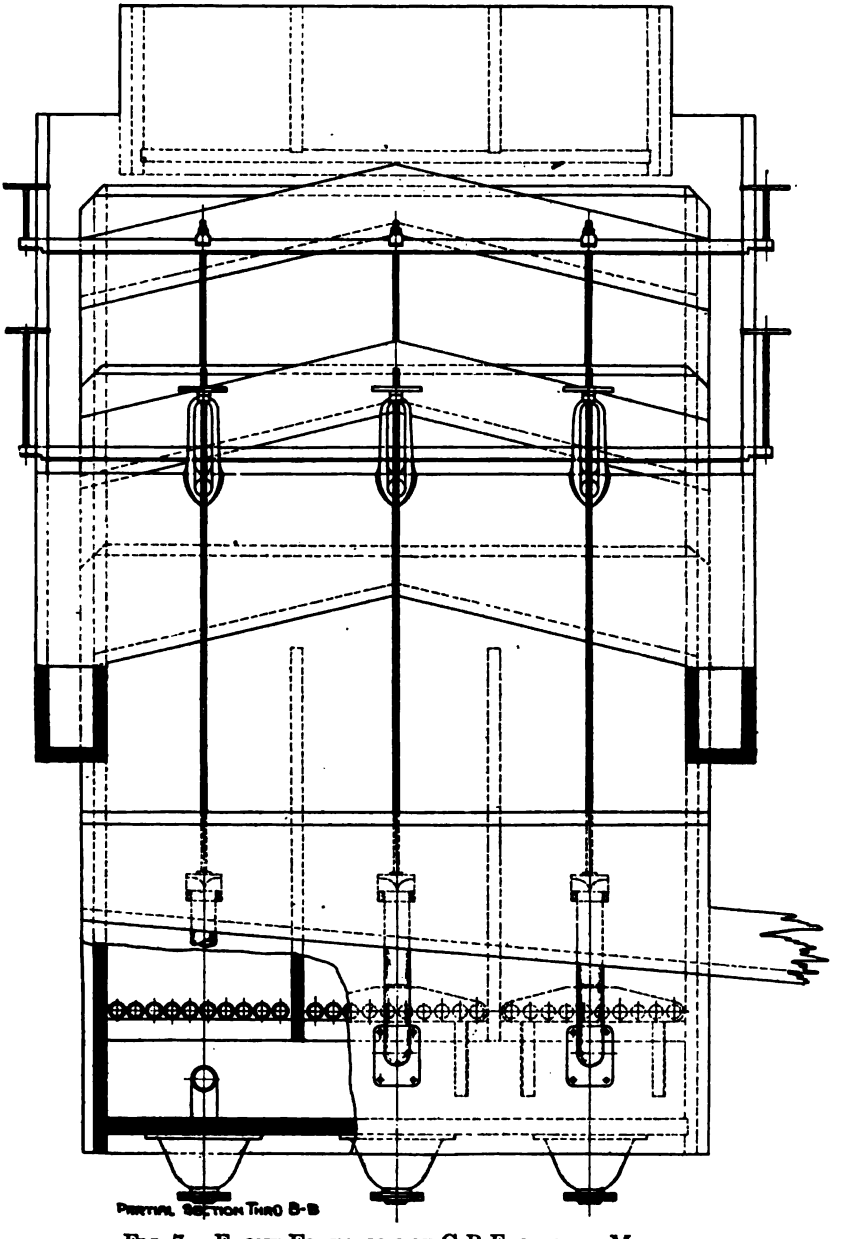

FIG. 7.—FRONT ELEVATION OF C-B FLOTATION MACHINE.

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As mentioned before, the air and power consumption was not deter mined in the C-B installation because there was no way to take eve approximately correct measurements. But the experience at Inspira tion, where the C-B and Callow systems were also being operated i parallel, showed that substantially the same amounts of air and powe were used by each system.

Experience suggested wider launders for froth, larger cleaner capacity and simplified tube-grate construction of the C-B unit. These ideas ar incorporated in the new design shown in Figs. 5, 6 and 7 in which it wi be noted that the machine is merely a stationary wooden box suitabl arranged to receive air-emitting tubes which are dropped in from the to

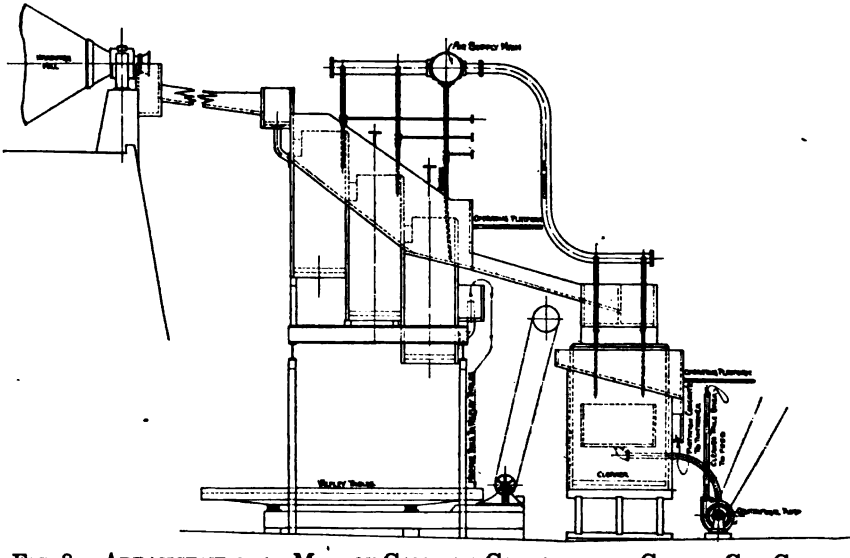

FIG. 8.—ARRANGEMENT AT MILL OF CANANEA CONSOLIDATED COPPER CO., CANANI MEX., USING THE C-B 3-STAGE FLOTATION MACHINE.

and rest upon ledges at the proper level in the pulp to be treated. Thes air-emitting elements are connected to the air supply by the use of rubb hose. They can be taken out or put in without cutting off the feed shutting down the machine, in case this should be of advantage. An a pressure of 5 lb. is required. The air-emitting surface in the C-B machi is more than twice the complete cross-sectional area of the frothing con partments, and even if a ridge of sand should lodge upon the extreme to of the tubes and partially cut off the air supply, the remaining unob structed area would still be larger than the whole cross-section.

Fig. 8 shows the improved form with cleaners as arranged in the m of the Cananea Consolidated Copper Co., and is typical of the arrang ment adopted for the later models. There are eight units in this install tion. They are operated under conditions varying greatly as to quali

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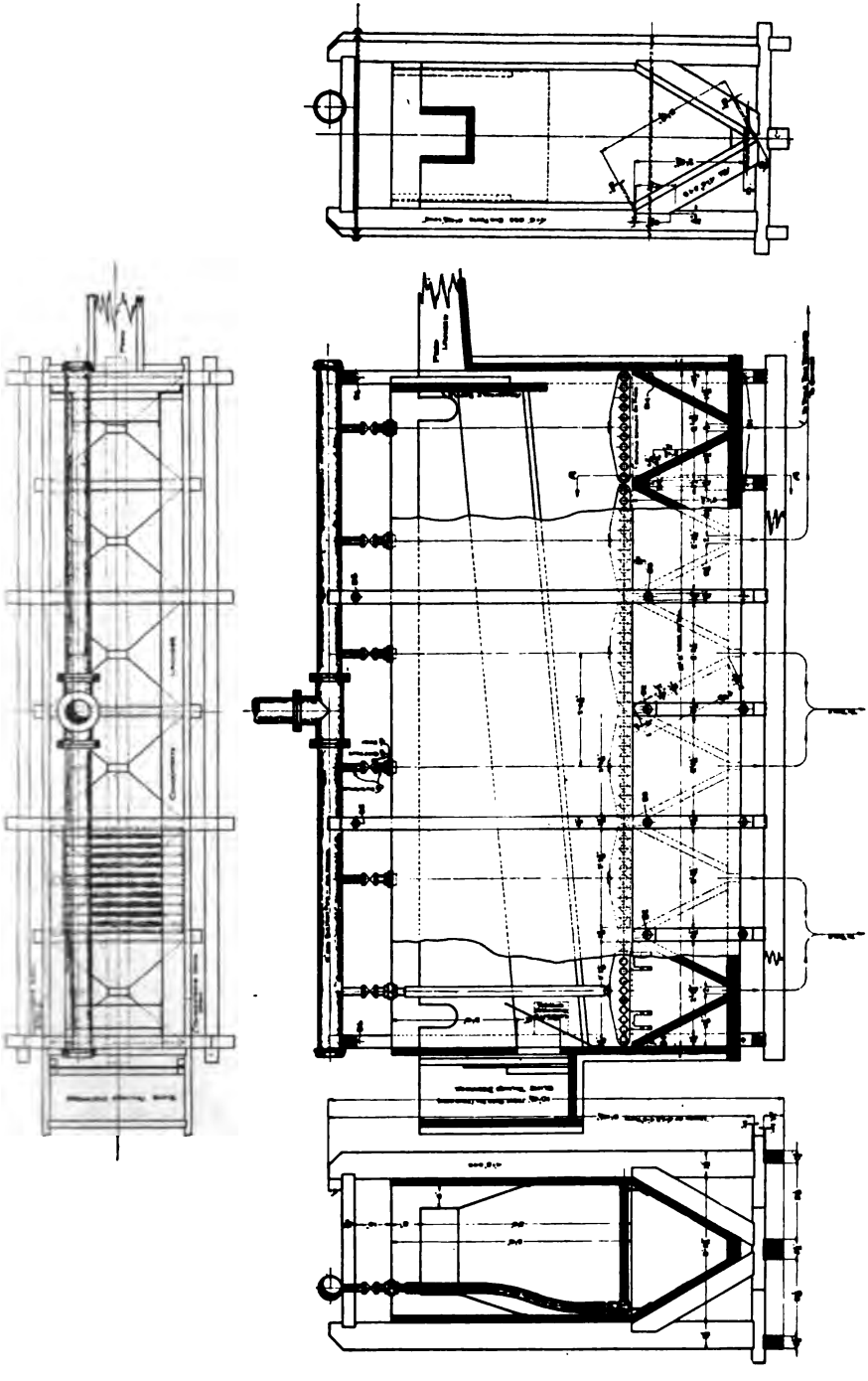

FIG. 9.—SPITZKASTEN FROTHER EQUIPPED WITH TUBE-GRATE AIR FILTER.

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of ores, there being a wide range of iron and copper sulphide condition and all conditions of oxidation in the ore handled. The operation of th Cananea machines has been repeatedly interrupted by the internal stri in Mexico and no deductions of value are at present available.

Fig. 9 shows an application of the tube-grate idea to a spitzkaste type of frother. One of these machines was made and installed in No. Concentrator at Morenci but was taken out before it was tried. seems to embody advantages of much promise in a frothing first flo sheet and will soon have a trial to determine its value in the simplifi concentration of ore that is amenable to flotation.

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ION OF THIS PAPER IS INVITED. It should preferably be presented in person at the ing, September, 1916, when an abstract of the paper will be read. If this is impossible, on in writing may be sent to the Editor, American Institute of Mining Engineers, 29 reet, New York, N. Y., for presentation by the Secretary or other representative of its ss special arrangement is made, the discussion of this paper will close Nov. 1, 1916. on offered thereafter should preferably be in the form of a new paper.

History of the Flotation Process at Inspiration

BY RUDOLF GAHL,* PH. D., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

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*Metallurgist in charge of Concentrator, Inspiration Consolidated Copper Co.

THE history of flotation in America is very short, at least as far as the large-scale application of the process is concerned. It is remarkable how many important developments have taken place in the last few years and are already being extensively utilized. What was new a year ago, seems almost commonplace now. For this reason it is with hesitation that I follow the suggestion of Dr. Ricketts, President of the Institute, to describe the experiences of the Inspiration Consolidated Copper Co. with the flotation process.

TESTS CONDUCTED IN SMALL TEST MILL

When the Inspiration company first decided to build a concentrating plant, nothing was known about flotation, and the process was to be gravity concentration pure and simple.

Demonstration Tests Conducted by Minerals Separation Co.

While plans were being prepared by H. Kenyon Burch (who had been intrusted with the design and construction of the concentrator) the Minerals Separation Co., a concern at that time little known in America, asked and obtained permission to demonstrate the value of its flotation process for Inspiration ore. As a consequence, a small 50-ton flotation machine of standard design was added to the company's test plant and started to operate in the beginning of 1913. This marks the beginning of flotation at Inspiration.

The results obtained with this machine, which was operated by members of the Minerals Separation staff, so far surpassed what this company anticipated that it was decided to continue flotation tests for this purpose, and two of the flotation experts of the Minerals Separation Co., I. L. Greninger and G. A. Chapman, were retained. L. R. Wallace, now superintendent of the Miami works of the International Smelting Co., was at that time metallurgist of the Inspiration company and in that capacity took an active part in these tests. Great credit is due to him for his quick recognition of the possibilities of flotation.

Flotation Tests Conducted by Inspiration Co.

The tests led to the conclusion that it would be advisable to incorporate flotation into the concentrating process. Doubt existed only as to the extent to which this should be done. The first design brought out by Mr. Burch only called for flotation treatment of the concentrator tailings. While the tests were progressing, it became more and more evident, however, that flotation should form a more essential part of the milling process, and it was finally decided to leave out all the complica-

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n are usually adopted in gravity concentration plants for the recovering as much as possible of the mineral values of the o rely on flotation alone for this purpose.

Sampling of Orebody

ecision was reached only after it had been established by ests that the orebody as a whole was suited for flotation treat- was found, it is true, that a portion of the ore in the mine was for flotation by the process in question, but the amount of s established to be only a small fraction of the total. Besides, roved that the ore contained in this fraction lost its refractory when mixed with the rest of the ore.

eninger gives the following description of the manner in which were carried out:

e lots, each amounting to about 10 tons and representing from were blasted from the back and sides of the drifts and after ushing and grinding treated in the small flotation machine above.

esults were very erratic, some showing good recovery with a of concentrate, while others returned concentrate of a very low also showed a low extraction.

a considerable number of tests had been made, it was found od results were always obtained when treating ore of a schist ile poor results resulted when the gangue was the altered and granite which forms a part of the Joe Bush orebody.

on samples were taken from various parts of the mine for the determining the extent and amount of this granite ore. These aken from the drifts crossing the orebody from south to north, the granite ore and continuing along each drift until the schist s encountered.

of these tests the results were uniformly good while treating ore and poor while treating the granite ore.

quently, another series was inaugurated with the view of g the amount of granite ore that could be mixed with the schist interfering with the treatment of this mixed ore by flotation. us percentages of granite ore were mixed with clean schist and e treated by flotation. It was found that 10 per cent. of granite hange in the behavior of the ore in the flotation plant and when t. of the granite was mixed with the schist ore only a slight was noted, this difference consisting in a slight lowering of the e concentrate with a corresponding falling off in extraction."

nditions described by Mr. Greninger are well illustrated by a ade by C. E. Arnold of the mine-engineering staff of the com-

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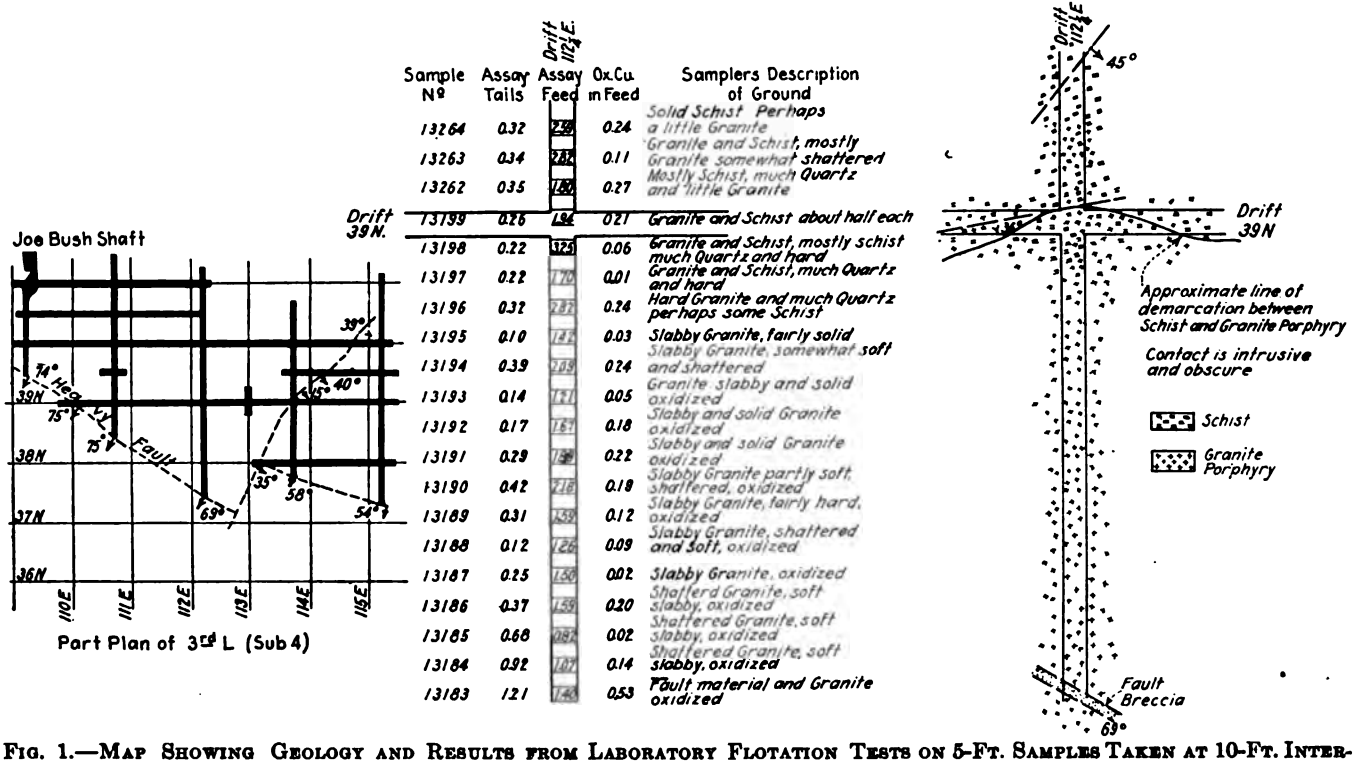

FIG. 1.—MAP SHOWING GEOLOGY AND RESULTS FROM LABORATORY FLOTATION TESTS ON 5-FT. SAMPLES TAKEN AT 10-FT. INTERVALS ALONG DRIFT BETWEEN 3740 AND 3935 N.

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pany, which represents laboratory flotation results on samples taken from a drift in the Joe Bush orebody (Fig. 1). The results show clearly that the granite by itself does not interfere with flotation, but only the fault material, evidently corresponding to what Mr. Greninger terms kaolinized granite.

Other Results Obtained in Small Test Mill

The test conducted in the small test plant established in a general way the physical conditions under which Inspiration ores could be treated advantageously; for instance, it was decided that raising the temperature did not improve the results obtained in proportion to the extra expense of such procedure.

As far as flotation oils are concerned, those in charge of the tests came to the conclusion that cresylic acid (98 per cent. pure) should be used as the main flotation agent, and should be supplemented by crude turpentine. As the most important result brought out by these tests, I consider the discovery that the flotation agents may profitably be added in the grinding machines, while it had formerly been the customary practice to add the "oils" in agitating tanks especially provided for the purpose. This discovery had an important bearing on the later developments in the Inspiration milling practice, inasmuch as it paved the way for the use of much heavier oils; for instance, coal tar which it is impossible to amalgamate thoroughly with the pulp in agitating tanks. Mr. Chapman, I think, made this important discovery during this period (U. S. Patent 1,102,874).

The details of the flow sheet to be followed were left to be decided by tests on a larger scale. Only this much was settled: That no concentration of any kind, either flotation or gravity, should be attempted before the ore was reduced to the fineness required for the flotation process. This decision was brought about by the fact that excellent recoveries were obtained when this procedure was followed, and was, of course, also strongly urged by considerations of simplicity in the milling process and cheapness of milling operations.

In these tests a 50-ton flotation machine of Minerals Separation Co. Standard type was used. In the general design, it was similar to the larger machine later put in use. It had eight agitating compartments with propellers of 12-in. diameter revolving with a peripheral speed of from 1,200 to 1,400 ft. per minute.

Tests in Large Test Mill

Flow Sheet

In order to test on a large scale the points already settled in the small test mill, and to decide the points left undecided by the small-scale experi-

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mefits, a 600-ton test plant was built and put into operation at the begi ning of January, 1914. It was my privilege to conduct those tests i coöperation with Mr. Greninger, who represented the Minerals Separatio Co., and with the representatives of other concerns, who in the course time decided to submit flotation machines to the consideration of th Inspiration company. The flow sheet of this 600-ton test mill was e tremely simple. It is illustrated in Fig. 2. The ore after being crushe to the desired fineness by machinery, which was tested out at the sam time, was sent to tables; from these the concentrates went to the conce trate bins, while the tailings passed on to an 8-compartment flotatio

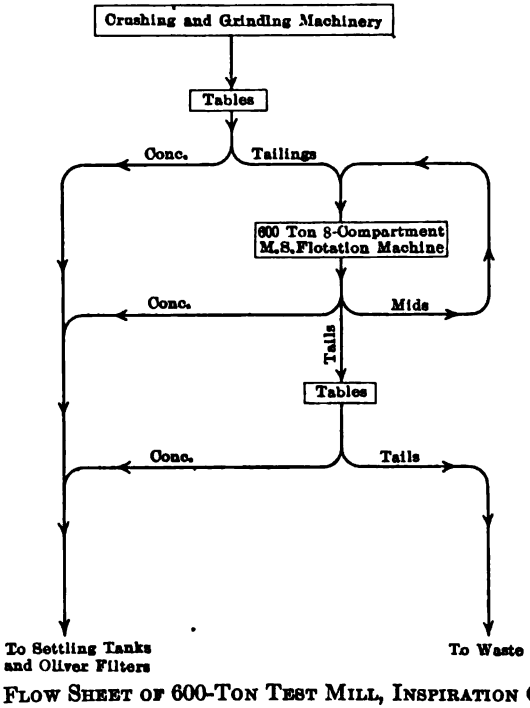

FIG. 2.—FIRST FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATE

machine of the Standard Minerals Separation type with 24-in. stirre Later a 12-compartment machine of the same type was added a operated in parallel with the 8-compartment machine. The tailings fro the flotation machines were passed on to other tables, the concentrat from which, combined with the concentrates from the upper tables a the flotation machine, went to the concentrate bins, while the tailin went to waste. The flotation machine was operated in the standa manner, the feed being introduced into the first agitating compartme passed to the first spitzkasten, drawn to the second agitating compa ment, sent from there to the second spitzkasten and drawn to the thi

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agitating compartment, etc., the agitators forming the necessary suction for the transportation of the pulp from the spitzkastens to the following agitating compartments. A final concentrate was made from one or more spitzkastens, while the concentrate from those remaining was considered a middlings product and was returned to the head of the machine for retreatment.

This flow sheet was extremely simple, but after a while even this was considered too complicated, inasmuch as the necessity of table treatment ahead of flotation treatment was doubted.

Discussion of the Value of Preliminary Table Treatment

The advantages pointed out in favor of the preliminary table treatment were about as follows: Since the tables would make a certain recovery, an impoverished flotation feed would result and assist the flotation machine to make a tailings product low in copper. However, the validity of this argument was doubted for the following reasons:

In the first place, if complications are to be avoided, the preliminary table treatment has to be of a comparatively rough character; the refinements of hydraulic classification have to be dispensed with, as by its use more water would be introduced into the pulp than would be advisable for the following flotation process. It must be taken into consideration that the pulp delivered to the tables already contains about 3 tons of water to 1 ton of solid matter, experience having shown that the grinding machines, consisting of either ball or pebble mills, deliver a product of the desired fineness, about 1 or 2 per cent. on 48-mesh, when the consistency of the overflow from drag-belt classifiers, working in conjunction with the grinding mills, was carried at about this figure; experience also showed that this consistency is suitable for the flotation treatment, while greater dilution of the pulp resulted in an increased copper loss. The logical way out of this difficulty would be to introduce settling tanks, for which, however, the modern mill designer has a just abhorrence; at least, when they are to be placed in the middle of the mill. Neither the liberal use of dressing water nor a low tonnage rate would be permissible for the same reason. On this account, a high recovery could not be expected from a preliminary table treatment. During our tests it averaged around 33 per cent.

Furthermore, the assumed fact, that an impoverished feed results in a better recovery of the flotation machine, was strongly doubted by some of the flotation experts, who claimed that in order to form a froth which had the necessary carrying power for mineral, it would be better to leave as much mineral as possible in the feed to the flotation machine. In other words, their advice was to leave out the tables in order to permit the flotation machines to do more efficient work.

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There is no doubt, however, that tables will catch a certain amou of mineral (especially coarser grains) which will escape in the flotati process. For this reason, tables have to be used to insure the highe recovery,[1] but it seems that tables following the flotation treatme would make up for this deficiency of the flotation machine better th tables ahead, inasmuch as they would not work under the disadvanta known to every millman, of receiving too rich a feed. It is well kno that it is possible to make a much lower table tailing working on fe containing a small percentage of copper than on a feed rich in copp From theoretical speculations, therefore, no valid reason was advanc that increased recovery should result from table treatment ahead the flotation treatment.

The next argument advanced by the supporters of a preliminary tal treatment was that it would result in an improvement of the grade the general concentrates, inasmuch as it would be possible to make a ve clean product on the tables, thereby raising the general average of t concentrates. There is evidently some sound foundation for this poi how much, could not well be investigated without resorting to actu experiments.

A third point seemingly in favor of preliminary table treatment this: Flotation concentrates offer more or less difficulty in mechani handling; vacuum or pressure filters have to be resorted to for t purpose. It was pointed out, therefore, that if a certain percentage the total concentrate could be saved on the tables, and a table conce trate produced containing only a small amount of the slime, so troub some in filter treatment, it might be a decided advantage; the quantity concentrate to be handled on filters might be materially reduced a economies effected in this manner. At the same time, experience sho that the flotation concentrate resulting under those conditions, becau it contains less of the coarser sand, is more difficult for the filters to hand than it would be were the sand left in. Very likely, therefore, no decid improvement in the handling of the concentrates would result from t separation of the table concentrates from the flotation concentrates, least under the conditions prevailing here.

The same objections do not hold true for tables when used after t flotation process, as in that case no limitations are imposed regardi the hydraulic and dressing water, and the tables cannot help but ma an additional small recovery of concentrate. On account of the fact th the table feed is necessarily low in copper, the tailings will be cor spondingly low. In all cases, therefore, where flotation treatment do not make a full recovery, or practically so, of mineral values, tabli after flotation seems imperative.[2]

[1] This applies only to the flow sheet under discussion.

[2] This remark again refers only to the flow sheet under discussion.

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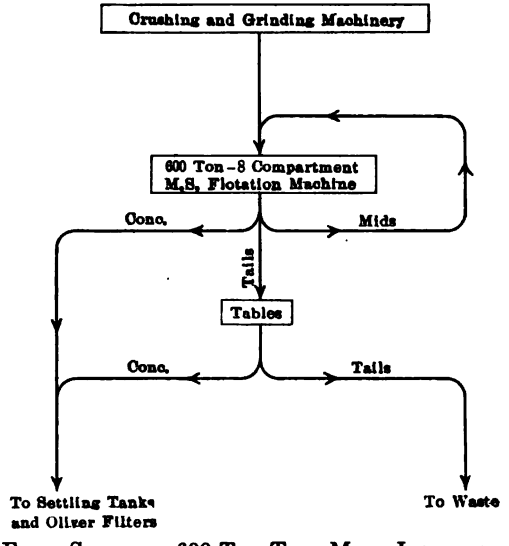
gh such theoretical considerations were indulged in, no deci-
ased on them. To settle the main points in question, a series
s carried out. On alternate days, the "upper" tables were by-
g. 3) while on the remaining days, they were utilized (Fig. 2).



COND FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATED.

1.—*Comparison of Efficiency of Flow Sheet No. 1 (Fig. 2) and Flow Sheet No. 2 (Fig. 3)*

Description	Flow Sheet	
	No. 2	No. 1
l feed	1.72	1.67
tation machine feed	1.72	1.32
tation machine tails	0.46	0.43
er table tails	0.29	0.30
neral concentrates	32.71	31.72
per tables + flotation machine	.....	75.30
tation machine	74.30	
ecovery on lower tables	9.40	7.40
ry	83.70	82.70
deck on lower tables	25.70	23.20
deck on upper tables	.....	87.00
tion, cresol + turpentine in pounds per ton	0.95	0.82

s.—The recoveries are calculated from the feed and tailings assays and general concentrates.

IONS.—Roughing on upper tables reduces the tailings assay of the flota-e slightly, while it does not seem to affect the assay of the lower tables Increased recovery results from the use of the lower tables.

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Table 1 gives the résumé, on the strength of which it was decided ot use secondary tables only and to leave the preliminary table treatment (ahead of the flotation treatment) out of the flow sheet to be adopted in the concentrator under construction. That this conclusion is justified seems to be borne out by the following facts:

In January and February, 1914, the lower table floor was not in operation. For this reason, the flotation tailings were not treated on tables. There was, however, table treatment ahead of flotation, the following results being obtained: Table recovery, 39.42 per cent.; flotation recovery, 33.35 per cent.; total recovery, 72.77 per cent. From March to August, 1914, various flow sheets were tried; tables were sometimes used preceding and sometimes following flotation, and sometimes in both places. The figures on recovery are, for this reason, not comparable with the preceding ones. From September to December, 1914, no tables were used preceding flotation; the flotation tailings were, however, treated on tables. The results obtained were: Flotation recovery, 70.83 per cent.; table recovery, 7.51 per cent.; total recovery, 78.34 per cent.

Callow Flotation Installation

While the test mill was in operation, Mr. Callow, President of the General Engineering Co., advised the Inspiration company that he had invented and perfected a new flotation process which, in his opinion, would give as good, if not better, results than the machine of the Minerals Separation Co. As a consequence, arrangements were made to add a unit of machines of the Callow type. The flow sheet using Callow cells is illustrated in Fig. 4. It consisted of four rougher flotation cells, which served for the production of a low-grade concentrate, and another cell of the same construction which was supplied for the purpose of cleaning the concentrates made on the rougher cells. The tailings resulting from the cleaner cells were returned and mixed with the pulp entering the rougher cells. With the machine was furnished a Pachuca tank for the purpose of mixing flotation oil into the pulp entering the plant, should it be more desirable to do this in addition to or in place of feeding the oil to the grinding machines, as had proven useful both in the small and in the larger test plants. As a matter of fact, the use of the Pachuca tank was abandoned after several months' operation.

The Callow cells work on a different mechanical principle from that underlying the Minerals Separation Standard machine, inasmuch as no movable parts are used for the purpose of producing the fine dissemination of air with the pulp, which seems to be essential for any kind of flotation machine; while the Standard Minerals Separation machine depends on the operation of fast-revolving impellers, the dissemination is effected in the Callow machine by blowing air through the pores of a porous blanket, which forms the bottom of each cell.

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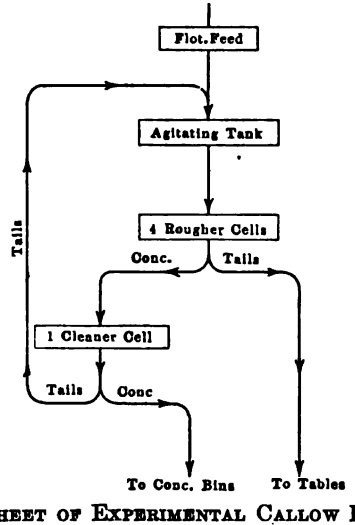
llow machine, illustrated in another paper, is ingeniously
ne feed enters at the upper end of the inclined bottom, while
are discharged through the float valve at the lower end,
trates overflowing at the top of the flotation tanks. The
om of the cell is set on an incline to make possible the treat-
d containing coarse sand. The movement of such sand par-
the feed to the discharge end is thereby accelerated.
chanical principle of aeration and agitation by the admission
ugh a porous medium, which has been termed "pneumatic
was undoubtedly discovered by Mr. Callow[3] and his asso-



.—Flow Sheet of Experimental Callow Flotation Plant.

pendently of other inventors, although an earlier patent
aken out for the same thing in England by Minerals Separa-
h Patent 10,929, May 3, 1910) and long before the installa-
Callow machine at this mill, Inspiration ore had been tested
rk by a pneumatic flotation machine constructed jointly by
nn and Towne.[4] The results of this test were so good that a
lant of this system was installed at Inspiration.

ow refers to his installation at the National Mill, Mullan, Idaho, April,
rst successful commercial plant utilizing the pneumatic principle.
was made in the presence of Dr. L. D. Ricketts, Consulting Engineer,
ornton, Vice-President of the Inspiration company. According to F. B.
on plant of the Flinn-Towne flotation system was shipped to the Tezuit-
o. in Mexico in the spring of 1913. This could not be put into operation
the political conditions prevailing then. Otherwise, this would have
commercial pneumatic flotation plant. R. C. Canby informs me that
rried out by him suggested the idea to Messrs. Flinn and Towne of con-
tation machine to produce the desired fine dissemination of air by blow-
gh a porous medium.

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Flinn-Towne Installation

The Flinn-Towne machine, as mentioned, utilizes the pneumati principle also, but the application is somewhat different. An illustra tion of a single Flinn-Towne machine is given in Fig. 5, while Fig. shows the outline of the installation. The cells are constructed in th

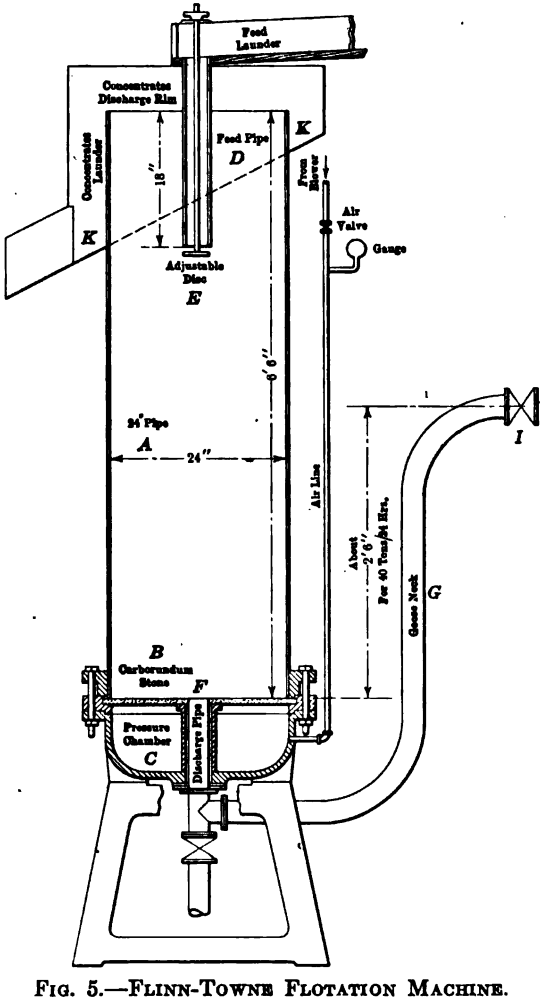

FIG. 5.—FLINN-TOWNE FLOTATION MACHINE.

shape of cylindrical tanks, the bottoms of which are formed by the poro medium. The feed enters near the top in the center of the cylind while the tailings leave the machine through a center hole in the poro medium. In the Flinn-Towne installation, only one roughing machi was provided, while additional cells served for the purpose of cleani

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both rougher tailings and rougher concentrates. The tailings produced by the concentrate cleaner cell were returned to the head of the roughing cell. In place of the canvas blanket of the Callow machine, Messrs. Flinn and Towne in the demonstration test at Inspiration used carborundum stones as the porous medium through which the air is injected into the pulp.

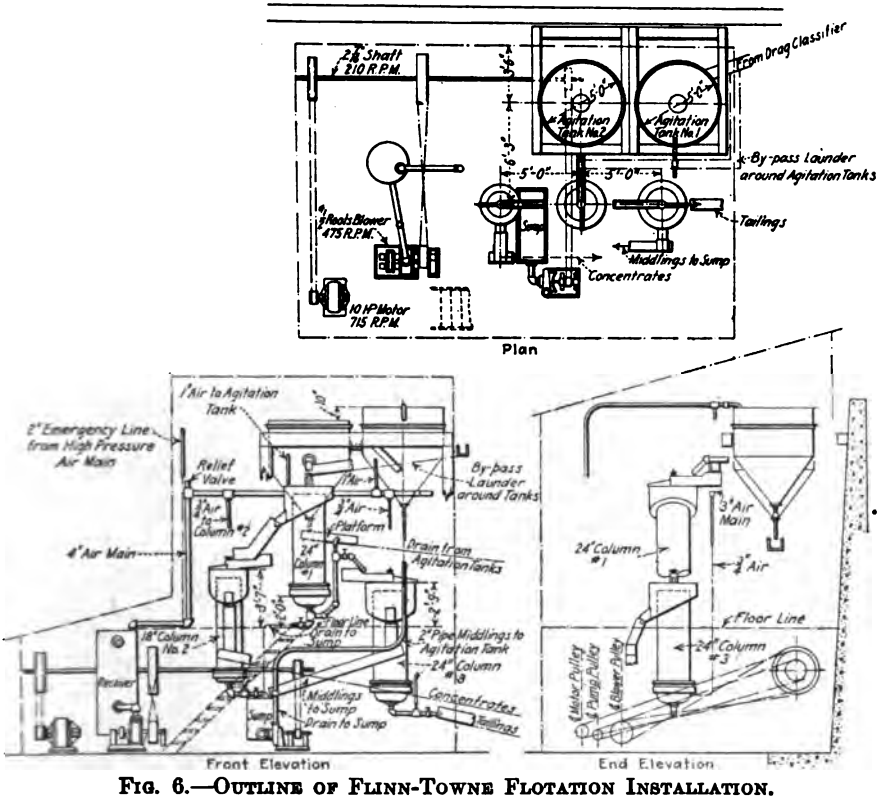

FIG. 6.—OUTLINE OF FLINN-TOWNE FLOTATION INSTALLATION.

Cole-Bergman Installation

The Flinn-Towne machine was withdrawn from the contest after a competitive test between this machine and others had run for several months, although the results obtained looked very encouraging. Its place was taken by a flotation machine designed by Messrs. Cole and Bergman.

Their machine is illustrated in Fig. 7. In principle the cells are similar to those of the Flinn-Towne machine. Evidently, the designers tried to improve on this system by mechanically developing the idea underlying the machine, and by changing one point, which in their opinion formed a weak part of the Flinn-Towne machine. This is the

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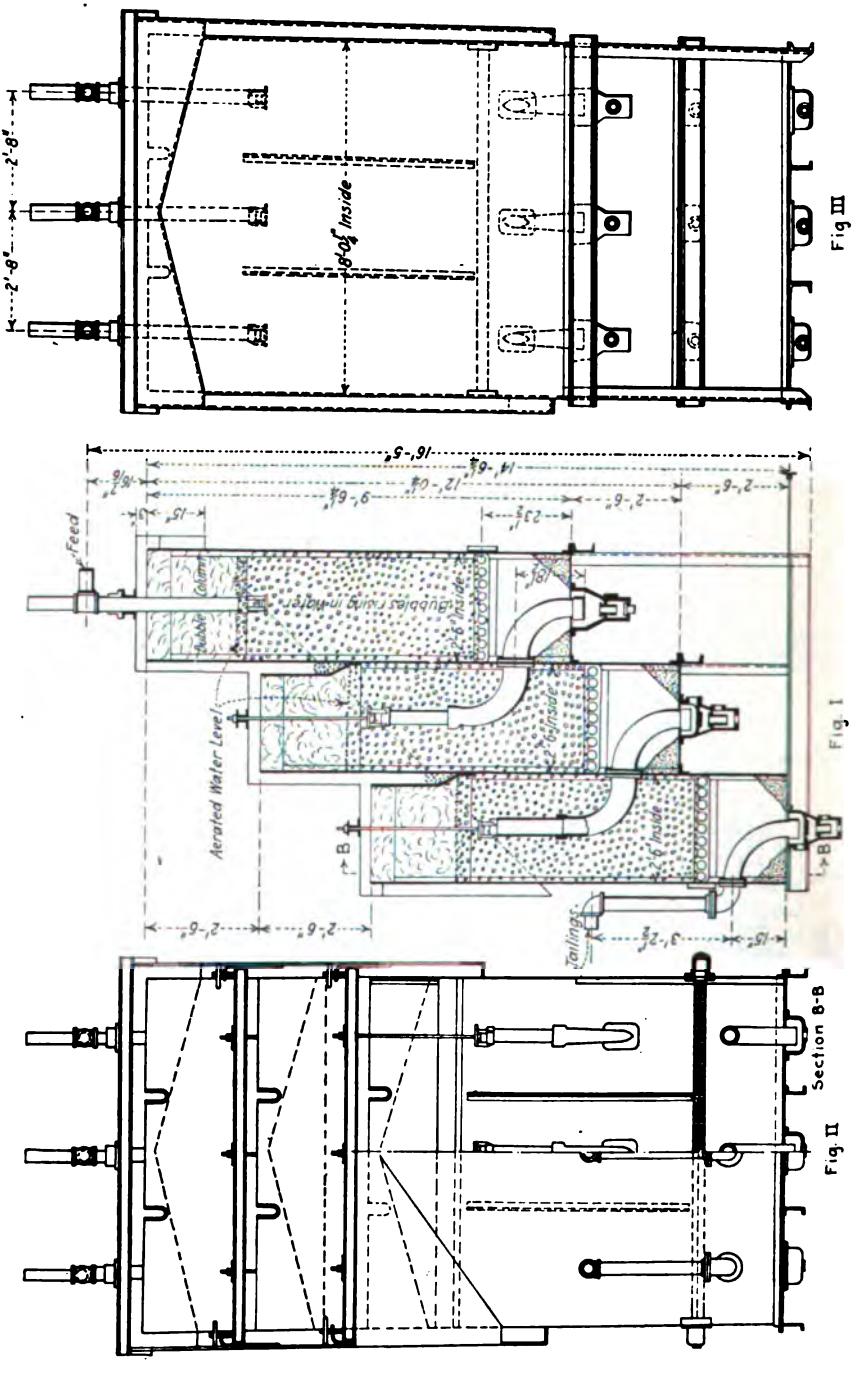

FIG. 7.—COLE-BERGMAN FROTHING CLASSIFIER.

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construction of the porous diaphragm, which, as explained below, was formed by a round carborundum disk. While carrying on the test of the Flinn-Towne machine, it proved necessary occasionally to wash the carborundum stones by injecting a water pipe from the top and even by removing the stones and cleaning them with water, acids, etc. Messrs. Cole and Bergman ascribed this deficiency to the fact that on account of the horizontal position of the porous medium, sand had a tendency to lodge thereon and to impede or prevent the passage of air through the pores. They substituted, therefore, a system of perforated tubes covered with a suitable fabric, such as canvas or flannel. This led to the construction of a porous medium in the form of a grate, as shown in the illustrations. Their idea seems to be a very happy one, as it makes it possible to apply the flotation process to ore pulps containing comparatively coarse particles of sand; for instance, it seems possible to treat with this machine ore mixtures that contain sand particles as coarse as 10-mesh and perhaps even coarser. Using this machine, the millman is now in a position to treat slime by flotation without the necessity of removing it from admixture with sand. The Cole-Bergman machine has given good results in our tests, as have the rest of the machines utilizing the pneumatic principle. It was operated in conjunction with a smaller machine of the same type installed for the purpose of cleaning the concentrates produced on the larger machine.

Methods Followed in Competitive Tests

In the interest of a fair contest between the flotation machines of the different types, each machine was provided, as nearly as possible, with the same character of pulp as the rest. In the beginning this was accomplished by providing each machine with one or more pebble mills to which feed was sent in such quantities that a mill product of practically the same fineness resulted in each case. When the results of the competitive tests began to approach each other very closely, the equalization of the pulp furnished to the different machines was still improved upon by mixing the ground product discharged from all the mills, and sending it to a divider which permitted the division into as many parts as there were competing machines, and in any proportion desired. As it had been decided in former experiments not to install tables ahead of the flotation process, no preliminary table treatment was used during the competitive tests, but each flotation system was furnished with a set of tables for the retreatment of the flotation tailings. It developed that this was essential in trying to arrive at a fair valuation of the machines, as some of the machines produced tailings from which additional mineral could be extracted by the table treatment more easily than from the tailings of other flotation machines, the cause evidently being that

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flotation machines of one type have a tendency to save more of the fine particles, while flotation machines of another type permit of a bette recovery of coarser grains. Those flotation machines that make a goo recovery on the sand, but leave a larger percentage of mineral in th slime, will not benefit from a subsequent table treatment as much as thos of the other kind.

Advantage of Pneumatic Flotation Machines

In the course of the tests at Inspiration, the fact became apparen that, by the application of the pneumatic principle, a higher recovery o the slime is made possible than without the utilization of injected ai although the Standard Minerals Separation machine also gives excellen results when the tonnage is reduced to a considerably lower figure tha the machine is supposed to be able to treat economically.

Therefore, in cases where high power consumption is of little impor tance, the Standard Minerals Separation machine will fill the requirement of a flotation machine remarkably well. When, however, the power con sumed has to be seriously considered, it seems advantageous in the ligh of the Inspiration experiments to make use of pneumatic flotation for th reason pointed out above, *i.e.*, that poor work of flotation machines o sands can be made up by a subsequent table treatment, while poor wor on slime cannot. Within certain limits, it is more important to insi on machines effecting a good slime recovery than on machines making good sand recovery.

Pneumatic Machines of Minerals Separation Co.

The Minerals Separation Co., realizing this, offered to put in the subaeration type of machine which added the advantages resulting fro the use of injected air to the advantages which their Standard machin seemed to have in saving coarser mineral. This resulted in the constru tion and testing successively of two of their subaeration type machine one in which the old spitzkastens still were retained, and another one which they were dispensed with. As seen from the illustration, Fig. provisions were made in the former machine to admit air to the pu conduits carrying the pulp from the spitzkastens to the agitating co partments next in succession, in order to make this air available to l mineral to the top of the spitzkastens where it can be skimmed off. T agitating compartments are closed at the top by covers set on an incli in such a way that the air rises to the top in the agitating compartmer and is conducted to the spitzkasten, being thereby utilized for the purpo of carrying the mineral values.

This construction of the Minerals Separation Co. was, however, o

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an intermediate step toward the design of the second-named machine, a machine of greater simplicity, which is similar in principle to the one

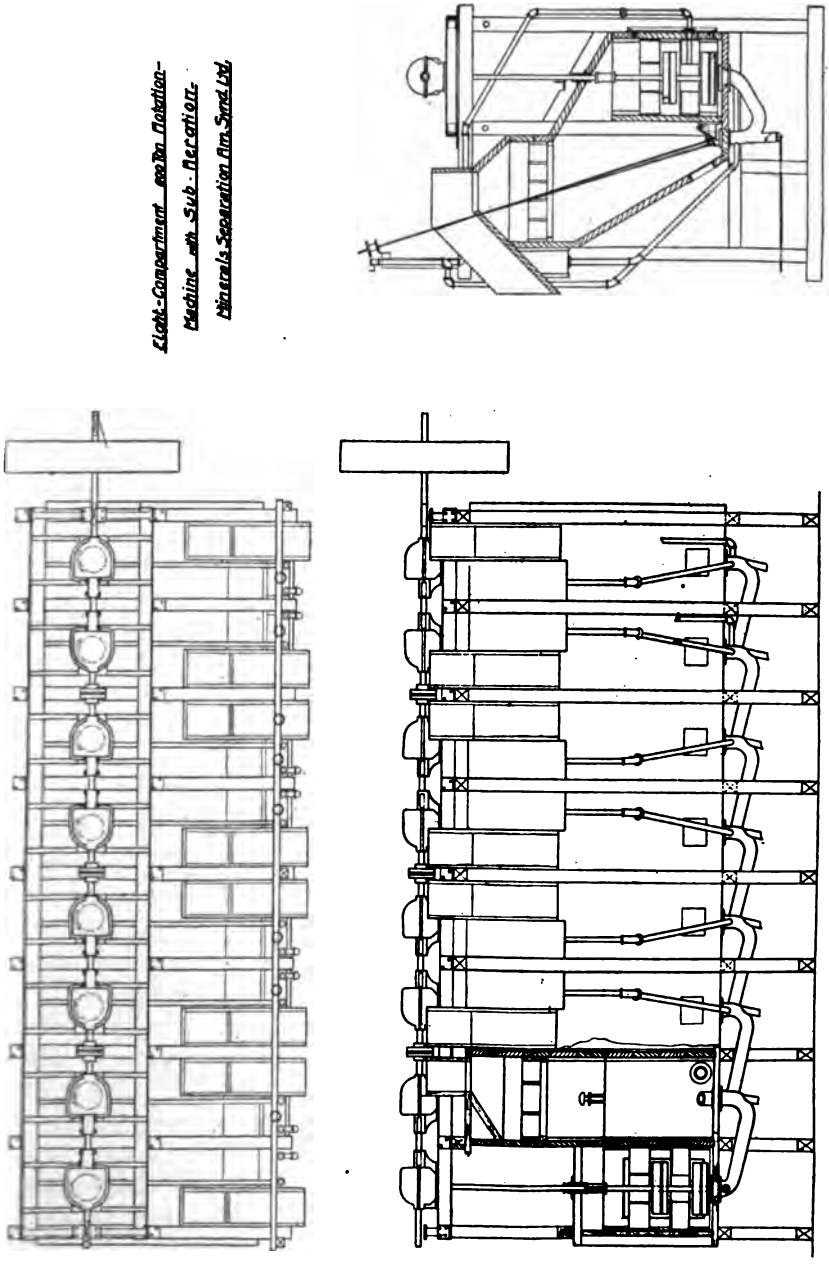

FIG. 8.—MINERALS SEPARATION SUB-AERATION TYPE OF FLOTATION MACHINE.

illustrated in Fig. 9. The machine consists of a long rectangular tank without any partitions, in which a number of agitators revolve, while air is

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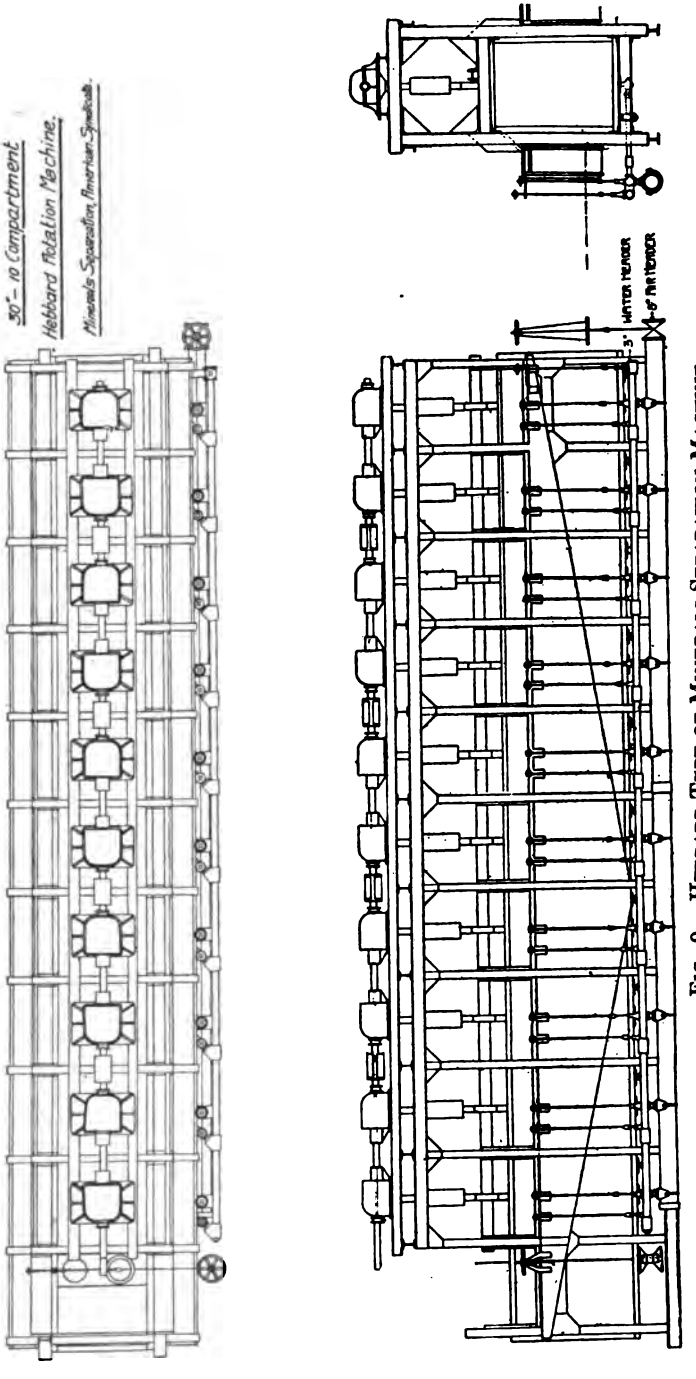

FIG. 9.—HEBBARD TYPE OF MINERALS SEPARATION MACHINE.

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injected underneath each impeller. We call the machine the "Hebbard Type" Minerals Separation Machine.

In order to limit the agitation to the lower part of the machine and to provide an area of comparative quietness in the upper part, a system of baffles is arranged above the agitators. As a consequence, the froth is given a chance to separate out in the upper portion of the rectangular tank, and flows off on both sides lengthwise into launders provided for the purpose.

This machine marks an important progress in the development of the Minerals Separation flotation systems toward increased recovery of copper, at least as far as the Inspiration ores are concerned. In our tests it effected a higher recovery of the fine mineral particles than the old machine had accomplished. The machine also has the advantage of greater simplicity and relatively low power consumption. The only drawback that we have found in the operation of the machine, extending over a number of months, is that the agitating shafts, which are suspended from above, have a tendency to bend and, if not straightened, soon cause the impeller blades to strike against the baffles, breaking one or the other.[5]

Float Skimming Device

While the competitive tests of the different flotation machines were being conducted, it was observed that large quantities of mineral froth frequently appeared in the tailings launders. The idea suggested itself to remove the froth, thus increasing the recovery obtained in the flotation machines and concentrating tables by an additional small amount. After some experimenting, a way was found in which this could be easily accomplished. The tailings launder was widened and deepened for some distance. The tailings stream was forced to pass underneath a baffle on entering the widened space, and was also forced to travel under another baffle board at the end of the space, thus being made to rise through the restricted area formed by the tailings end baffle and the back-wall of the float-saving device, which is illustrated in Fig. 10.

Contrary to what might be expected, tailings sand of the fineness of Inspiration tailings will not lodge in the widened-out space unless the length is greater than a certain critical distance, nor will it choke the upward channel in the tailings end of the machine. When conditions demanded the construction of the float-saving device in greater length, it was found that the accumulation of sand, which has a tendency to form in the middle of the machine, can be avoided by arranging an additional baffle not reaching quite to the bottom in the center of the machine. If

[5] This trouble is alleviated in the Hebbard machines turned out more recently by the Minerals Separation Co. by the provision of suitable bearings for the impeller shafts in the bottoms of the machines.

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the machine is built longer yet, more than one intermediate baffle necessary. The explanation for this behavior of the pulp, which at fi glance might seem paradoxical, is that, on account of reduced passagew between the lower part of the baffles and the bottom of the machine, pulp is forced to travel through at speed so high that it will not per the settling out of sand. As a consequence, a hydrostatic head establish itself between front and back of such a baffle and has to be tal care of in the design. It is found, however, in actual practice, that

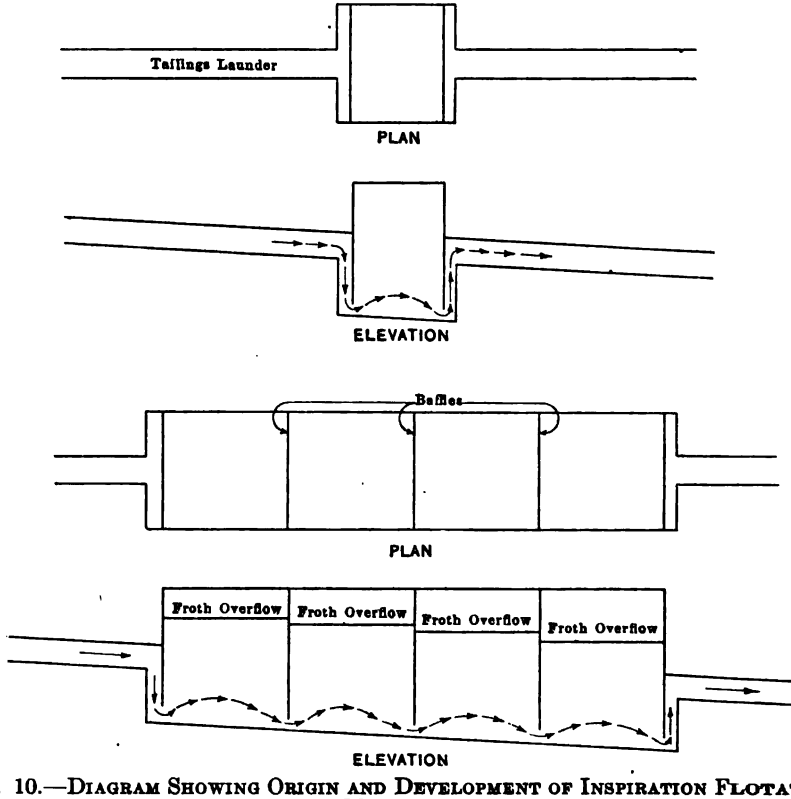

FIG. 10.—DIAGRAM SHOWING ORIGIN AND DEVELOPMENT OF INSPIRATION FLOTA MACHINE.

hydrostatic head required to keep the sand from settling out by crea an accelerated current underneath the baffle is relatively small.

A similar consideration explains the fact that the upward tail passage at the end of the machine has very little, if any, tendenc choke; to take care, however, of possible choke-ups that might be ca by the discharge of coarse rocks, pieces of wood, etc., into the pulp, it found advisable to arrange some air jets in the bottom of the upv tailings passage to assist in clearing it whenever necessary.

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Inspiration Flotation Machine

This device was in actual operation for several months at our test mill and made itself pay well by adding about 1 per cent. to the recovery

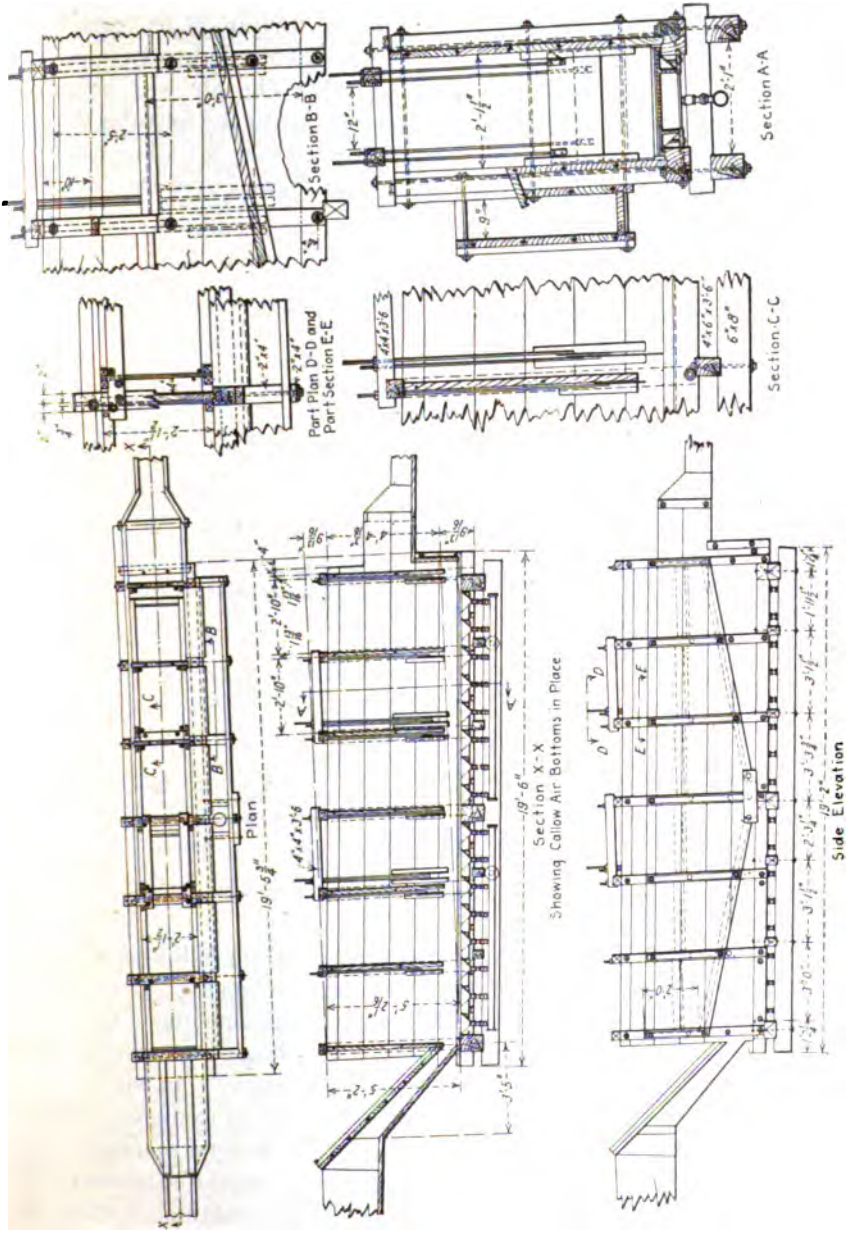

FIG. 11.—EXPERIMENTAL INSPIRATION FLOTATION MACHINE.

actually obtained. The question then presented itself as to whether this device, which was successful in saving mineral escaping from the flotation

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machines could not be transformed into a flotation machine itself by arranging for the injection of finely distributed air into the pulp passing through the machine. The development of the idea led to what we call our Experimental Inspiration Flotation Machine, which was built on a small scale, that is of a size to treat up to 100 tons in 24 hr., and was placed in competition with the other flotation machines already in existence. This machine is illustrated in Fig. 11. The air was injected through a porous bottom arranged in a way similar to the bottoms of the Callow and Flinn-Towne machines (as a matter of fact, a Callow bottom was used in the first tests) and the concentrate flowed over the edge on

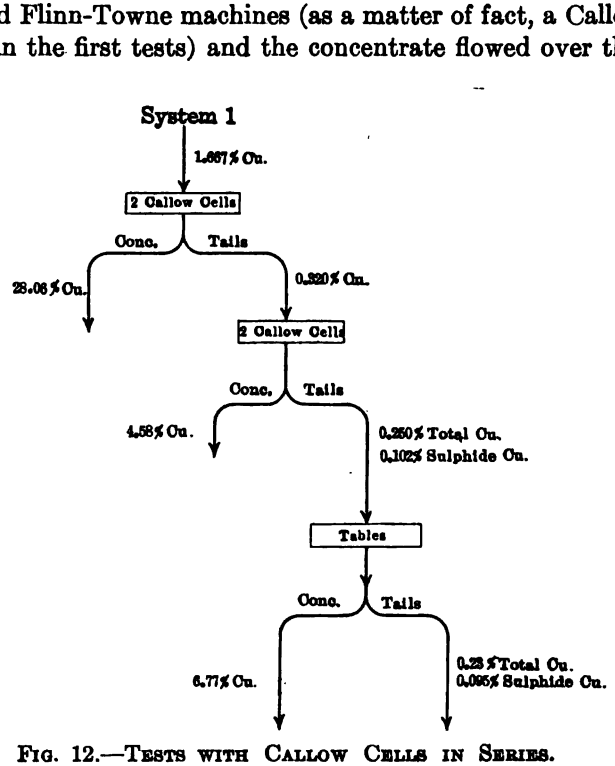

FIG. 12.—TESTS WITH CALLOW CELLS IN SERIES.

one side of the flotation tank. The mode of operation followed was al similar to the one developed for the other air machines. A low-gra concentrate was produced and retreated on a similar machine of the sa construction, called the cleaner machine, the tailings from which w returned to the head of the roughing flotation machine. As will be se from the drawing, the construction of this machine is extremely sim and comes pretty near to an ideal of Mr. Mills, our general manager, w prophesied that the flotation machine of the future would be nothing l a launder with provisions for injection of air. The method of effect the concentration in two stages was also followed when the new type the Minerals Separation machines were put into commission.

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Arrangement of Callow Cells in Series

In principle, the Inspiration machine shows one difference from the Callow machine; viz., that instead of splitting the pulp between a great number of machines, it is forced to travel through a series of compartments in succession. This is the same policy that had also been followed in the construction of the Minerals Separation machines. The question then came up, whether it might not be of advantage to arrange the Callow cells also in series. To test out the idea, the flow sheet of the Callow plant was changed (see Fig. 12). The primary feed was sent to two of the cells, the other cells receiving the tailings from these primary cells.

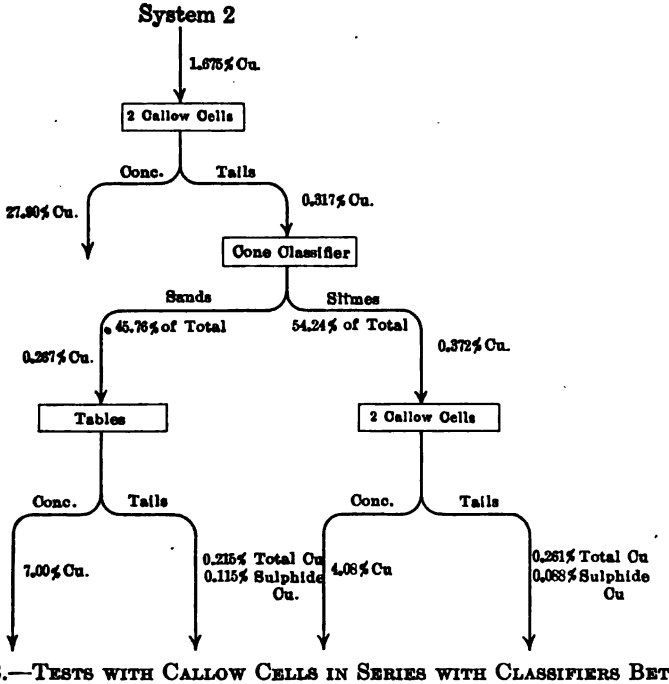

FIG. 13.—TESTS WITH CALLOW CELLS IN SERIES WITH CLASSIFIERS BETWEEN.

This arrangement showed at least no disadvantage over the multiple treatment; it seemed superior to the other system in the one respect that it showed at a glance whether the cells were operated well, because in that case most of the concentrate would be produced on the primary cells while the secondary cells would produce a rather light-colored froth and would add only a small fraction to the recovery made on the primary cells. Both primary and secondary concentrates were treated jointly in a cleaner cell, while the tailings from the secondary cells were conveyed to a number of concentrating tables for retreatment.

A variation of the latter flow sheet was also tested (compare Fig. 13).

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Between the primary and secondary cells, a cone classifier was interpos
for the purpose of making a separation between the sand and slime. T
slime was sent to the secondary flotation machines, while the sand w
passed to tables. This latter flow sheet was finally decided upon for t
Callow installation in the large concentrator. Tests made with the obje
of establishing which of the two is superior, resulted in a tie. Some of t
figures on this point are given in Table 2.

TABLE 2.—*Retreatment of Tailings from Two Callow Cells. Comparison
Results Obtained from Two Different Methods as Shown in Figs.* 12 *and* 1
Screen Analysis of Feed and Tails of Secondary Callow Cells

Mesh	System 1									System 2							
	Feed				Tails					Feed				Tails			
	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Sol. Cu	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Weight		Per Cent. Cu	C Co te
	Cum.	Indiv.			Cum.	Indiv.				Cum.	Indiv.			Cum.	Indiv.		
65	5.0	5.0	0.29	0.014	5.0	5.0	0.21	0.011	0.05								
100	19.0	14.0	0.25	0.035	19.2	14.2	0.15	0.021	0.06	3.8	3.8	0.20	0.007	4.0	4.0	0.17	0.0
150	32.2	13.2	0.29	0.039	32.4	13.2	0.16	0.021	0.10	10.6	6.8	0.25	0.017	10.8	6.8	0.14	0.0
200	42.4	10.2	0.33	0.034	42.8	10.4	0.21	0.022	0.11	18.6	8.0	0.30	0.024	19.0	8.2	0.25	0.0
	100.0	57.6	0.35	0.201	100.0	57.2	0.27	0.155	0.20	100.0	81.4	0.40	0.326	100.0	81.0	0.30	0.2
Assay calc.....			0.32	0.323	.....	.....	0.23	0.230	0.15	.....	.....	0.374	..?..	.....	.....	0.28	0.2
Assay direct....			0.32	.....	.....	.....	0.23	.....	0.15	.....	.....	0.370	.....	.....	.....	0.26	...

NOTE.—The concentrates from the flotation cells were in both cases retreated
half-size Callow cells, the retreatment tailings being returned to the head of the p
mary cells.

	Ton per Primary Cell
Tonnage rate in system 1.....	49.3
Tonnage rate in system 2.....	47.5

CONCLUSIONS.—No advantage is apparent in favor of either system. System
be installed should be decided from mechanical points.

Variations of Flow Sheet

In the way of other flow sheet variations, etc., only one thing was tri
to any extent; that is, whether it is wise to send the cleaner tailings ba
to the head of the rougher machine for retreatment. In the limited ti
at our disposal, it proved impossible to substitute something better, b
the chances are that the same holds true of flotation that is true for wa
concentration; viz., that middlings are snnt back for retreatment mai
because the designer does not know anything better to do with the
In principle it seems wrong to follow this practice, because, when hea
copper losses occur in concentrating machines which express themsel
in high tailings, there is always a doubt as to whether this is due to p

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recovery made in the primary treatment or to heavy losses incurred by middlings sent back for retreatment.

Coal Tar as Flotation Agent

It was to be expected, when the flotation process was installed in our test plant, that there would be ups and downs in the recovery because the process was rather new, especially in its application to chalcocite ores. For the month of June, 1914, the recovery obtained in the test mill showed a sudden drop, and the serious problem confronted us of establishing the cause and finding a remedy. Some of the flotation experts suggested that it might be due to the fact that a new shipment of cresylic acid which had been received and used at that time might not fill the specifications of being 98 per cent. pure. We did not feel competent to say whether the impurities actually amounted to more than 2 per cent. We were, however, inclined to think that perhaps cresylic acid, which is, as is well known, one of the products resulting from fractional distillation of coal tar, might not represent the fraction most suitable for the flotation of our ores. Having no coal tar available in the neighborhood, we proceeded to make some by distilling a sample of ordinary New Mexico soft coal and separating the tar thus formed into the fractions distilling off at different temperatures.

Our facilities for testing oils were limited at that time. The Minerals Separation Co.'s representative did not believe in small-scale tests, and for this reason did not recommend experiments with small testing machines. Nevertheless, it seemed desirable to have something with which to carry out small-scale laboratory experiments. Dr. Ricketts, who was aware of our troubles and realized the importance of such tests, was kind enough to send us a little electrically operated emulsifying machine, which served admirably for qualitative tests. We also built a testing machine based on the principle of the Standard Minerals Separation machine, with the difference, however, that instead of sending the pulp from one agitating compartment to a spitzkasten and then into another agitating compartment and spitzkasten, we made the pulp return from the first spitzkasten to the original agitator, forcing it to revolve in a closed circuit. Our laboratory machine is illustrated in Fig. 14. Lately, a machine based on the same principle has been put on the market and is sold by the Denver Fire-Clay Co. Thus, we had a chance to try the different fractions of our home-made coal tar.

The chemist who conducted these tests (Mueller) hit on the idea that it might be well, in addition to trying the different fractions, also to test the coal tar as a whole. The results were very surprising, since they showed that by the addition of crude coal tar we could effect a greater recovery than we were able to obtain by the use of highly refined cresylic

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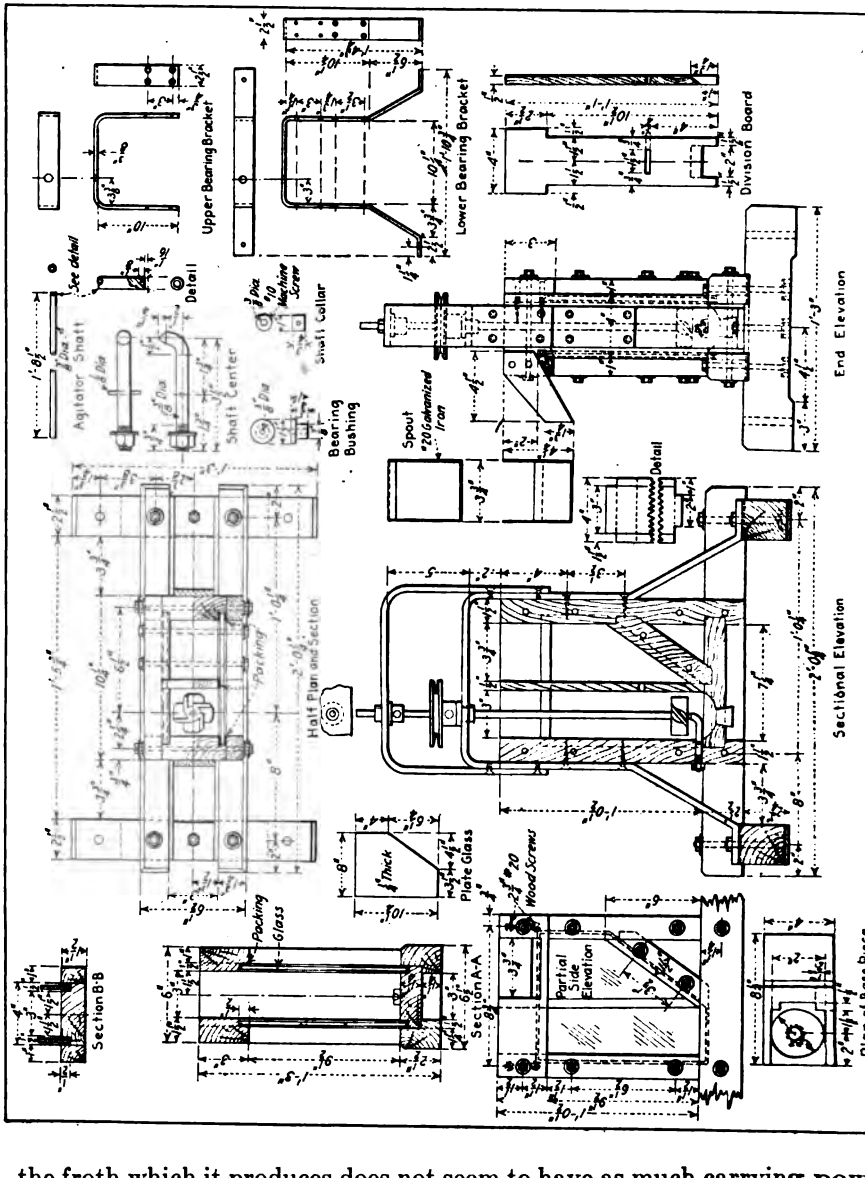
acid. From this point dates our experience that it is better to use coa
tar than soluble flotation agents like cresylic acid to save coarse mineral
Cresylic acid is an extraordinarily good agent for producing froth, bu



the froth which it produces does not seem to have as much carrying pow
for coarse mineral as that produced by coal tar. Not all coal tars a
equally good for this purpose. Tests in laboratory machines easily sho
the difference between coal tars of different origin.

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The system that we used to carry out such tests has been described, although without authorization, by William A. Mueller, one of my former assistants who took part in these laboratory tests.[6]

It is difficult to utilize coal tar in plants using flotation supplementary to gravity concentration, on account of the fact that it is not easy to effect a good amalgamation of tar with the pulp in agitating tanks, and even in mechanical flotation machines. The use of coal tar lends itself very well indeed to the system of feeding tar into the grinding machines, a system that, as mentioned above, had been worked out in our small test mill and patented by Mr. Chapman.

The company is indebted to Mr. Callow for proving the merits of coal-tar creosote as a flotation agent by using it in his demonstration plant at Inspiration. After we had established the value of coal tar by laboratory tests, and while efforts were being made to obtain it commercially, he applied creosote successfully. We have continued to use it for a long time, mostly in combination with coal tar, and have only recently dropped it, as we find crude coal tar cheaper and better.

Experience with Primary Slime

After the first difficulty that we encountered in our large-scale tests had been solved by the introduction of coal tar into the flotation-oil mixture so far used, things went along fairly well for some time until a new difficulty was encountered. It happened that once in a while an abundant froth was produced on the flotation machines, but this froth seemed to have very little carrying power for the mineral contents of the ore and held mainly finely divided gangue. It was observed that this phenomenon occurred with special severity whenever the ores shipped to the test mill contained a large amount of fines and a small amount of coarse rock. It was attributed, therefore, to the presence of what may be called primary slime, *i.e.*, slime not formed by the crushing of ore in the mill, but originating from the mine. That the falling down of the flotation machines was caused by variation of the ores was proven by the fact that samples of the refractory ores when treated in the laboratory testing machine, gave as unsatisfactory results as the corresponding ore did in the big flotation machines. To demonstrate that the presence of the original slime caused the trouble, samples of the mill feed were separated by screening them on a 200-mesh screen. The oversize, when reduced to the proper fineness for flotation treatment, did not offer the least trouble and yielded a high-grade concentrate in the laboratory machines, while the undersize proved extremely refractory (refer to Table 3). The same experiment was repeated on a large scale,

[6] William A. Mueller: Use of Coal Tar in Flotation, *Engineering and Mining Journal*, vol. 100, No. 15, p. 591 (Oct. 9, 1915).

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the products from a Marcy ball mill crushed to about 6-mesh being treated on a drag classifier. The oversize, after being reduced in pebble mills to the necessary fineness for flotation treatment, was sent to one group of flotation machines, while the overflow from the drag classifier was sent to another group. Fig. 15 illustrates the results of this experiment. While the reground oversize yielded remarkable lean tailings

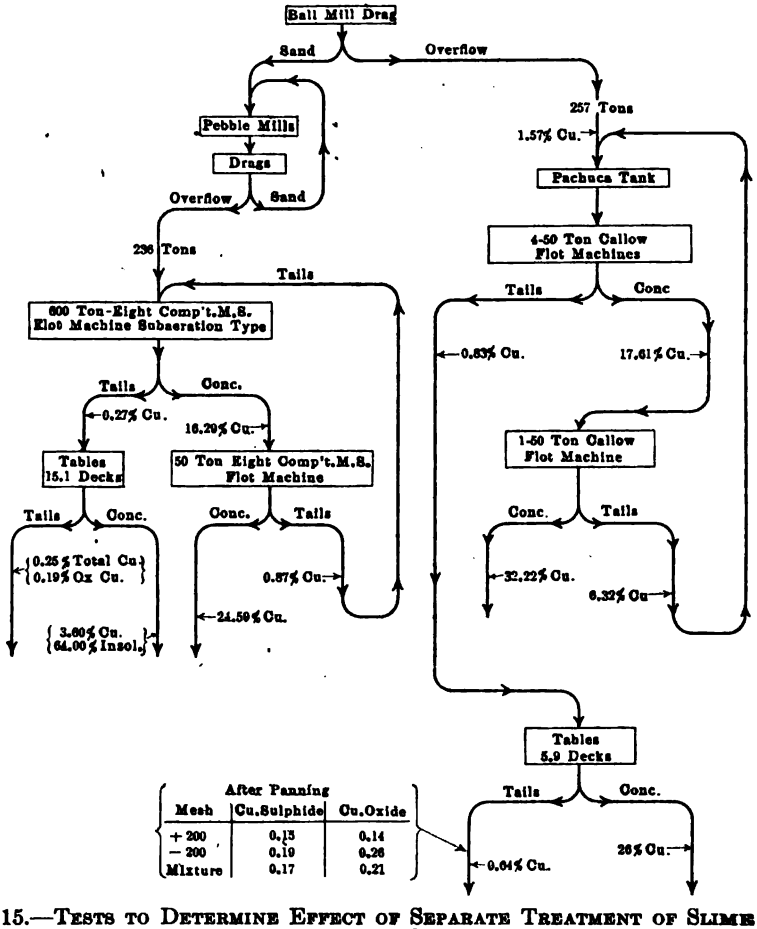

FIG. 15.—TESTS TO DETERMINE EFFECT OF SEPARATE TREATMENT OF SLIMES AND REGROUND SAND.

the product containing the primary slime could not be treated advantageously by flotation. We were almost on the point of concluding that in order to get the best results, a separation of the primary slime should be made in our large mill and flotation should not be entirely relied upon for the treatment of the slime. Before finally deciding on this point however, some additional laboratory experiments were made to study the influence of the primary slime on flotation. These tests established

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TABLE 3.—*Screen Analysis of Ore Drawn from Mill Bins and Flotation Tests of Different Screen Sizes*

Screen Size	Weights, Per Cent.		Per Cent. Total Copper	Copper Contents	Per Cent. Oxidised Copper	Contents Ox. Cu	Flotation Results		Recovery Per Cent.
	Cumul.	Indiv.					Tails	Midds.+ Conc.	
Inches									
+ 1	30.8	30.8	1.05	0.324	0.13	0.40	0.22	12.8	80.5
+ ½	47.9	17.1	1.34	0.329	0.11	0.19	0.22	17.4	84.7
Mesh									
+ 4	70.9	23.0	1.44	0.332	0.12	0.28	0.17	17.1	89.0
+ 8	75.2	4.3	1.42	0.061	0.12	0.05	0.17	16.3	88.8
+ 14	80.3	5.1	1.77	0.068	0.13	0.07	0.26	16.8	86.7
+ 28	84.4	4.1	2.14	0.080	0.14	0.06	0.61	16.3	74.3
+ 48	87.5	3.1	2.62	0.081	0.19	0.06	0.72	11.3	77.5
+ 100	89.9	2.4	3.29	0.079	0.24	0.06	0.63	22.5	83.1
+ 200	91.9	2.0	3.16	0.063	0.27	0.05	0.60	20.1	83.5
− 200	100.0	8.1	1.53	0.124	0.43	0.35	1.57	1.48	None
	.....	100.0	1.44	1.441	0.16	0.157			

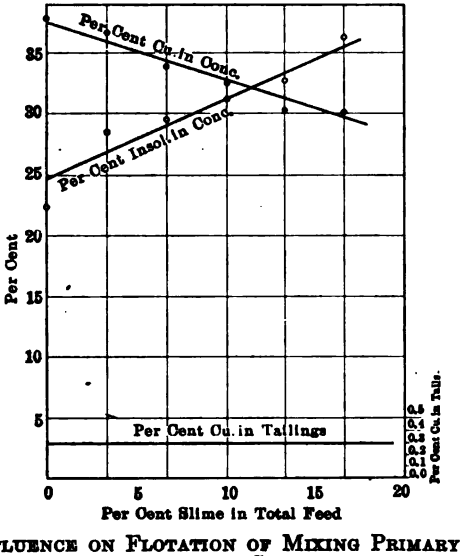

FIG. 16.—INFLUENCE ON FLOTATION OF MIXING PRIMARY SLIME AND REGROUND SAND.

The + 1-in. and + ½-in sizes resulting from the screen analysis were ground to 10 mesh, mixed with varying amounts of − 200 material and the mixtures ground with the usual amounts of oil in the laboratory ball mills. After this preliminary treatment, the samples were treated in the laboratory flotation machines.

one fact quite well, which is, that when refractory slime is mixed with a sufficient quantity of coarse ore ground to the necessary fineness, in other words, when the percentage of primary slime in the flotation feed is kept

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low, the slime loses its refractory character. I do not mean to say tha by reducing the quantity of primary slime, the deleterious influence i reduced in proportion so that it cannot be detected as easily, but conclud from our tests that when the flotation feed contains a sufficient percentag of comminuted coarse rock, the primary slime contained in it can b treated more advantageously than it could by itself. Some tests bearin on this point are illustrated in Fig. 16. They indicate that as high a amount as 20 per cent. primary slime may be mixed with ground coars ore without causing increased tailing losses.

The concentrates, according to this set of tests, seem to have a tend ency, however, to carry more insoluble matter with an increasing per centage of slime.

Influence of Iron on Flotation

While these tests were in progress, we made another accidental dis covery which proved very helpful to us. In our tests on the most eco nomical way of reducing the ore to the fineness necessary for flotatio we had, among other machines, a ball mill in competition with pebbl mills. In the ball mill, steel balls performed the duty that in pebbl mills was done by flint pebbles.

For a while, the ball-mill discharge was treated on one flotatio machine, while the pebble-mill discharge was treated on a group of others While this flow sheet was being followed, we thought we noted that flotation machine treating the ball-mill product showed the influence o the primary slime to a lesser extent than the flotation machine treatin the pebble-mill product. In a discussion with Dr. Ricketts and Mr Mills, the question was raised as to whether the iron introduced in th mill pulp by the attrition of the balls might not have something to d with the fact. The question was accordingly made the subject of som laboratory experiments. The results of a series of such experiments ar represented in Table 4, and proved conclusively that the iron had beneficial influence on flotation in counteracting the harmful effect o the primary slime. This discovery was one of the inducements for in stalling ball mills in the big concentrator, while originally pebble mill had been considered for this purpose.

We have not yet reached a point where we can safely give the reaso for the action of the iron introduced into the flotation pulp. It is sure from the experiments referred to, that the same results, as by grindin with balls, could be obtained by introducing the iron in finely divide form, say in the form of filings, into a pebble-mill pulp. We suppose for a while, and I am not yet certain that this supposition is incorrect that the metallic iron might react on the impurities contained in solutio in the mill water and introduced therein with the primary slime. W find, as a matter of fact, that our ores contain very little in the nature

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TABLE 4.—*Effects of Iron and Other Solids on the Flotation of Refractory Copper Ores*

Test No.	Grams Ore	Per Cent. Copper	Grams Copper	Concentrates			Recovery Per Cent.	Remarks
				Grams	Per Cent. Cu	Grams Cu		
F21	750	2.01	15.07	45	23.66	10.65	70.7	Added 10 g. iron filings.
F22	750	2.01	15.07	47	29.52	11.52	76.4	Added 10 g. iron filings.
F27	750	2.01	15.07	43	27.10	11.65	77.5	Added 2 g. iron filings.
F28	750	2.01	15.07	47	26.90	12.64	84.0	Added 2 g. iron filings.
F45	750	2.01	15.07	51	23.84	12.16	80.7	Added 10 g. iron filings.
F46	750	2.01	15.07	48	25.60	12.29	81.5	Added 10 g. iron filings.
F47	750	2.01	15.07	27	8.80	2.38	15.8	Blank with no solids added.
F48	750	2.01	15.07	28	8.34	2.34	15.5	Blank with no solids added.
F49	750	2.01	15.07	62	20.14	12.49	83.0	Added 10 g. miscellaneous iron filings from shops.
F50	750	2.01	15.07	63	20.10	12.66	84.0	Added 10 g. miscellaneous iron filings from shops.
F51	750	2.01	15.07	60	18.82	11.29	75.0	Same as F49 and F50 by different observer.
F52	750	2.01	15.07	62	19.54	12.11	80.5	Same as F49 and F50 by different observer.
F53	750	2.01	15.07	29	6.16	1.79	11.9	Blank with no solids added.
F54	750	2.01	15.07	30	7.96	2.39	15.9	Blank with no solids added.
F55	750	2.01	15.07	59	21.08	12.44	82.7	Added 10 g. iron filings.
F56	750	2.01	15.07	56	25.80	14.45	96.0	Added 10 g. iron filings.
F64	750	2.01	15.07	65	19.52	12.69	81.9	Ground in mill with steel balls instead of pebbles.
L27	750	1.10	12.75	37	28.42	10.51	81.7	Blank on good flotation ore.
L28	750	1.10	12.75	34	28.46	9.85	78.3	Blank on good flotation ore.
L29	750	1.10	12.75	76	5.66	4.60	33.4	Identical conditions as L27 and L28 but added 10 g. zinc filings.
L30	750	1.10	12.75	83	5.06	4.20	32.9	Identical conditions as L27 and L28 but added 10 g. zinc filings.

soluble salts, and that whatever they do contain is mainly confined to the primary slime. For this reason, in laboratory tests we have tried repeatedly to substitute pure water for the water contained in the mill pulp. In every case, we have noted some improvements in the results obtained. We have also found that when we separate the water from the refractory pulp, treat it with iron filings, and add it to the original pulp again, we get a certain improvement in the recovery, but we have not been able to get an improvement equally as good as that obtained by direct introduction of finely divided iron into the pulp. For this reason, we have often thought that the effect of iron is physical rather than chemical in character. The iron exists in the pulp, at least partly, in the metallic form, as can be proven by the use of the magnet. If necessary, the effect of the iron could be increased by removing the iron contained in the tailings pulp by means of electromagnets and returning it to the mills or the flotation machines. Although we have considered the possibility of applying this fact commercially, we have not yet done so.

I have been told by several people that they have tested the influence

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of finely divided iron on their ores and have not obtained any improvement whatever. This shows, evidently, that metallic iron is not a universal remedy for all flotation troubles, but as far as our primary slime is concerned, our experience leaves no doubt about its usefulness, and I believe the figures quoted above are positive enough to bear out my statement. As a curiosity I have included in Table 4 some figures showing the influence of the addition of zinc filings to ore that without the addition does not offer any difficulty in the flotation treatment.

Influence of Chemicals on Primary Slime

We also found that the harmful influence of primary slime can be counteracted by the introduction of certain chemicals; for instance, we

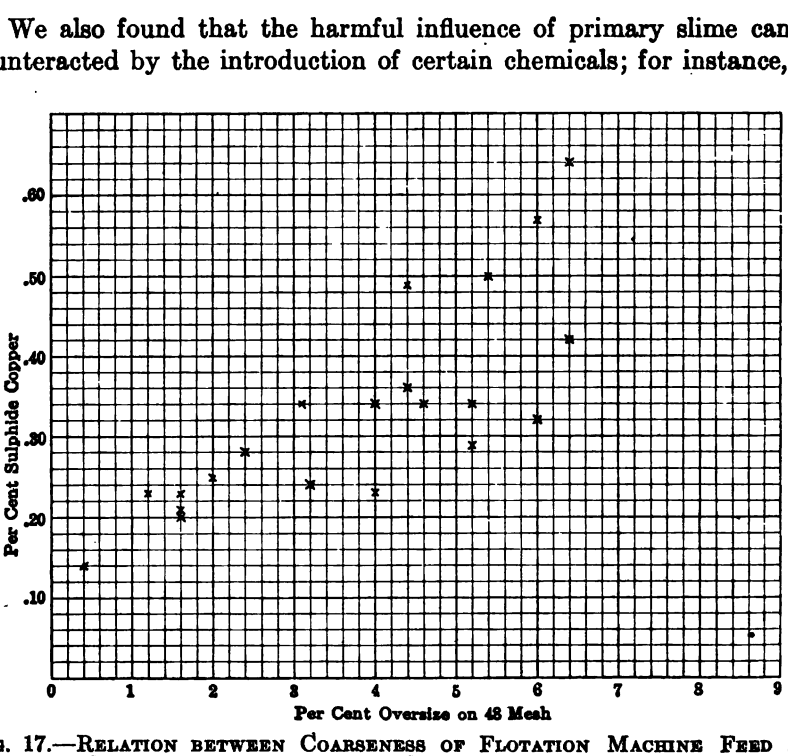

FIG. 17.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW FLOTATION TAILING.

have used advantageously sodium hydroxide or potassium cyanide. The use of sodium hydroxide was recommended to us by the Minerals Separation Co., which, I understand, applied it first in the plant of the Caucasus Copper Co.

Fineness of Crushing Desired

Tests to determine the most advantageous fineness for treatment of the ore according to our flow sheet, *i.e.*, flotation machines followed by

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tables, naturally formed an essential part of our tests. Results of this character are plotted in Figs. 17 and 18 and seem to indicate that to get the best results grinding should be carried to such a point that not more than about 1 or 2 per cent. of the pulp will remain on a 48-mesh screen (Tyler type). The grinding at our large concentrator is at present carried closely to this point.

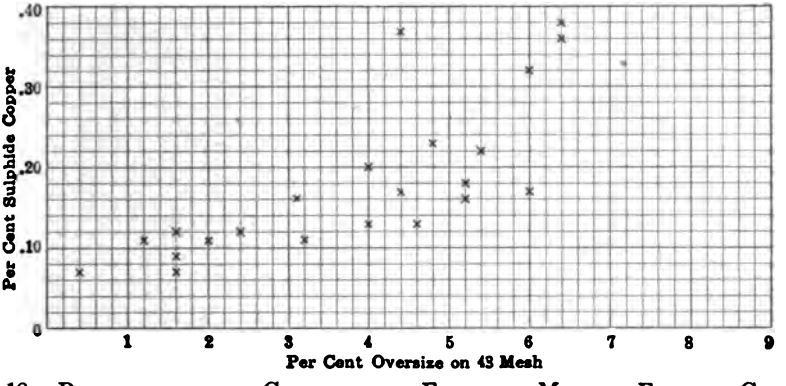

FIG. 18.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW TABLE TAILING.

OPERATION OF LARGE CONCENTRATOR

Flow Sheet of Callow Sections

The first four sections of the Commercial Concentrator of the Inspiration company were equipped with Callow flotation machines. The flow sheet followed in these sections is illustrated in Fig. 19. As will be seen from this illustration, the ore, after having been reduced to the desired fineness, and having been oiled in the Marcy ball mills, is sent to 12 (in other sections 8) Callow cells. The tailings from these primary cells are sent to a drag classifier designed by Mr. Burch, which effects a separation of sand and slime. The slime is retreated in 12 (in other sections 16) additional Callow cells, while the sand is sent to a hydraulic classifier and then to tables. The concentrates made both on the primary and the secondary Callow cells are retreated in a group of four Callow cleaning cells from which the tailings are returned to the head of the primary Callow cells. On the tables of the Deister Machine Co. a separation is made between concentrates and tailings, but no middlings are produced and no part of them is reground. This, however, is probably only a temporary arrangement, the idea being that a suitable regrinding and reconcentration process will be installed later, after having been worked out by extensive tests.

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Flow Sheet of Inspiration Sections

Thirteen sections of the Inspiration concentrator are equipped with Inspiration flotation machines. The flow sheet of these sections is represented in Fig. 20. The ore, after having been crushed and oiled passes to the head of the flotation machines and traverses their whole

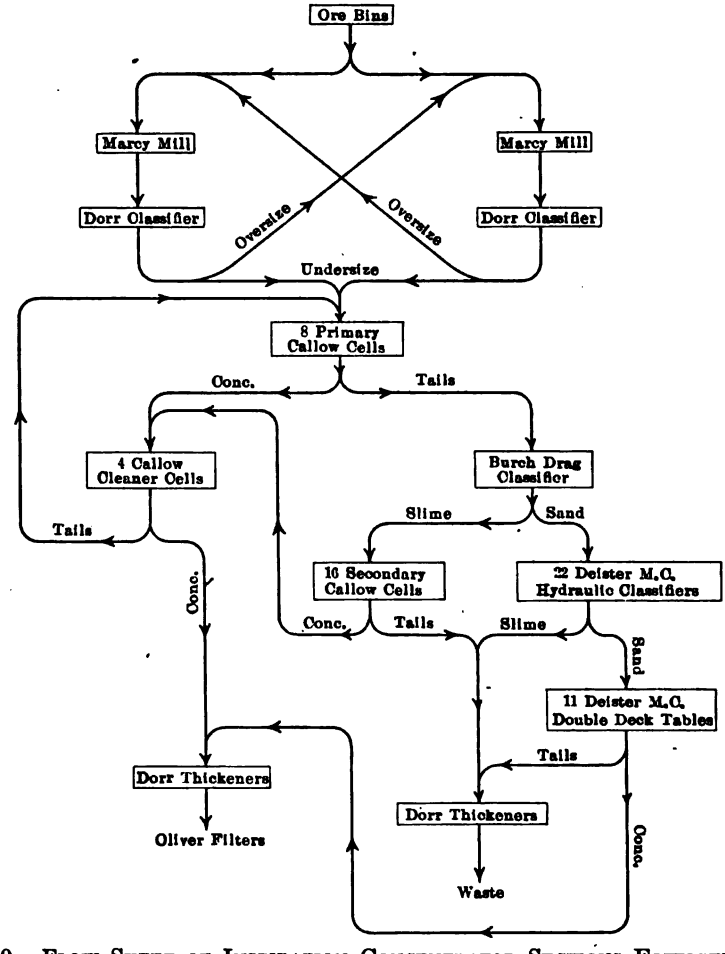

FIG. 19.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WIT CALLOW FLOTATION MACHINES.

length. The resulting tailings are split, by the same kind of a drag a mentioned before, into a sand and a slime product. The slime produc is run to waste. This is thought permissible because, after having passe through the whole length of the flotation machine, the slime has bee impoverished to such an extent that retreatment seems unnecessary

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The sand product undergoes the same retreatment as that described for Callow sections.

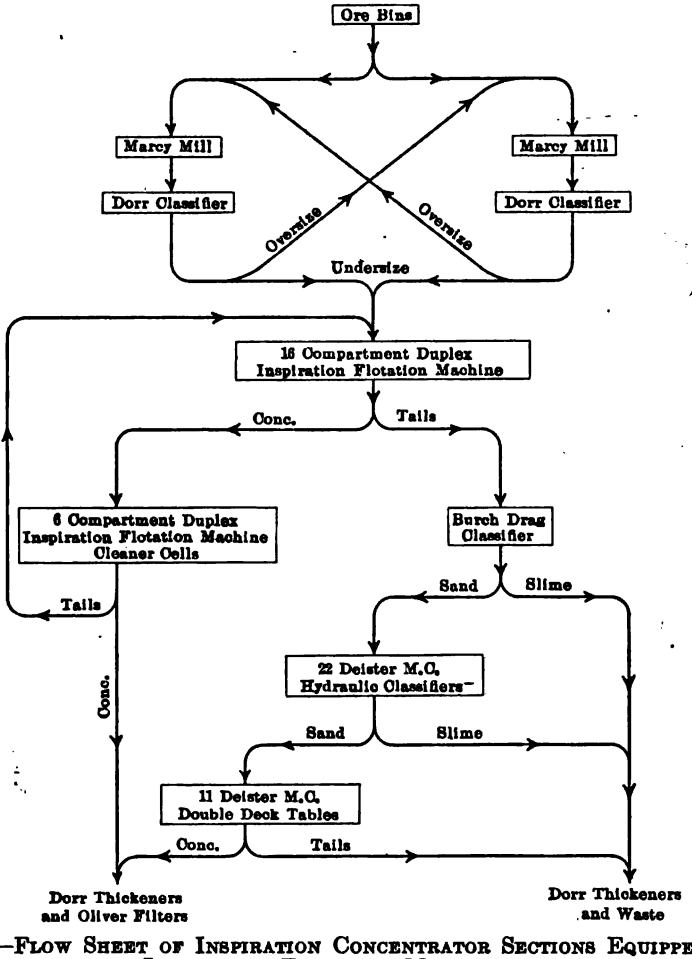

FIG. 20.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WITH INSPIRATION FLOTATION MACHINES.

Inspiration Flotation Machine of Steel Construction

Fig. 21 shows the drawing of an Inspiration flotation machine of steel construction as designed by Mr. Burch's office. At the end of each section of eight compartments, a pulp overflow is provided which regulates the level of the pulp in the preceding compartments of the flotation machine. The subdivision into two sections of eight is made in the interest of closer regulation of the pulp levels. It had been feared that if no such subdivisions were made, and the regulation of the levels accomplished at the tail end only, appreciable fluctuations might take place in the com-

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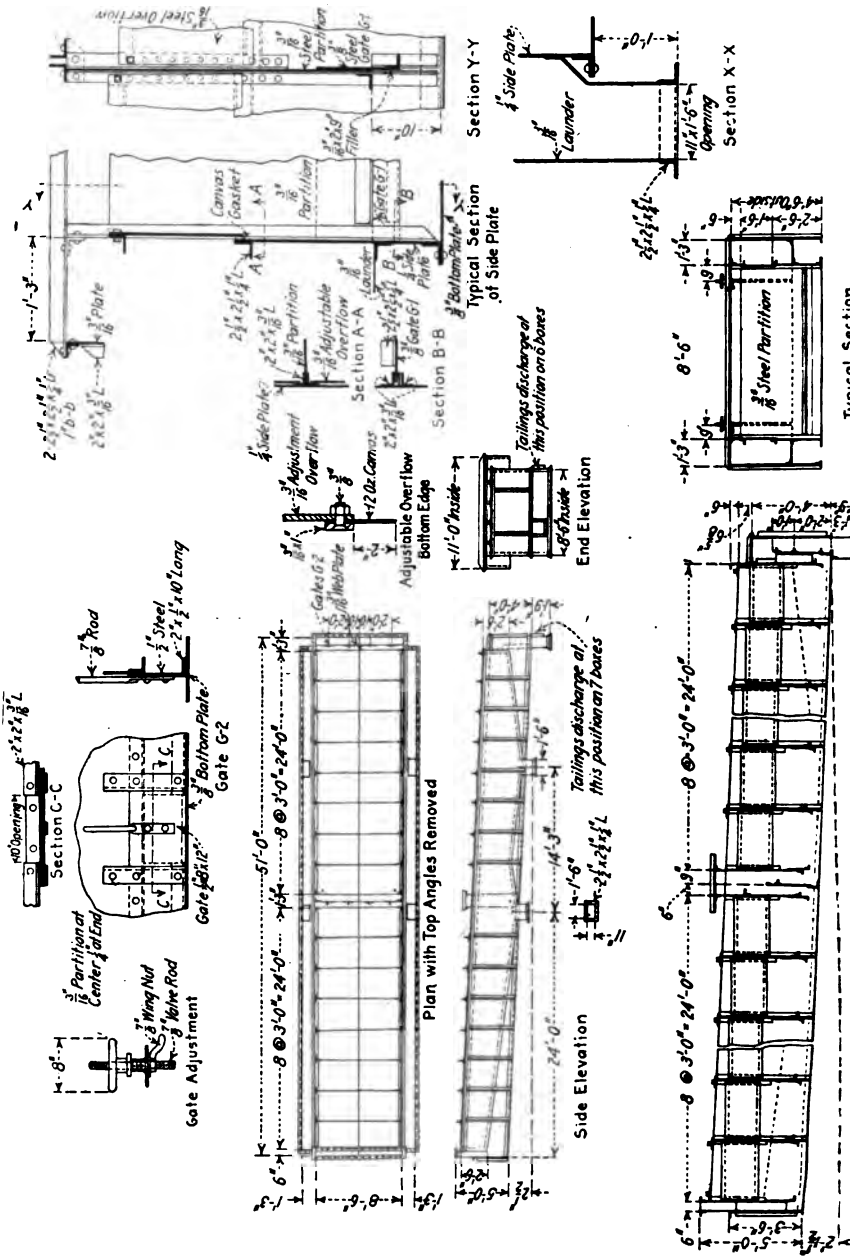
partments near the feed end due to changes in the volume of pulp treated, and might prove highly undesirable.



One point in the construction of the Inspiration flotation machine in which we take considerable pride, is the method in which the air

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introduced. The machines have solid bottoms; all pipe connections are made from above and are at all times visible. The porous bottom rests on

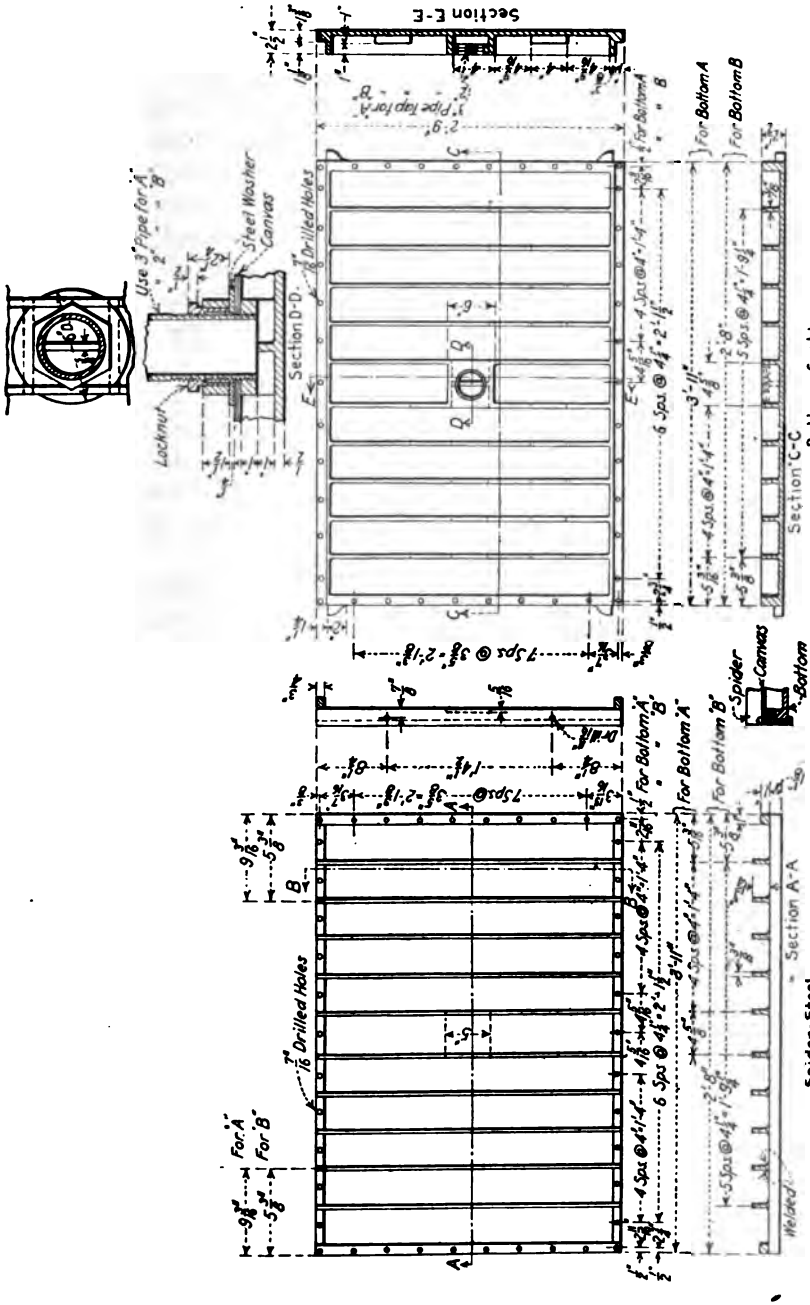

FIG. 22.—INSPIRATION FLOTATION MACHINE BOTTOMS.

top of the regular bottom of the machine and is introduced from above. It is illustrated in Fig. 22. As will be seen, it consists of a lower air

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chamber having partitions running lengthwise, which, however, do n extend down to the bottom of the air chamber and, for this reason, not interfere with the interchange of air between the narrow compa ments thus formed. A grate, which forms the upper part of the poro bottom, can be fastened to the lower part by a number of bolts. Befo putting them together, a porous medium (for instance, canvas) is plac between the two pieces. The air enters through a pipe from the t which screws into the lower casting. Arrangements are made, of cour to secure an air-tight joint where the pipe passes through the poro medium. When assembled, the bars of the upper grate form chann in the lengthwise direction of the machine. The bars are the only pa that protrude above the canvas and, on account of their arrangeme longitudinally, they do not interfere with the passage of pulp through t machine.

The cleaner cells consist of six compartments in series. Bo rougher and cleaner cells are divided by a partition in the middle, ru ning the whole length of the machine. The operator is, therefore, enabl to throw all the feed on one side of the machine when desired, thus p mitting repairs to be made on the compartments of the other side whe ever necessary.

In the beginning, it had been supposed that throwing all the fe on one side of the flotation machine might seriously overload that si and for this reason, in the first sections installed, provisions were made permit removing and replacing the porous bottom from any one compa ment without interrupting the flow of pulp. As far as lifting out t bottom is concerned, this, of course, can be done in any case, but wh one of the bottoms is withdrawn sand will pile up to a certain extent the middle of the compartment. To remove this sand before returni the porous bottom, a system of perforated water pipes was installed un the air chamber. They allow the washing out of the bottom of t machine by the application of water pressure.

Minerals Separation Section

One section of the concentrator is equipped with machines of Minerals Separation Co., Hebbard type. The flow sheet, as illustra in Fig. 23, is practically the same as the one shown in the Inspirat sections. The only difference is that two cleaner cells in parallel used in place of the one cleaner cell of the Inspiration type. The dr ings of the machine are reproduced in Fig. 9.

Filter Plant

It is necessary to reduce the ore to a certain fineness before the mine contents of the feed can be treated by flotation, and as fine concentra

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have the characteristic of retaining water with greater tenacity than coarse concentrates, a filter plant is a necessary adjunct of a flotation plant. In the Inspiration mill, both the flotation and table concentrates are sent to Dorr thickening tanks, five of them being 60 ft. in diameter and three of them 80 ft., representing a total area of about 29,217 sq. ft.,

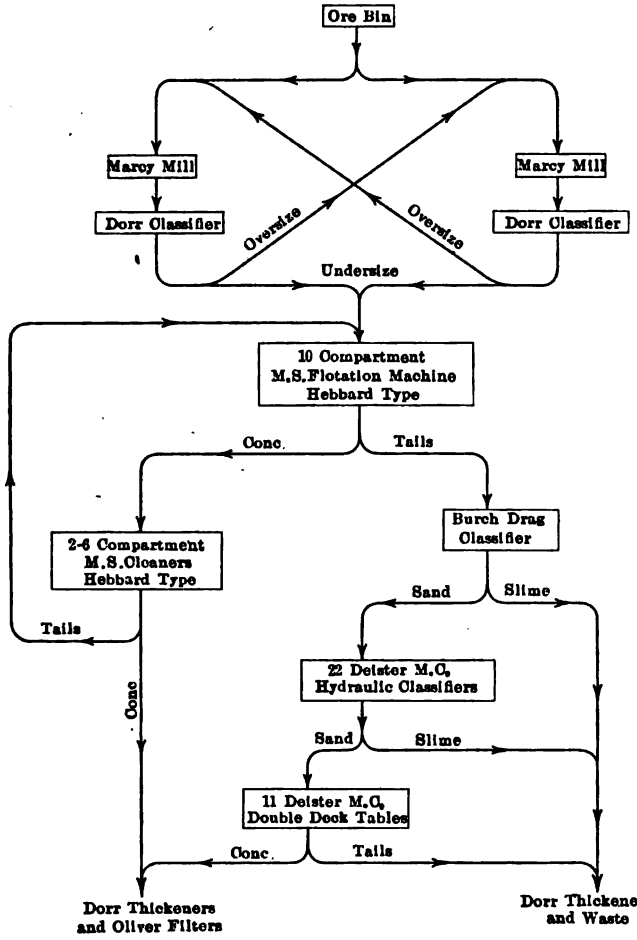

FIG. 23.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTION EQUIPPED WITH MINERALS SEPARATION FLOTATION MACHINES.

or, as the daily production of concentrates amounts to about 600 tons, a settling area of around 48.7 sq. ft. per ton of concentrate sent to the settling tanks. The settling tanks are each provided with a double ring of high baffle boards to prevent the foam that forms on the top from contaminating the overflow. The settling used to be carried out in two steps, some of the tanks serving as preliminary settling tanks, while

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others resettled the overflow from the preliminary tanks. At prese
we operate all the concentrate settling tanks in parallel. The spig
discharges feed six Oliver filters. The ratio of solids to water is about 1.
to 1 at this point. The filters reduce the concentrate moisture to
average of about 17 per cent. of the wet pulp. This figure varies, ho
ever, within rather wide limits from day to day. The moisture conten
of the concentrate seem to depend in the first place on the percentage
insoluble material contained in the concentrates, as will be seen from t
graphic representation of the relation between the two things in Fig. 2
It will be noted that a rise in the percentage of insoluble matter general
corresponds to a rise in the concentrate moisture and *vice versa*.
moisture of 17 per cent. is, of course, in excess of what seems desirab
in view of the fact that the smelter penalizes water contained in t

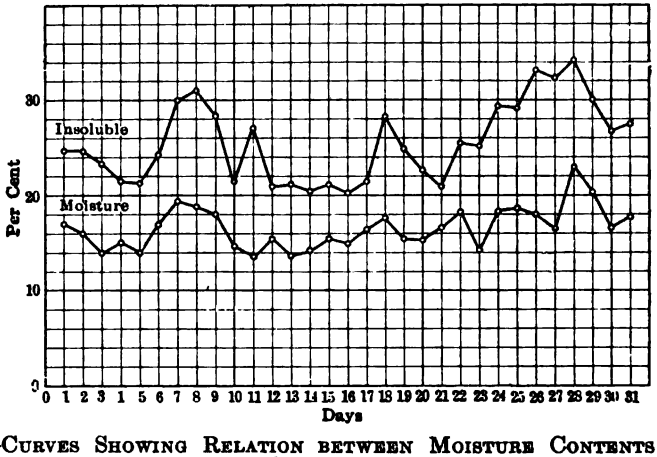

FIG. 24.—CURVES SHOWING RELATION BETWEEN MOISTURE CONTENTS AND
SOLUBLE MATTER IN CONCENTRATES. OCTOBER, 1915.

concentrates. For this reason, several attempts have been made
reduce the moisture. The first plan followed was to inject steam i
the filter tank, with the object of heating the pulp. An apprecia
increase in the capacity resulted from this, while no marked reduction
the moisture contents took place. The same effect was obtained by
addition of slacked lime, which we tried lately in our Oliver filters.
shows itself very readily in the formation of a thicker filter cake, t
causing a larger output of the filters. We have, however, not been a
to effect a reduction in moisture by the use of lime. We have adop
the addition of lime to the filter pulp in regular operation on account
its decided benefit as far as the capacity is concerned. As a rule,
filters can handle the whole output of the concentrator; that is, each
by 12-ft. Oliver filter has a capacity of about 120 tons in 24 hr.

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Results Obtained in Commercial Concentrator

On account of the fact that the concentrator operated by the Inspiration company has only just left the stage of construction, it is not possible to give results obtained in the operation of it as a whole. As, however, an interval of several months has elapsed between the starting of the first and the 18th section, we are in a position to give at least preliminary results derived in actual operation on a fairly large scale.

Tonnage

Each of the sections of the Inspiration concentrator has a rated capacity of 800 tons; at present most of the sections exceed 900 tons actual capacity. Ordinarily, this tonnage is treated on one Standard duplex Inspiration machine consisting of 16 double compartments, while the concentrates are cleaned on a cleaner cell with six double compartments. The size of the roughing compartments is 3 ft. by 4 ft. 3 in.; the size of the cleaner compartments is 3 ft. by 3 ft. Accordingly, the total area of combined rougher and cleaner cells is 492 sq. ft. Assuming the tonnage handled to be 800 tons, this makes a tonnage treated of 1.62 tons per day per square foot of surface. In the course of construction, the erection of flotation machines did not always keep pace with the erection of the grinding mills. For this and other reasons, it has frequently been necessary to combine the product from the two grinding sections in one flotation section; in other words, to throw an overload of 100 per cent. on the flotation machines. Our experience is, that overloading the machines by 100 per cent., or raising the tonnage to 3.24 tons per square foot of surface, does not increase the tailing losses very materially.

It follows, therefore, that the machines handle an overload without serious additional copper loss, and it has been our experience that there is no mechanical trouble whatever, connected with overloading the flotation machines to such an extent. We have also had to resort to overloading the Callow cells, and have likewise found that the recovery does not fall off seriously.

For the Minerals Separation machine of the Hebbard type, which is installed in our Section No. 10, we have no figures of a similar character.

Air and Power Consumption

We have found that three blowers, of a capacity of 8,000 cu. ft. per minute each, are sufficient to supply air for six sections of flotation machines. Accordingly, one section requires 4,000 cu. ft. of air per minute. As the flotation machines in an Inspiration Section have a surface of 492 ft., it follows that the air consumption per square foot of porous surface

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is 11.8 cu. ft. per minute; the maximum pressure required is 4½ lb. the blowers and practically the same at the flotation machines, as the is no serious loss of pressure in the air-conduits. The power consum for furnishing air has been determined carefully by wattmeter readir of the motors furnishing power for the blowers. The actual power co sumed is 87.5 kw. per section; or, figuring again with the rated secti capacity of 800 tons, 2.63 kw.-hr. per ton of ore treated. We have r been able to make separate tests for the Callow and Inspiration sectio but assume that the air consumption is approximately the same in bo We have no figures yet for the Hebbard type machine. To get the to amount of power consumed in connection with the flotation machin a small amount of power, probably about 10 kw. or 0.30 kw.-hr. per t of ore, has to be added to the power consumed for the production of on account of the fact that the cleaner tailings are pumped back to t rougher cells and retreated, and that in the Callow sections the sli overflow of the drag classifiers is repumped to the secondary flotati cells. In the Inspiration sections, 0.425 sq. ft. of actual open porc surface and 0.615 sq. ft. of flotation machine area are provided for t treatment of 1 ton of ore in 24 hr.

Floor Space

The total floor space of the flotation floors amounts to 25,200 sq. f or 1.75 sq. ft. per ton of ore treated in 24 hr. (rated capacity). The to floor space occupied by the concentrator proper (not including, howev the crushing plant located at the mine) is 81,900 sq. ft. or 5.68 sq. ft. p ton of ore, and the total floor space, if the settling-tank installation outsi the main mill building is also included, is 250,890 sq. ft., or 17.4 sq. per ton of ore (rated capacity).

Water Consumption

The water added to the pulp where it reaches the flotation machi is about 3 tons of water for each ton of solid pulp. The subsequ treatment, in our case, requires a further addition of nearly 3 tons water per ton of dry ore. However, as it is carried out on pulp that l been deprived of its slime contents by passing it over drag classifiers, water thus entering the table tailings can be easily removed again. T is accomplished in our case by settling the table tailings separately fr the slime tailings. The settling of the sand tailings is effected in a settl tank of home construction, consisting of a rectangular box with a slop bottom into which a number of Caldecott cones are inserted. The resu ing overflow is not quite clear, but is made so by resettling in three 60 Dorr tanks.

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Out of the total quantity of water actually added to the ore during its passage through the mill, amounting approximately to 6 tons of water per ton of ore treated, as mentioned above, nearly 3 tons, *i.e.*, the amount added during the table treatment, can, as explained above, be reclaimed rather easily. For the reclamation of the rest of the water, settling ponds, of the well-known construction in which the reservoir is formed by building a retaining wall of sand across a gulch, are resorted to in addition to Dorr thickening tanks.

Recovery

The recovery obtainable by the application of our milling process is determined entirely by the composition of the ore, *i.e.*, by the relation between sulphide and oxide copper contained in the ore. Our average sulphide copper extraction has been 90.39 per cent. for the months of March, April and May, 1916, the last months for which figures were available at the writing of this paper. A certain recovery of the oxide-copper minerals, especially carbonates, is made in the flotation process as well as in the gravity concentration process. The percentage of such mineral recovery is low, probably about 25 per cent. For this reason, the recovery obtainable on ore containing a high amount of oxide, such as surface ore, is correspondingly lower. We have worked in the laboratory with the object in view of increasing the oxide recovery; for instance, by adding certain chemicals to the flotation pulp. An account of the results obtained will be found below, but we have not yet applied this method to an operating scale, nor have we decided on using one of the other methods applicable for this purpose, such as leaching.

Table 5 gives average screen analyses of the feed and the general tailings of the Inspiration concentrator for the months of March, April and May. A segregation is made in the copper assay between sulphide and oxide copper, because, considering the present stage of the art, we feel satisfied with our mill work whenever the sulphide copper content of the mill tailings is low. As will be seen from the tabulations, a better recovery is made on the -200 material than on the coarser constituents of the ore, which proves the point that for ores of the character of Inspiration ore sliming is not to be feared since the introduction of the flotation process.

Oil Consumption

Experience has shown that in the flotation treatment of our ores, we consume flotation agents up to 1½ lb. per ton of ore. At present, the oil mixture contains about 95 per cent. crude coal tar and a little less than 5 per cent. of oils derived from the dry distillation of wood.

The different tars that we have tested during the operation of our mill have shown greatly varying qualities as far as their flotation value is

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concerned. The first tar that we tested and made use of was hom made from domestic coal, and happened to be a very serviceable flo tion agent. Since that time, we have tested tars from several Sta and have found some that are suitable for our purposes, while others a less so, and still others even entirely unsatisfactory. We have obtain satisfactory tar products from New Mexico, Colorado, Missouri a Illinois. These States furnish at present as much as we need for c consumption. For awhile, it seemed possible that we might have import from a long distance the large quantities of tar that we requi During that period we tried to find substitutes, and looked especia toward the utilization of fuel oil for this purpose, but we have not be able to get as good results with any kind of fuel oil as with crude coal t

TABLE 5.—*Average Screen Analyses for the Months of March, April and May*, 1916, *of Flotation Feed and General Tails*

Mesh	Flotation Feed				General Tails							
	Per Cent. Weight		Copper Contents		Per Cent. Weight		Sulphide Copper Contents		Oxide Copper Contents		Total Cop Content	
	Cum.	Indiv.	Per Cent.	Grams	Cum.	Indiv.	Per Cent.	Grams	Per Cent.	Grams	Per Cent.	Gr
+ 65	9.5	9.5	0.45	0.042	9.5	9.5	0.18	0.017	0.12	0.011	0.30	0.
+100	21.2	11.7	0.86	0.101	21.2	11.7	0.19	0.023	0.14	0.016	0.33	0.
+150	33.5	12.3	1.91	0.235	33.5	12.3	0.11	0.014	0.19	0.023	0.30	0.
+200	39.2	5.7	2.69	0.154	39.2	5.7	0.14	0.008	0.19	0.011	0.34	0.
−200	....	60.8	1.85	1.125	....	60.8	0.06	0.036	0.47	0.286	0.53	0.
Totals.....	....	100.0	....	1.657	....	100.0	....	0.098	....	0.347	....	0
Assay direct	....	....	1.62	....	....	....	0.101	....	0.318	....	0.419	
Oxide.....	....	....	0.36									

Our experience is, that we can get along with coal tar alone. is beneficial, however, to add wood-distillation products in small quan ties, for instance, those containing pine oil. While coal tar makes a v strong and heavy froth, such as appears to be required to keep coa mineral particles in suspension, the wood-distillation products have characteristic of producing a multitude of froth bubbles, such as se necessary to furnish the large surface required to save the very mineral particles. Because the finer ore particles (slime), on acco of their small diameter, expose a large surface, it is evidently necess to produce a correspondingly large surface of froth in order to s them by flotation.

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Operating Cost for Flotation

The number of men necessary for the operation of large flotation machines is remarkably small. At the Inspiration plant, one operator supervises four sections of flotation machines. Two Mexican helpers assist him in washing the bottoms, thus insuring a free passage of air through the porous medium. At the prevailing high prices of American and Mexican labor, this means an expense of somewhat more than 1.5 c. per ton of ore treated. The total expenses representing flotation proper were as follows for the months of March, April and May, 1916:

	Cents per Ton
Labor	1.62
Flotation oils	1.65
Other supplies	0.35
Power	2.14
Total	5.76

The subsequent table treatment of flotation tailings, the filter treatment of the concentrates and other operations connected with the process of concentration, belong more or less to flotation treatment, and their expense should also be considered when the cost of the flotation process is to be established. The total milling cost, exclusive of crushing and grinding, has been for the past few months in the neighborhood of 20 c. When the cost of crushing and grinding is included, the cost is about 40 c. per ton of ore. Royalties for the use of the flotation process are not included in any of these cost figures.

Discussion of Results Obtained

The criticism has occasionally been raised against the Inspiration company that they were slow in deciding on the design of a concentrator. The reason for such slowness was, of course, that the development of the flotation process coincided with the period during which the fundamental points in the design of the concentrator had to be settled. The management, therefore, thought it best to be very conservative in installing standard concentrator equipment, which, in case the new process held what it seemed to promise, might prove to be entirely superfluous. The main point of interest, therefore, is whether it was wise for the company to wait instead of installing a gravity concentrating plant as had been originally considered.

As far as the comparison of a flotation plant, as at present operated by the company, with the gravity-concentration plant, as originally considered, is concerned, the recoveries actually obtained in the company's flotation plant are so much higher than those indicated by tests in the

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original test mill as being obtainable in a gravity-concentration plant ore of the same character, that there cannot be the least doubt whi system is the better.

In discussing the second question which comes up in this connecti whether, for ores of the character of Inspiration ore, gravity concentrati supplemented by flotation would be preferable to the simple flotati process that has been adopted by the Inspiration company, the cruc point is this: Can the slime be reduced to lower copper contents by combined process than by the process followed here? The reduction the copper in the sand is, as millmen know, simply a question of f grinding combined with suitable concentration.

In support of the principle followed by the Inspiration company desire to call attention to the screen analyses of our general tailings rep sented in Table 5. They show that the sulphide copper left in t −200-mesh material is very low.

In addition, the floor space required for a plant like this is materia smaller than for a plant of the combination plan, which also means th the construction cost is much lower; and, further, the operating costs low by reason of the greater simplicity of design. I hope these facts sufficient to decide the point at issue as far as the ore of this district concerned.

I hope that our neighbors of the Miami company, whose mill was bu before the flotation process was known, will uphold me in my statemen If this point is admitted, it will endorse the principle followed in the desi of the Inspiration mill, of doing away with reduction in stages and co centration in stages.

It must be granted that the settling of flotation-concentrate pu requires extensive floor space, as shown by the figures referred to abo and that settling and subsequent filtering absorb a certain fraction of t mill operating costs. However, as a ratio of concentration in the Inspi tion mill is 25 into 1, only $\frac{1}{25}$ of the ore is thus treated. The actu expenses increase, of course, proportionately for ore which does not p mit concentration with such a high ratio as the ore existing in the spiration mine. Therefore, there will be a point beyond which a differe system of concentration is preferable. Just what the critical ratio concentration is, must be determined by calculation and testing bas on individual conditions.

Prospects for Future Development of the Flotation Proces

The flotation process is in its infancy. For this reason, therefore, concentrator must be necessarily in the first stages of its developme In what direction future changes may take place, is perhaps indicated tests which have been made partly on a laboratory scale and partly o

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somewhat larger scale, but which have not yet been incorporated into our regular milling process. Of these latent developments, I will try to give an outline in the following:

Porous-Bottom Experiments

The porous bottom is, as one may imagine, the most essential part of a pneumatic-flotation machine. Our experience with the porous bottoms of the different constructions brought out very clearly the principal difficulty attached to them, which is, that the pores have a tendency to contract gradually and thereby to retard the passage of air through them. This tendency was more pronounced in the solid porous bottoms employed in the Flinn-Towne flotation machine than it was, for instance, in those of the Callow type, although the latter also show a tendency in this direction. Our first supposition was that the choking was due to the fact that the air entering below the blankets carried particles of dust, which would settle in the fine pores and reduce their area. Indeed, a canvas blanket will, after a certain length of service as a porous medium, always show some ring-shaped spots of dark color opposite the air inlets, clearly indicating that a deposition of dust particles on the blanket actually does take place. To make sure of this point we cut out round disks from a Callow blanket that had been used for some time and investigated their porosity by using them as porous bottoms in a glass tube standing in a vertical position. Air under pressure could be applied to an air chamber located underneath these disks, and the air passing through the porous blanket could be measured by a gas meter. The quantity of air discharged through the porous medium offers a measure of the porosity of the blanket, and for this reason, the velocity or speed with which the counter of the gas meter revolves, gives an indication of the porosity of the porous disks being tested. To our surprise, we found that the darkest points of the blanket were not those of lowest porosity. On the contrary, the points farthest away from the air inlet showed the greatest tendency to choke. An explanation of this paradoxical behavior seems to be offered by the fact that an air blanket is kept in a state of more or less agitation near the air inlet (in the Callow machine this happens to be a point remote from the places where it is held rigid) while farthest away from this point the blanket assumes a state of comparative rest. Incrustations, due perhaps to the presence of soluble salts in the water in conjunction with fine slime, always form to a greater or less extent in the top layer of the blanket. Evidently, the agitation counteracts the formation of the incrustation, while there is no such counteracting influence in the portions which are essentially at rest. For this reason, we concluded that a solid porous material is not suitable as a diaphragm in a flotation machine of the pneumatic type, if a bottom of long life is required. As a matter of

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fact, the experience of everybody who experimented with solid botto
seems to have pointed in the same direction. Mr. Cole for a while test
out carborundum tubes in his machine. We tried carborundum ston
in the flotation machine of the Inspiration type and abandoned the
and I believe that even Messrs. Flinn and Towne have, in the meantim
given up the solid bottom of their original design.

A necessary condition for a serviceable flotation bottom appea
therefore, that the porous medium be of a flexible nature. The 4-p
canvas stitched every half inch or so which Mr. Callow's first cells co
tained and which we have used for considerable time in the Inspirati
machines, seems to answer this purpose fairly well. We find, howeve
that to keep it in good working condition and prevent incrustations fro
forming on the top, we have to clean it frequently. This is done
dipping an iron pipe connected with a water hose into the compartmen
and sweeping the canvas bottom with the jet of water discharging fro
the lower end of the pipe. The canvas blankets seem to last for abo
6 months at the most. As they are inexpensive, the replacing of a botto
after that time is not a serious item in the operating costs. The givi
out of the canvas is due to the wear caused by the frequent cleanin
The top layer wears out first, the holes created by the stitching formi
nuclei for the formation of larger holes. By the time the top layer h
a number of holes the canvas blanket is generally discarded. In t
interest of greater economy, we intend giving up interstitching the laye
of canvas. We are trying to decide whether it is better to use sing
sheets of thicker fabric or to use canvas similar to the kind that we ha
been using and to put several layers on top of one another witho
interstitching them. The latter has the advantage of requiring the d
carding of only one layer, when it becomes defective.

There will always be some tendency to form incrustations so long
canvas is used for flotation mediums. Their formation will be entire
prevented only by substituting an altogether different material.
have made experiments in this direction. One of my former assistan
R. H. Haskell, deserves credit for suggesting them. For instance,
substituted for the canvas blankets, thin rubber sheets perforated wit
multitude of needle holes and obtained an excellent froth. The obj
tion to their use is that their life is limited. When sheets of rubber
an increased thickness are used, the needle holes require too much pr
sure to form openings of sufficient size for the passage of air, and to ma
a thick rubber sheet suitable for this purpose, slits several millimet
long have to be substituted for needle holes. We have had one or t
rubber bottoms of this design in operation, but, just at present we
not ready to substitute them for canvas blankets. We also tried
blanket made from a material that goes under the name of sponge rubb
and can be produced with rather fine texture. We have not been ab

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however, to obtain lastingly good results from the use of this medium. Furthermore, we tried a woven fabric containing rubber threads in one direction and threads of cotton or the like in the other direction and a rubberized canvas made by the Goodrich Rubber Co. We are not prepared to use any of these materials on a commercial scale.

The advantage of rubber should be, in the first place, that on account of its smoothness it would have less tendency than canvas to permit the formation of incrustations. Besides, an elastic medium should have the additional advantage of avoiding the danger of catching small sand or slime particles in the pores of the medium, as an expansion of the medium (that may be effected, for instance, by increasing the pressure) would widen the pores and remove such particles. We think that our experimental work in this direction is encouraging.

Raising the Grade of Concentrates

The recovery that it is possible to effect in a flotation plant depends largely on the grade of concentrate desired. With a low grade of concentrate, low tailings can be made, but when a high grade of concentrate is stipulated, increased tailing losses cannot be avoided. A question that suggests itself in this connection, and which we have tried to answer by laboratory experiments is, "How can we raise the grade of our concentrates—that is, reduce the percentage of insoluble matter contained in them—without entailing additional copper losses?" We know from laboratory experiments that this can be done by expensive methods—for instance, by heating the solutions—but such a procedure would be undesirable from an economical standpoint. Experience has shown us that concentrate produced in the first compartments of the cleaner cells is always freer from insoluble matter than the concentrate produced in the last compartments. The problem then resolves itself into finding a suitable cleaning process for the concentrate from the last compartments of the cleaning cells. By treating this low-grade concentrate hot, with the addition of caustic soda, we have been able to separate it into a high-grade concentrate and fairly low tailings. This method necessitates only the expense of heating a small fraction of the pulp and may be a commercial possibility.

Recovering Carbonates by Flotation

Another subject on which we have spent considerable time in our laboratory is the problem of recovering copper carbonates by flotation. When we started our flotation plant, we discovered, to our astonishment, that the machines not only saved a high percentage of copper sulphide but that they also recovered some of the carbonates. Ever since that time, we have tried to find means of improving the carbonate recovery.

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In the first place, we studied all of the oils that seemed to have tendency to cause the flotation of such minerals. Later on, we tri other means in addition to the variation of the oils. One way in whi copper carbonates and similar minerals might be recovered was outlin by Alfred Schwartz in his United States Patent No. 807,501. T process consists in first artificially producing a sulphide coating on su oxidized minerals by the introduction into the pulp of soluble sulphid and then adding suitable "oils" and effecting the flotation. If it we possible to thus chemically produce coatings of sulphide identical wi the surface of the minerals formed by nature, this process would wo well, as evidently the nature of the surface is the only characteris that determines whether a mineral will float or not.

The Minerals Separation Co. owns a number of patents coveri this subject. Their English Patent No. 26,019, issued to Sulman a Picard, describes the flotation of oxide copper minerals by similar mea

I am not aware that equivalent patents have been issued in the Unit States. The English patent in question is of a later date than t Schwartz patent above mentioned. The representatives of t Minerals Separation Co. have experimented more or less extensive with this system, while demonstrating their machine to the Inspirati company. As far as I know, they have not proven its practicabili In the course of their experiments, they tried the application of sodiu sulphide and sodium polysulphide for this purpose. The latter w produced by treating sulphur with hot caustic soda. At the time th experiments were made, I was not familiar with the chemical acti taking place, which, as much as I know now, actually results in the form tion of polysulphide mixed with thiosulphates and other oxygen-sulph compounds. The failure of their experiments, I therefore ascribed the fact that perhaps a polysulphide, which they were anxious to ma was not actually produced. I proceeded to make sodium polysulphi by a method which I knew, that is, by the treatment of a sodium s phide solution with sulphur powder. When we applied this reag to some of our carbonate ores in laboratory flotation experimen we noted that a good recovery was obtained. The composition of t compound was varied materially in order to find just what compositi gives the best results in the flotation of carbonates. Our experie seems to indicate that sodium sulphide alone encourages the flotati of carbonates, but that sodium polysulphide, or sodium sulphide wh contains more sulphur than would correspond to the chemical form Na_2S gives better results. The addition of caustic soda besides sodium polysulphide was found beneficial.

The question then arose as to why we succeeded in effecting flotation of oxidized copper when the experiments of the members the Minerals Separation staff failed. Tests along these lines brou

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out the fact that the Minerals Separation compound when applied to our carbonate ores also worked successfully, but that it did not on our regular milling ore. Our own compound when added to our mill ore mixture increased the recovery of the carbonates, but evidently interfered with the sulphide extraction, and for this reason seemed to be of as little use as the compound of the Minerals Separation Co. When applying reagents of this character to tailings resulting from ordinary flotation treatment, with the point in view of effecting a sufficient sulphide extraction by the regular flotation process, and using the compound in question only for the purpose of increasing the carbonate extraction, we have found so far that the increase in copper-carbonate recovery over the one obtained without the addition of such chemical compounds is not worth going after.

But this is only a consequence of the fact that carbonates exist in very small amounts only in our milling ore and are partly saved by the ordinary flotation process.

There is no real difficulty about saving carbonates by the method mentioned, if they exist in quantities that make it worth while to save them. That copper carbonates can be recovered may easily be demonstrated by treating a deslimed feed in a series flotation machine. If at the point of the machine, where the sulphide recovery is nearly finished, sodium sulphide is added, the decidedly green color of the concentrates in the following compartments leaves no doubt on this point. The desliming of the feed seems to assist in the carbonate recovery.

It is of considerable (even if only theoretical) value to establish why sodium sulphide and polysulphide tend to increase the recovery of copper carbonates. A coating that might be expected to form cannot be detected. The concentrate resulting from the treatment of pure carbonate ore is decidedly green; besides, when an alkaline condition of the pulp is used there is very little, if any, tendency for any sulphide coating to form, and the alkaline state of the pulp is (as explained above) exactly the condition under which the best carbonate extraction results. Another point that seems to contradict the explanation of these results by the assumption of a sulphide coating is, that when we proceeded exactly as suggested by Mr. Schwartz—*i.e.*, when the application of soluble sulphide was followed by the addition of flotation agents and by the actual flotation—we seemed to obtain poorer results than when the procedure was reversed by applying the oil first and following with the application of some soluble sulphide, although the latter method would certainly seem less favorable to the formation of a sulphide coating, and perhaps for this reason has not been suggested by Mr. Schwartz.

Another theory that has been mentioned as an explanation of this phenomenon is that colloidal sulphur is formed by the solution of sodium polysulphide in water, which, as is known, is a good flotation agent. For

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instance, it is pointed out in the United States Patent No. 1,140,86 taken out by Dr. R. F. Bacon of the Mellon Institute in Pittsburgh that by setting free colloidal sulphur, say by the reaction of a solub sulphide with sulphur dioxide, good flotation results may be obtaine as far as the flotation of sulphides is concerned. To make the proces available for the flotation of carbonates and other oxidized copper mi erals, he suggests that a sulphide coating be first formed on the mineral *i.e.*, to follow Mr. Schwartz's idea. Whether the colloidal sulphur b itself has a beneficial influence on the recovery of the carbonate (as he been suggested in explanation of our observations) seems rather doubtfu when it is considered that we have obtained good results in alkaline solu tions in which colloidal sulphur does not seem to separate out from poly sulphide containing only a limited amount of sulphur such as was used i our tests. The full theoretical explanation of these facts must therefo be left to future investigations.

Recovery of Silicates

In our experiments with the object of saving the oxidized copp minerals, we soon found that we could save some of these minerals, whi others were entirely refractory to the method above mentioned. T establish which minerals could be saved and which not, we attempte an analytical separation into carbonates and silicates. The chemic methods which we tried for the purpose of distinguishing between th two proved unreliable, however, and we had to resort to the separatio by specific gravity (panning). The carbonates of copper (malachite an azurite) are heavier than gangue, and the silicates (chrysocolla) a lighter. The separation is rather difficult, owing to the small differen in specific gravity, and the results are therefore far from being altogeth reliable, but they seem accurate enough to indicate that the method saving carbonate copper above referred to is of value only for the recove of carbonates and does not apply to silicates. This fact seems to another corroboration of the assumption made above, that carbonates copper do not float simply because of the formation of a thin surfa coating of copper sulphide. It can easily be verified in the laborato that silicates can be coated with copper sulphide fully as easily as copp carbonates. For this reason, if the filming theory is right, it should possible to float silicates just as well as carbonates. There is no dou that they can be floated by transformation into sulphides, only th transformation must not be confined to the surface, but must go deepe Our experience is, that to effect a good recovery, it is necessary to acidi the pulp so strongly that practically all of the silicates of copper are d solved and by the action of hydrogen sulphide or other soluble sulphid are transformed into the state of chemically precipitated copper sulphi

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In this form there is no difficulty about the recovery of the copper by flotation, but this procedure is not entirely without objection.

In case hydrogen sulphide gas is used, the acid combined with copper is regenerated. This tends toward a low acid consumption and a good copper extraction, on account of the fact that the treatment winds up with a small percentage of copper in solution and free acid present, both of which are desirable in the light of the law of chemical mass action. But hydrogen sulphide is not a desirable reagent. The fact that it is a gas and not a liquid introduces complications in the apparatus which are accentuated by the fact that it is poisonous and obnoxious otherwise.

Other soluble sulphides used in place of hydrogen sulphide will neutralize some sulphuric acid with the result that the acid consumption will be higher and the copper extraction lower than in case of hydrogen sulphide gas.

As far as acid consumption is concerned, it is pointed out that the free acid lost with the pulp may be settled out in ponds and re-used. However, the re-use of acid diluted to such an extent is a more serious problem than is generally realized.

The treatment of concentrates that are "colloidal," to a much greater extent than ores which millmen have been in the habit of calling colloidal, offers additional problems, which, however, may prove not to be as serious as they look.

Everything considered, I cannot see that the flotation treatment of oxidized copper ores after previously leaching them offers better prospects than straight leaching by decantation and precipitation by other methods.

General Theory

Very much has been published recently about the theory of the flotation process, and very many suggestions have been made that will probably prove valuable after it has been shown by critical tests how far they explain the facts.

It seems to me that an explanation of the qualities of the flotation oils is not as difficult as it might appear. The problem only seems so complicated because the flotation qualities of an oil or an oil mixture have not been separated into their components. In fact, it requires a combination of qualities to make a successful flotation oil. In the first place, the flotation oil has to coat the mineral particles. That there is a tendency for the formation of such a coating can easily be seen from simple experiments. For instance, if samples of copper sulphide (chalcocite), copper carbonate (malachite) and gangue (silica) of the same screen size are spread out on watch glasses and then moistened with a drop of coal-tar creosote, it will be seen that the drop of creosote soon disappears through absorption by the copper sulphide, while it takes a much longer time for it to be absorbed by the copper carbonate and a still longer time

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with the gangue. On the other hand, when a drop of water is placed on the same minerals, it will disappear on the gangue first, later on the carbonate, and finally on the copper sulphide. This evidently proves that in a mixture of water and oil, the oil will attach itself with preference to the sulphide particles while the water will have the greater tendency to wet the gangue.

The second quality which at least is sometimes required of a flotation oil, is that it has to form a stable froth. In such a case, the stability may be secured by more firmly cementing together the mineral, air and oil. To accomplish this, oils are used which have a tendency to float finely divided gangue particles. The action is characteristic of the heavier pine distillates like pine tar and the lighter ones like turpentine if they are crude, unrefined products; in other words, when they contain some of the heavier distillates. I am not quite sure, however, whether the beneficial influence of oils of this group is not perhaps rather due to the fact that they remove colloidal material from the pulp and thereby improve its tendency to float minerals.

A third quality demanded of a successful oil mixture is that it must be able to produce a sufficient volume of froth. This property is exemplified best by oils of the soluble type—cresol, pine oil, alcohols and other substances. It can easily be proven that when oils of this type are used, although they may be considered insoluble, the water acquires the frothing qualities of the oil. It may be demonstrated by shaking an oil of this character with water and permitting the oil to separate out again. It will be found that the water has acquired frothing qualities by undergoing this treatment. It is perhaps even likely that the soluble portion of the oils belonging to this group is the only portion that is active in this manner. The difference between the oils of group 1 and group 3 may be studied, for instance in a flotation machine of our type. It will be noted that the heavier mineral runs over the concentrate discharge largely in the first compartments forming a heavy, dark froth and the heavy insoluble portions of the flotation oil mixtures apparently go with it. Toward the tailings end of the flotation machines, most of this dark material has disappeared and the froth is lighter and of a more watery nature. The pulp, however, has not lost the quality of forming froth even after it gets to the last compartment of the flotation machines. This permits the conclusion that the frothing characteristics follow the tailings pulp. The water settled out in tanks and tailing ponds has decided frothing qualities. Such water behaves in a similar way to certain alcoholic solutions with which we are used to associate this characteristic, for instance, beer or champagne. The experience of mills using the flotation process, that when the tailings water is reclaimed the quantity of frothing oil may be considerably reduced, further supports the assumption that the formation of froth is caused by water-soluble substances.

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Just how the surface tension of water and air must be modified to permit the formation of froth has been made the subject of some speculations recently published by different authors. We have also devoted some thought and a few experiments in our laboratory to this question. A discussion of these matters belongs to the realm of physics, however, and is outside the scope of this paper. But I might add this remark to the discussion of the subject of flotation oils, that most flotation oils not only have the characteristics of one group, but may at the same time possess those of another one. For instance, coal tar has qualities 1 and 3. For the flotation of our ordinary milling ore we do not require much of the quality of oil classified in group No. 2, and, therefore, can get along with coal tar alone. But we find it advisable to add to the coal tar more of the qualities characteristic of the third group, and for this purpose, we add about 5 per cent. of the total in the form of crude pine oil. In many cases it is found that the flotation oil has the characteristics of group No. 2 to such an extent that it is impossible to make clean concentrates. Various chemical means, such as the addition of acid or alkali, are used to counteract this.

Conclusion

In summing up I want to say that the fact that the Inspiration company has been able to design a commercially successful flotation plant and has found ways that hold out prospects of raising the plant to a very high state of efficiency, must be attributed to the policy followed by the company of spending great sums of money for the purpose of investigating the flotation process on a commercial scale. In carrying out these investigations, a close coöperation between laboratory and operating force helped us, I believe, more than anything else. I would like to give credit to each person who had a share in contributing toward the success of the work, but cannot do it, because the list would be too long.

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A New Flotation Oil

BY MAXWELL ADAMS, RENO, NEV.

(Arizona Meeting, September, 1916)

CONSIDERABLE interest has recently been developed in sage-brush oil because of its possible utilization as a flotation agent in the mining industry. A list of some of its physical properties, together with the method used in its extraction, may prove of interest at this time.

Something over a year ago, a study of the essential oils in desert plants was begun in the Chemical Laboratory of the University of Nevada. None of the oils so far studied possess properties of special interest to engineers, except the oil of sage, *Artemesia tridentate*, which has exceptional power as a flotation agent. This plant, known as common sage brush, also called black sage, is widely distributed over the semi-arid West, being found quite generally on most of the dry plains and mountains west of Missouri.

The method of extracting the oil followed in these experiments is very simple. The leaves, twigs and small branches are placed in an air-tight drum, having a capacity of about 27 cu. ft. Steam is admitted through a number of small openings at the bottom of the retort, and the pressure maintained at 20 to 25 lb. per sq. in. for 3 hr. The escape of the steam from the retort is regulated by allowing it to pass through a stop-cock into a condenser. The water in the receiver is drawn off from time to time and the oil, which is insoluble and floats upon the water, is thus collected. At the end of 2 hr. most of the oil has been driven out, though traces continue to come over for a much longer time. By raising the pressure, the time required could probably be shortened and the yield increased, but the lack of laboratory equipment has prevented the carrying out of this experiment.

The stock wood, bark and branches contain no oil, the distribution of the oil being limited to the leaves and young shoots. There is a seasonal variation in the amount of oil contained. Samples collected on different dates gave the following amount of oil: May 1, 0.42 per cent.; May 27, 0.6 per cent.; June 30, 0.72 per cent.; Aug. 1, 0.9 per cent.; Sept. 10, 1.0 per cent. The increase appears fairly constant from early spring, when the leaves first appear, until light frosts occur in the autumn. When the plant is air-dried there is some loss of oil, as the

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following data will show: Two 100-lb. samples were collected at th same time. One was distilled when green; the other was air dried fo 10 days before distillation. The green sample yielded 275 grams, an the dried sample 248 grams of oil, showing a loss of about 10 per cent.

A laboratory experiment can furnish little data useful in forming a estimate of the commercial cost of production. A man working for hr., and using a pair of common pruning shears, collected twigs whic yielded 1 lb. of oil. Since only a small percentage of the oil is lost the brush is dried, the most economical method of production woul perhaps be to collect it in large quantities, by using a tractor engine an a drag, in some such way as land is cleared for farming. When the brus is dry, the leaves and young shoots are easily shaken from the limb Thus the amount of material to be distilled would be greatly diminishe and the oil perhaps obtained at a cost and in quantity sufficient to ma it available as a flotation oil, if not alone, possibly as an ingredient, increase the flotative power of other oils.

The crude oil is dark in color. When redistilled with steam it water-white at first, changing gradually to a straw-yellow color upo standing. It has the following physical properties: Density at 15°C 0.9206. Refractive index at 20°C., 1.4732. Rotation at 20°C., −4.6 At 98°C., a light oil, with a very sharp and pungent odor, begins to di till, but only after the temperature is above 165°C. does rapid distillati take place. At 180°C., the oil turns dark and decomposition begin At a pressure of 12 mm., and below 125°C. almost all the oil can b distilled.

The chemical properties of the oil are as yet undetermined. The are small quantities of alpha and beta pinene. The main part of th oil has a camphor-like odor and taste, but has failed to give the ordinar tests for ketones. The fraction boiling at 175° to 180°C. has some the properties of ordinary cineol, but is acted upon by metallic sodium which indicates that the chief ingredient is not cineol. The chemic composition, which has little interest in this connection, will be worke out later. The important question for the engineer is: Can the o be produced in quantity and at a cost that will make it available for o flotation?

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A New Source of Flotative Agents

BY G. H. CLEVENGER, PALO ALTO, CAL.

(Arizona Meeting, September, 1916)

THE reagents now used in flotation consist of various acids or salts, which may be either electrolytes or non-electrolytes, dissolved in water and some substance or combination of substances which function as collecting or frothing agents. At times the only dissolved salts present are those naturally occurring in the water used. The general effect of the electrolyte is to greatly sharpen the separation between the gangue and the concentrate. Examples of this are: The use of sulphuric acid with zinc ores; and of sodium carbonate or calcium oxide (lime) with silver-gold ores. Crude pyroligneous acid is also sometimes used when available. Various oxidizing agents, such as permanganates, bichromates, etc., are added in the selective flotation of lead-zinc ores. Many other reagents for performing certain specific functions, either real or imaginary, have been proposed, and a number of them have been tried upon a working scale. The wild orgy of experimentation which is now going on in flotation exceeds even that which followed the introduction of the Washoe process for the treatment of Comstock ores, when, among other things, sage-brush tea and tobacco juice were reagents added to the pans. Out of all this will, of course, eventually come a more or less standard practice for the treatment of each class of ore.

Omitting a discussion of the functions of frothing and collecting agents, the reagents used in flotation for this purpose may be classified under five general heads: (1) Essential oils; (2) fixed or fatty oils; (3) alcohols and their combinations with organic acids; (4) coal tar and its refined products; (5) petroleum and its refined products.

A flotative agent of some kind is required in flotation as now practiced. A single reagent, as, for example, certain of the essential oils, may perform the dual function of frothing and collecting agent, or a mixture of substances may be required.

In this country, the essential oils used have been wood products, and have been almost exclusively confined to the steam-distilled pine oils. At the present time, the supply of these is limited, and the cost almost prohibitive, so that their use has been dispensed with as far as possible. The oils and tars resulting from the destructive distillation of

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the various species of the coniferæ find a more general use on account
the greater supply and lower cost, although even these have risen in pri
and the future supply is somewhat problematical if the demand f
them in flotation continues to increase at the same rate as in the past.

The fixed or fatty oils, with the exception of oleic acid and cru
pyroligneous acid, are scarcely ever used, since they usually give po
results as compared with other flotative agents.

Coal tar is a common and cheap product, but not all coal tars a
suitable for use in flotation. Furthermore, the marketing of coal tar
coming more and more into the hands of large distributors, who contra
for the output of the various gas plants. The refined products of co
tar are useful flotative agents, but in the case of certain of these, for e
ample, carbolic acid and cresol, the price at present has become prohi
tive. In general, crude coal tar yields the best results when mixed wi
other oils. It is a frequent practice to use a small proportion of pine o
in order to modify the froth. Fortunately, in many cases these cheap
mixtures yield very satisfactory results, and can now be obtained at
reasonable cost.

It appears that only the crude petroleum oils having an asphal
base are generally useful in flotation. These are invariably mixed wi
other oils. The refined products of petroleum are usually not sat
factory, except as constituents of oil mixtures. The one exception
this is perhaps the case of kerosene acid sludge, which is a byproduct
petroleum refining. Kerosene acid sludge from eastern refineries ga
unsatisfactory results at Anaconda, but California kerosene acid sludg
resulting from the refining of crude oils having an asphaltic base,
satisfactorily used in conjunction with a small proportion of wood cre
sote for the treatment of Anaconda copper ores. While California ker
sene acid sludge is a satisfactory flotative agent with many ores, t
available supply tends to limit its use. Formerly California refine
threw it away, but now little of it is available upon the open mark
since practically the whole of the present production is contracted for
long time in advance by pioneer users.

A suitable oil supply, both as regards the character of oil to give t
best results and its present and future availability, is a matter of seric
consequence to companies operating flotation plants. It was with th
in mind that I began the investigation of possible sources of flotati
agents, with the particular object of affording relief to mines located
the arid regions of the West, where none of the common flotative agen
are locally available, and where transportation costs upon those from t
outside are high.

Certain plants and shrubs have the peculiar property of secreting
in the new growth, during the growing season, and particularly in t
leaves. Botanists appear to be uncertain regarding the function of t

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oil in the metabolism of the plant. Some hold that it is a reserve food supply for the plant, while others believe that it is waste product which the plant fails to throw off. Be this as it may, many plants, shrubs and trees contain oil in the leaves and new growth. A good example of this is the case of the eucalyptus tree, from the leaves of which essential oils, which are used for various purposes, are recovered by steam distillation. The leaves of the variety known as amygdalina, unique as being the tallest tree in the world, have afforded a large proportion of the flotative agent used in the concentration of complex Australian ores. This variety of tree is fortunately very plentiful in close proximity to these ore deposits.

In the great arid and semi-arid mining regions of the West the most common of the few plants and shrubs native to the region are the varieties of sage brush known as mountain sage, pasture sage, wormwood sage, etc.; also, in certain regions, greasewood and other shrubs of a similar nature. It is, therefore, to sage brush that my attention has been directed in the search for flotative agents for concentrating Western ores.

It appears that the various varieties of wild sage were first investigated by certain members of the Department of Agriculture, with regard to the possibility of producing from them by steam distillation essential oils suitable for pharmaceutical use. Steam-distilled oil was prepared by Rabak[1] from the variety *artemisia frigida* in 1905, and during the summers of 1907 and 1908 larger quantities of oil were prepared from specimens of this plant, collected in South Dakota. In 1912, an essential oil prepared by the steam distillation of *Ramona stachyoides* (black sage) from southern California was also reported by Rabak. This oil was said to contain 40 per cent. of camphor. In 1914, Charles E. Burke and Charles C. Scalione[2] gave an account of an investigation of an essential oil which had been prepared from the same shrub by the steam distillation of several hundred pounds of leaves and twigs collected from brush growing near Riverside, Cal. The yield of oil in this case was 0.9 per cent. of the weight of the material used. This yield was somewhat higher than that reported by the Department of Agriculture. This is perhaps accounted for by the fact that the brush was collected later in the season. This oil is reported as having the following composition:

	Per Cent.
Pinene	6
Cineol	30
Dipentine, terpinene, etc.	25
Thujone	8
Camphor	25
Resinous material	5

[1] Frank Rabak: Wild Volatile-oil Plants and Their Economic Importance, *Bulletin* No. 235, p. 22, *Bureau of Plant Industry.*

[2] Charles E. Burke and Charles C. Scalione: Investigations on Oil of Black Sage, *Journal of Industrial and Engineering Chemistry*, vol. 6, p. 804 (1914).

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Camphor was separated from a portion of the oil, thus demonstratin the rather interesting possibility of black sage as a source of borneo camphor. Upon request, Mr. Scalione furnished me with a sma sample of the original oil. This oil is clear, with a very slight yellowis tinge, and has an agreeable odor. In fact, so far as appearance and gener behavior goes, although the chemical composition is somewhat differen

FIG. 1.—ARTEMISIA TRIDENTATA.

it very much resembles steam-distilled eucalyptus oil from the Australi variety, *amygdalina*. This oil is a good frothing agent, and it yield quite satisfactory results upon lead and zinc ores in a qualitative w although the amount available did not permit of thorough investigati The idea of investigating sage brush and greasewood as possible sour of flotative agents was conceived early in January, but it was not u early in March that the first 100 lb. of sage brush was forwarded to

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through the courtesy of G. B. Lantz. This lot was collected near Goldfield, Nev., and proved to be the variety *Artemisia tridentata.* "In the Western Arid Transition zone the flora consists largely of the true sage brush, Artemisia tridentata,"[3] therefore this would be the variety available in the greatest abundance near to the mines of this region (see Fig. 1).

An apparatus for destructive distillation, capable of treating 30 lb. of brush at a charge, was constructed, and two charges of the brush were distilled (see Fig. 2). The products which first came over consisted of acid liquor resembling the crude pyroligneous acid obtained from wood distillation, a black oil or tar, and inflammable gas. Finally these products ceased to come over, but, upon raising the temperature, a consider-

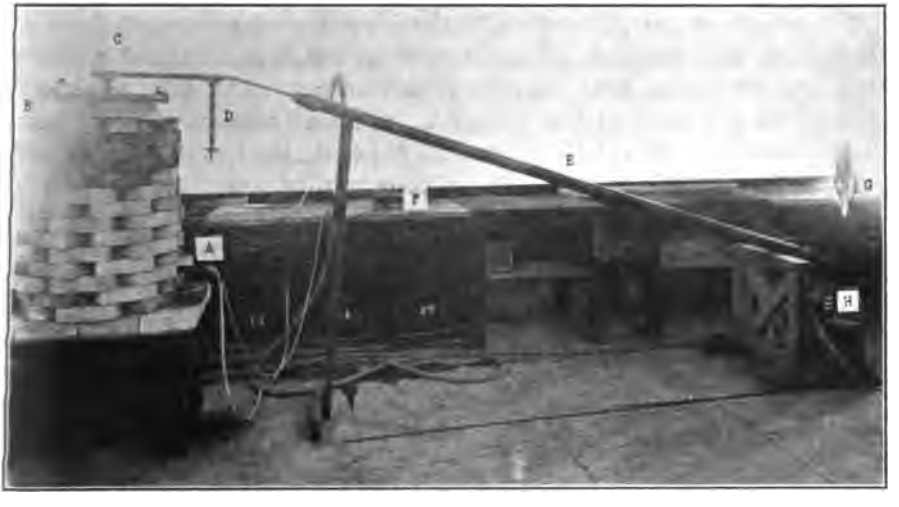

FIG. 2.—EXPERIMENTAL APPARATUS USED FOR THE DESTRUCTIVE DISTILLATION OF SAGE BRUSH. A, burners; B, retort; C, cold junction; D, tar tap; E, condenser; F, galvanometer; G, gas; H, distillate.

able amount of gas was given off and a rather thick reddish-brown liquor, having an alkaline reaction, began to come over, and with it was a small amount of tar similar to that which came over with the acid liquor. The brown liquor had the characteristic fishy odor peculiar to the amines. There was also at times a distinct ammoniacal odor. The acid and alkaline liquors were kept separate, while the tar from both was combined. These three products were first tried qualitatively in the flotation of finely ground samples of various minerals such as galena, cinnabar, pyrite, etc.

The acid liquor behaved very much as does ordinary pyroligneous acid. The alkaline liquor was a good frothing agent, but the froth

[3] *Encyclopædia Britannica*, Eleventh Edition, vol. 27, p. 634.

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carried up little mineral. The tar which came over with the acid liqu proved to be a splendid flotative agent.

Following this, quantitative tests were made upon a number of di erent ores, employing the tar produced from this lot of sage as a flotati agent. These results, a number of which are given below, were in gene very satisfactory.

A sample of zinc ore from the Butte Superior mine, Butte, Mon containing 22.39 per cent. of zinc, when tested in a Janney laborato machine employing a solution of 0.25 per cent. of sulphuric acid, ga 97.0 per cent. extraction of the zinc. The first concentrate contain 53.8 per cent. of zinc, the second 48.9 per cent., the third 40.5 per cen and the fourth 18.1 per cent. The oil consumption was at the rate 0.4 lb. per ton of ore.

A sample of mercury ore from the New Almaden mine, Californ containing 0.26 per cent. of mercury, when tested in a Janney machi employing a 0.2 per cent. solution of sodium carbonate, gave an extr tion of 90 per cent. of the mercury. The first concentrate contain 3.6 per cent. of mercury, the second 2.5 per cent., the third 1.55 per cen and the fourth 0.95 per cent. The oil consumption was at the rate of 1 per ton of ore.

A sample of lead ore from the Coeur d'Alene region, containing per cent. of lead, when tested in a Janney machine employing a 0. per cent. solution of sodium carbonate, gave an extraction of 92.2 p cent. of the lead. The calculated lead content of the total concentr was 37.3 per cent. The oil consumption was at the rate of 0.67 lb. p ton of ore.

A sample of silver-gold ore from the Ophir mine, Virginia City, Ne assaying 0.46 oz. gold and 7.4 oz. silver per ton, when tested in a Jann machine employing a 0.1 per cent. lime or sodium carbonate soluti gave an extraction of approximately 90 per cent. of the silver and per cent. of the gold. The first concentrate assayed gold 5.3 oz. a silver 198 oz. per ton; the second, gold 3.75 oz. and silver 72.9 oz. per t and the third, gold 1.32 oz. and silver 35.3 oz. per ton. The oil consun tion was at the rate of approximately 0.6 lb. per ton.

The oil consumption when employing sage tar appears to be less th with most of the other oils experimented with in treating the same o and the extraction was in general better. Since it is the experience many that large-scale operation requires less oil than is indicated by sm scale tests, it is reasonable to suppose that the oil consumption in tre ing the ores cited would be materially lessened when working upon operating scale, and that possibly the extraction could be bettered. think the latter is particularly true of the Ophir ore.

Later in March, another lot of brush of the same variety was collec by Mr. Lantz from the same locality, more fully in leaf than that collec

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earlier in the season. Distillations were run on this lot, keeping the various products separate, so that they could be properly measured and weighed.

Temperature measurements were made at regular intervals by means of a thermocouple, and as a result there was better control of the heating than in former tests. The yield from the last of these tests, which was the most carefully conducted, and was therefore the most representative, was as follows:

	Pounds	Per Cent.
Weight of sage brush used	30.	
Acid liquor	8.4	28.00
Tar with acid liquor	1.1	3.66
Alkaline liquor	0.66	2.20
Tar with alkaline liquor	0.09	0.30
Charcoal	10.00	33.33
Gas (by difference)	9.7	32.51

The retort was slowly heated for 1 hr. before liquid began to condense. The temperature at the center of the charge at the end of this time, as indicated by the thermocouple, was 60° C.; the temperature at the sides of the charge was probably somewhat higher.

In the next period of 6 hr., during which the acid liquor and most of the tar came over, the temperature rose from 60° to 275° C.; the rise above 100° C. taking place during the last hour and a half.

The alkaline liquor and the last of the tar came over in the last period of 3 hr., during which the temperature rose from 275° to 611° C.

It is reasonable to assume that a yield of about 4 per cent. of the tar oil can be realized if the sage is collected at the proper season and the distillation carried on by the best methods. Then there is the acid liquor, which in certain cases could be used directly in flotation; or it might prove profitable to recover the alcohol, acetic acid, the dissolved tar oil, etc., which it contains. I have not had an opportunity to investigate the alkaline liquor thoroughly, but it would seem to present many interesting possibilities. Among other things, the first lot was found to contain 2.09 per cent., and the second lot 2.18 per cent. of nitrogen.[4] I suspect that this liquor may also contain phenolic bodies which would be useful in flotation when separated from the other constituents. The charcoal is fine, but it should be possible to utilize it for fuel in heating the retorts. One peculiarity is the high percentage of ash which it contains (10.5 per cent. in the one sample analyzed). This may be in part due to dust upon the sage brush, although the brush was chopped fine before distillation, and it would seem that a good deal of the dust would be shaken from it during this operation. If the charcoal were burned, as previously suggested, the alkaline ash might serve instead of lime or sodium carbonate in cases where flotation in alkaline solution was practiced.

[4] Analysis by Professor R. E. Swain.

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The inflammable gas would, of course, be burned under the retorts The proportion of the heat necessary for carrying on the operation which could be realized in this way is problematical; however, it is reasonable to assume that a considerable part of that required for destructive distillation could be produced by burning the gas and charcoal. Moreover another interesting possibility in connection with the heating of the retorts is the utilization of the waste heat from various metallurgical operations.

Through the courtesy of Charles Scalione, several ounces of steam distilled essential oil was prepared from the tip ends of a portion of the last lot of sage brush. Approximately 1 hr. was required for distilling each charge of 20 lb. The yield amounted to 0.43 per cent. of the whole plant. This oil had a greenish-yellow color when first distilled, but became yellow upon standing. It has the characteristic penetrating odor of sage, and a very decided tendency to creep up the sides of the glass containing vessel. This oil appears to have distinctly different properties from the essential oil resulting from the steam distillation of the black sage. If gives promising results with some ores, but in my opinion it is not nearly so good as the tar oil resulting from the destructive distillation of the same shrub. In fact, under ordinary circumstances, I do not think that the steam-distilled oil can be given serious consideration since the mines that would be most benefited are generally located where fuel is high, and the yield of oil is rather small, probably less than 1 per cent. In addition, there are no other valuable products, unless the tannin extract resulting during steam distillation should have a market value or a demand should arise for "sage tea."[5]

Time has not been available for making analyses and a study of all the products resulting from the destructive distillation of sage. However, sufficient work has been done to show at least that the light tar is an efficient flotative agent for a considerable number of ores, as indicated by the tests cited.

The idea of utilizing sage brush in metallurgy is by no means new, as is shown by the following quotation,[6] referring to early development of the Washœ process, and the wild riot of experimentation accompanying it:

"The native sage brush, which everywhere covered the hills, being the bitterest, most unsavory, and nauseating shrub to be found in any part of the world, it was not long before a genius in charge of a mill conceived the idea of making a tea of this and putting it into his pans. Soon, the wonders performed by the sage-brush process, as it was called, were being heralded through the land."

[5] Patents are pending covering the use of the various sage products in flotation.

[6] Dan De Quille (William Wright): *The Big Bonanza*, pp. 138, 140, American Publishing Co., Hartford, Conn., 1877.

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The Antecedent Mineral Discovery Requirement

BY E. D. GARDNER, M. E., MISSOULA, MONT.

(Arizona meeting, September, 1916)

APPARENTLY the widespread agitation for the codification of our mining laws has had its effect, and it is quite possible that Congress will take up the question during this present session. The greatest objection to our present statutes pertaining to metalliferous deposits seems to be directed against the law of the apex. Those who are not in favor of changing this law are very far in the minority and they apparently realize that it is useless to urge its retention.

Next in importance to the repeal of the law of the apex, seems to be a widespread advocacy for the repeal of the law requiring discovery of mineral before a valid location of a mining claim can be made. To a lesser extent there appears to be a desire to have the law changed so as to allow patent for claims without mineral discoveries.

The purpose of this article is to review the arguments in favor of the proposed change of the discovery requirement, to show some of the evils that may be expected to result therefrom, and to consider whether a system may be devised to correct the undesirable features of the present law in this respect and at the same time not expose the mining fraternity and the general public to the evils which would result from the repeal of this fundamental requirement.

Under the law as at present framed, the validity of a mining location is dependent upon the fact of discovery of mineral within the boundaries of the land claimed.

"* * * but no location of a mining claim shall be made until the discovery of the vein or lode within the limits of the claim located" (Sec. 2,320, U. S. Rev. Stat.).

All authorities agree that discovery is the source of the miners' title. Lindley, in his valuable work on Mines (3d Ed., Sec. 335), lays down the rule in part as follows:

"Discovery is the initial fact. Without that no right can be acquired. * * * Such discovery must precede the location, or be in advance of intervening rights. The proof of recording and marking a claim will not authorize the court to presume a discovery."

As is well stated by the U. S. Supreme Court in Erhard v. Boaro (113 U. S. 536):

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"A mere posting of a notice on a ridge of rocks cropping out of the earth or on other ground, that the poster has located thereon a mining claim, without any discovery or knowledge on his part of the existence of metal there, or in its immediate vicinity, would be justly treated as a mere speculative proceeding, and would not of itself initiate any right. There must be something beyond a mere guess on the part of the miner to authorize him to make a location which will exclude others from the ground, such as the discovery of the presence of the precious minerals in it, or in such proximity to it as to justify a reasonable belief in their existence."

Accordingly, if the miner has made a discovery and otherwise conformed to the requirements of the law, his possession of the claim located is exclusive, against all the world, even, in all probability, against the owner of the land, the Government, except on areas which Congress by special legislation has set apart and defined for paramount purposes, such as National Parks. He has an interest in the land which descends to his heirs, which he can dispose of by will, by deed, or other contract, and all that is required of him after location, until patent is secured from the Government, is to keep his claim alive by performance of the annual assessment work thereon.

In cases where two private locators claim the same ground, the one first making a discovery of mineral would probably get the area, irrespective of the relative time of beginning work or of posting notices. However, such cases are extremely rare, considering the total number of claims located each year. Strictly speaking, a prospector cannot make a valid location before he has discovered mineral, but he has the rights of possession, except against the United States; and, while there is a conflict in the decisions on this point, in many cases the rule appears to be that as long as he is working the ground and is in possession the law will protect him, and when a discovery is made his location is good.

In any attempt to remedy defects in the present system of Federal mining laws, the underlying intent and purpose of existing legislation on the subject should not be lost from view. As was well said by the Assistant Secretary of the Interior in Cataract Gold Mining Co. *et al.* (43 L. D., 248):

"The intent of the general mining laws was to encourage and promote the development of the mining resources of the United States."

Proposed amendments of existing law, not based on this broad policy but having in view the exploitation of the public domain in the interests of mere individuals without corresponding benefit to the people as a whole, should meet the condemnation of every honest and right-thinking man. Keeping in view, therefore, this beneficent purpose of the present law, let us consider whether the proposed elimination of the antecedent-discovery requirement from the law will operate to give greater effect to such purpose.

It is contended that, if the requirement of an antecedent discovery

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of mineral within the lines of the claim located were dispensed with, the result would be to hasten the development of the mineral resources of the public domain and so increase the wealth of the nation; furthermore, that the present discovery requirement is an unnecessary hardship in a great many cases, and should be dispensed with in justice to those who are honestly endeavoring to develop the mineral resources of the public domain.

It is urged that, if the antecedent-discovery requirement were abolished, the one first staking out a piece of ground would be protected in his possession of it even if a later locator on the same ground were the first to discover mineral.

This raises the question, to whom should the desired reward (viz., exclusive occupation of the ground) be given? Shall it be to him who is diligent in staking out his claim, or to him who is not merely diligent in that respect but, not content with this, pursues diligence to the point of discovering the hidden mineral wealth? Which of the two is more deserving of reward? Which of the two is bending his efforts toward the consummation of the policy of the law? Certainly the one discovering mineral has performed a very essential act in the development of a mine.

In some mining districts where outcrops are few, extensive and expensive development work is necessary before valuable mineral is found. Many long and costly exploratory tunnels have been driven and shafts sunk to develop tracts of land on which occur no surface indications of veins or mineral but which were favorably situated in mineral belts. Justification for prosecution of development work under such conditions depends to a great extent on a careful study of mineral showing on adjacent lands. The successful prosecution of such exploratory work is dependent on the raising of sufficient capital for the enterprise. The raising of capital, a difficult undertaking in connection with almost any project so essentially speculative in character as that of exploring for precious metals, is made more difficult if the promoters cannot show a fee-simple title to the property to be developed. Hence, the abolishment of the discovery requirement would have the effect of overcoming this one obstacle to the development of our mineral resources.

When valuable orebodies are found on unpatented claims, there is always the menace of litigation to prove the ownership of the ground. After a strike has been made, it often happens that abandoned locations, which at one time covered the land, are resurrected or the boundary lines of prior adjoining claims are moved in such a manner as to take in the more valuable ground. The indefinite manner of describing claims in location notices, and the great number of old illegible mining corners usually scattered over the landscape in mining districts, increase the difficulty of successfully defeating such contests. The richer the ore-

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bodies found in these cases, the stronger the efforts by contesting clai
ants to obtain the ground.

The danger of contests on unpatented ground after a strike has be
made will be augmented if valid locations can be made without a discove
of mineral. Contests initiated by the locators of abandoned prior clai
would be more likely to succeed if these claims had had a legal existenc

It may be well to say here, however, that it must not be taken f
granted that all contests are not made in good faith. Locators of adjoi
ing ground quite often do not know the position of each other's lines, a
conflicts occur, therefore, when mineral survey is made for one or bo
of the claims. A contestant is usually honest in his belief in his ownersh
of the disputed area. Private contests against the issuance of pate
for mining claims are being continually initiated by claimants for all
part of the ground.

Sometimes a locator tries to get part of another's prior location
getting his conflicting claim patented. I know of instances whe
valuable ground which was being held under location has been patent
by a subsequent locator before the owner of the prior claim knew wh
was going on.

If we shut our eyes to actual conditions and assumed that all locato
of mining claims act in perfect good faith in staking out ground wi
intent to develop it for its mineral contents, probably there would be
necessity for the antecedent-discovery requirement. But mining loc
tions, even now, are made to cover a multitude of frauds against t
public-land laws. Title to valuable water-power sites, immense quan
ties of timber and valuable town sites have been secured under the mini
laws. How much greater and more frequent would be these frauds
even a perfunctory discovery of mineral were not required to validate t
locations. By far the greater number of mining claims desired for pu
poses other than mining have had no discovery of mineral.

Care should be taken that the interests of all the people are not sac
ficed in our consideration of the welfare of the prospectors. The d
covery requirement is designed, and operates, as a check on the dispo
of lands of the people under the mining laws for purposes foreign to t
intent of the law, viz., the speedy and bona fide development of t
mineral resources of the public domain.

If the validity of the location is to be dependent merely on taki
possession of the ground without the antecedent discovery of mine
what is to prevent the staking of claims near pay ground with intent
hold them for speculation and hold-up purposes, and not for bona f
development of mineral value, to a much greater extent than at prese
What is to prevent irresponsible individuals from staking claims wh
can be held indefinitely by perfunctory assessment work, thereby ty
up large areas of valuable ground for the purpose of levying tribute

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those desiring to obtain the ground for mining or other purposes? Surely, to abolish the antecedent-discovery requirement is to open the door to fraud and deceit incalculable, and will tend to block actual development of large areas for many years to come.

The abandonment of the mineral-discovery requirement would naturally facilitate the patenting of mining claims; but, however desirable it may be in some cases to expedite the granting of patents, I believe easy patenting, on the whole, will not benefit the mining industry, but will retard it.

Most mining men know of many promising claims and prospects for which patents have been obtained which have been, to all intents and purposes, abandoned. A large proportion of the applications for patent comes from owners who are tired of doing the annual assessment work but still desire to hold the ground. Of such cases, only a small number of claims are ever worked again.

Nearly 10,000 mineral surveys for groups of claims, from one to thirty or more each, have been made in the State of Montana alone, and this number does not include placer claims that have been taken up by legal subdivisions. While patent does not necessarily follow a mineral survey, it does so in nearly all cases, and of the patented claims only a small number have been worked since final certificates were issued. Outside of the Butte district, I will venture to say that less than 5 per cent. of the patented claims in Montana are now being worked, when metals are higher than at any time in the last 50 years.

In such active districts as Butte and the Coeur d'Alenes, where large capital is required in developing properties, the patenting of claims, on the whole, does not discourage new operations, but perhaps helps them. In cases, however, where one dominant company patents the whole surrounding territory, independent operators who would perhaps be willing to take a chance on developing some of this ground, if open, are thereby kept out.

The patenting of the ground seriously handicaps the chances of small camps and new districts for becoming important producers. Numerous cases can be pointed out where the patenting of the claims has seriously retarded the development of camps, or altogether stopped it.

The majority of those who have written about the desirability of amending the present mining laws have done so from the standpoint of the mining operator or prospector. None of the writers on the subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some

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people would appear to believe. Individual claims on it are allow only by virtue of the laws of Congress, which prescribe the performa of certain definite requirements, and it does not belong to the first co solely on his assertion of a claim to any part of it.

Many abuses have been perpetrated by unscrupulous locators land under the mining laws. Mining claims have been, and are n located to control springs, water-holes, range privileges, power reservoir sites, town sites, rights-of-way, summer-residence sites, tim and agricultural land, natural curiosities, and other surface valu While, at the present time, land more valuable for other purposes t for mining cannot be patented under the mining laws unless min has been found and the required expenditure made, in the past, bef the Government inspected the ground before issuing patent, valua areas were patented when the ground was in no respect mineral. At present time, in some parts of the country, public business is seriou interfered with by unscrupulous locators of ground under the min laws. While the Government is now embarrassed by these hold locations (the mining industry at large has no idea how great the num is), the main expense is the examination of such claims and the de caused by them. If, by amending the present laws, valid locations be made anywhere on Government land without mineral discovery, Government or other activity on public land will be safe from extort and blackmail. Conditions are bad enough now. For instance, in so localities, as soon as a body of Government timber is advertised for sale, timbered area and ground over which the logs are to be moved to mar is promptly located as mining claims, or old claims are revived, and he demands made for the use of the surface of the ground. If, by l claims of this sort can be made valid, it will give unscrupulous individu a monopoly of the surface and prevent legitimate business over la areas. The owner of a single valid mining claim crossing a right-of-w of a logging road could exact a sum, for the privilege of crossing ground, large enough to take all the profit from the logging operatio As the cost of operation is directly related to what an operator can for the timber, the Government has a direct interest in all such ca It has been repeatedly held by the courts that a valid location h against the Government, even if abandoned for years. I know of ar from which the Government was contemplating selling the timber on which there were as high as 100 abandoned mining locations. such cases, if these locations had been valid without a mineral discov the locators on reasserting their rights to the ground as soon as they fo the timber was valuable could have kept the Government from sel it. There is hardly a stream course or canyon in many parts of the W that has not been at some time plastered with locations.

Even if valid locations could be made without a mineral discov

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only on lands classified as mineral, it would still affect the Government, for there are bodies of white pine, which are advertised for sale from time to time, on lands in National Forests which have been classified as mineral but not covered by mining locations.

It has been said that one must actually make a discovery of mineral on land within National Forests before he is allowed possession or allowed to prospect it. Such is not the case, however. A prospector or miner is in no way disturbed by Government agents in the enjoyment of his rights within National Forests. No examination is made of any ground located for mining purposes unless it interferes with the administration of the Forest. A claim may interfere with the administration of the Forest when it conflicts with areas occupied by individuals under what are known as Special Use Permits; when it is included within a tract from which the Government has sold the timber; or where the claim is so located as to control rights of way over which it is necessary to transport Forest products. Unless the claims are actually interfering with the administration of the Forests, no examinations are undertaken until applications for patent are made. The proportion of mining claims examined prior to application for patent is very small, probably less than 1 per cent. of the total number located. There are thousands of mineral locations within the National Forests of which no examination has been made by the Forest Service, and in all probability never will be except in cases where application for patent is made.

Mining locations within the National Forests which, after an examination by a competent man, are shown to be clearly invalid, may be disregarded as far as the surface is concerned, but no action is taken to dispossess the locator of the ground and no objection whatever is made to his development of the mineral possibilities. In fact, the Assistant Secretary of the Interior has decided (Nichols-Smith case, unreported) that the Department has no jurisdiction over locations and has authority to cancel a claim only after an application for patent has been made. This decision, however, is now before the Secretary for consideration on motion for the exercise by him of his supervisory authority.

The fact that the Government has placed a large part of its domain within the National Forests shows that this land is valuable to the Government for the benefit of the people at large. This fact alone is an argument from the people's standpoint that the requirements for patenting Government land by individuals, at least within the Forests, should not be made easier.

I believe that, while sometimes the law requiring a discovery before a valid location can be made has worked hardships on prospectors, the wrong that could be worked under a law not requiring a discovery would be far greater.

Under the general mining laws, lands are not required to be chiefly

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valuable for mineral before they can be entered. The Supreme Cour of the United States has held in U. S. *vs.* Iron Silver Mining Co. (12 U. S. 673) that the fact that land may incidentally possess advantage other than its valuable mineral deposits will not preclude its disposition under the mining laws.

An act of Congress of Aug. 4, 1892 (24 Stat., 348) provided:

> "That any person authorized to enter lands under the mining laws of the Unite States may enter lands that are *chiefly* valuable for building stone under the provi sions of the law in relation to placer mineral claims."

The point of difference between this act and the general mining law appli cable to mineral deposits is that the Act of 1892 requires the land to b chiefly valuable for building stone.

Power or reservoir sites situated on land containing no evidence o mineral deposits have been entered as stone placers, but the application for patent have been rejected by the General Land Office. If vali mineral locations could be made on such lands without a mineral dis covery, the land could be patented as mining claims.

If patent were allowed for Government ground as mining claims with out a mineral discovery there would be very few, if any, power project free for development at the present time, for the ground would be validl held or patented as mining claims. Most of the undeveloped availabl power sites on public lands have been included in power withdrawals o National Forests. The ground is, therefore, not subject to entry excep as mineral locations, and the only way to get a patent to such ground i under the mining laws. Nearly all power projects in the Northwest where there is any possibility of early development, are on ground covere by mining locations. In some cases the mining claims are valid unde the present law, but the majority have no mineral discovery. If the law were changed concerning mineral discoveries, there would be little nev development of hydro-electric power without paying tribute to the kin of prospectors who had the foresight to locate mining claims covering th ground. Also Federal control, due to an interest in the site, could b circumvented by first obtaining patent as mining claims to all the lan affected. In the Northwest there is a power project capable of develop ing 20,500 hp. One-half of the land affected by the project is within National Forest, and therefore the Government under the present pro cedure exercises control over the development of the site. The powe would be developed under a Special Use Permit issued by the Gover ment, which requires certain conditions to be complied with. All the public land has been located as placer mining claims, and if the claim were valid the Government under the present law would have no contr whatever over the development of the power. In 1913, a permit wa issued to a company to develop the power, but for some reason it was n

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able to go ahead with the project. This company, however, acquired the mining locations, and if the claims could be patented the owners could prevent anyone else from developing the power site. Flour gold is found in gravel on all of these claims, but nowhere in paying quantities.

Another project with which I am familiar is capable of developing 331,000 hp. by using storage, or 178,000 hp. by utilizing only the minimum flow of the river. The lower 12 miles of this project, which is capable of developing 260,000 hp. by storage, or 140,000 hp. without, is entirely covered by mining locations. The area is within a mining district and would probably be classified as mineral, but only a few of the locations have mineral discoveries. One side of the river is within a power withdrawal, and the other within a National Forest. In this case, as in the other, a permit was granted to develop the project, but it has not been done. The permittee has acquired the mining claims and he controls the site in so far as his mining claims are valid. If all these mining claims were valid without mineral discoveries, they could be patented and the Government would lose control of the project.

On March 1, 1906, Windfield Doern and two others located the Eagle placer mining claims, alleged to be chiefly valuable for building stone, on the Stanislaus National Forest in California (41 L.D. 655–659). On Oct. 25, 1907, the claim was conveyed to the Stanislaus Electric Power Co., who filed application for patent on April 5, 1908. The entry was contested by the Government. Stone was used from the claim in the construction of a dam on the claim, from which water was conveyed to a plant 15 miles below, where power was generated. It was proposed to build a second plant on the claim and bring water in a conduit from above. The patent was denied by the General Land Office, as it was shown at a hearing that the stone had no value outside of the construction of the dam, and for that purpose had no special value. The land, however, had value as a power site.

On Jan. 2, 1907, H. V. Gates made application for patent under Act of Aug. 4, 1892, for the Excelsior placer, area 140 acres, in northern California. The patent was protested by the Northern California Power Co. At a hearing, it was disclosed that the ground contained a deposit of basalt that had only a local use for building stone, such as the construction of a power dam, and was not as suitable for that purpose as other common rock. The claim covered a reservoir and tunnel site, and the claimant had a permit from the Forest Service to use a part of the claim for power purposes. The General Land Office held that the claim was more valuable for power than for building stone, and the entry was canceled.

On a river in a western State is situated a power site capable of developing 2,490 hp. The dam site is covered by an unpatented lode claim, and above the falls are situated four patented lode claims along the river in

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a row. Each of the four patented claims contains a valid mineral discovery, but no ore. Before proceeding to patent, the claimants received $5,000 from a power company for the privilege of flooding their ground.

Perhaps more fradulent mining locations have been made to acquire valuable timber than for any other purpose on non-mineral land. If the mineral-discovery requirement is repealed, large timber steals which have been prevented in the past will be successful. Many applications for timbered mining claims have been canceled after the Government has shown that no mineral discoveries have been made. It would not be possible to defeat such cases if the proposed change of the law is made.

Timber on patented mining claims can be disposed of by the owner as he sees fit. Timber on valid unpatented mining claims on proven mineral land outside of National Forests can be cut for certain described purposes as approved by Congress by Act of June 3, 1878 (20 Stat., 88). Timber, however, on an unpatented mining claim within the National Forests cannot be cut for any purpose other than the development of the claim, except by Special Use Permit or purchase from the Government.

It has been a common practice in some localities to locate mining claims on non-mineral timbered areas to give color of title to the ground while cutting the timber. Numerous suits have been instituted by the Government to recover the value of timber cut from such invalid claims.

In the case of Anderson *vs.* United States (152 Fed., 89 to 91), Anderson, Baker and Sandlin located as placer claims 1,200 acres of non-mineral Government land in the Boise land district, Idaho, from which was cut timber worth $17,751.79. Suit was brought by the Government to recover the value of the timber, and it was successful.

In the case of Powers *vs.* United States (119 Fed., 562), the Government brought suit to recover the value of 668,000 ft. b. m. of timber, worth $7,241, which was cut from non-mineral Government ground. The ground was covered by mining locations made by a third party.

About 10 years ago, a company applied for patent for 3,636.99 acres of Government land as placer claims. The magnitude of the case caused examination to be made by the General Land Office, and it was found that on one claim there was a small worked-out placer deposit of gold. There was no possibility of gold occurring in paying quantities on the rest of the group, although fine colors could be found almost anywhere in the country. This land contained an average stand of 15,000 ft. b. m. of white pine and other varieties to the acre, or about a total of 60,000,000 ft. b. m. on the group. The claims were continuous, but excluded and almost surrounded a small area from which the timber had been burned. The entry was protested by the Government and at a hearing held to determine the validity of the group, the claimants defaulted, thereby admitting the truth of the charges. The entry was finally canceled on Jan. 28, 1910. At this time, unpatented fraudulent placer claims were

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being held in this district which contained a stand of between 200 and 300 million feet of timber. During the summer of 1910, forest fires that destroyed the timber swept over this country, after which all of the timber-mining locations were abandoned.

In 1913, application was made for two groups of claims in Idaho. No mineral had been found on four of the claims of the group which contained a heavy stand of white pine worth $3,800. After a hearing, the entry was held for cancellation by the Commissioner of the General Land Office.

In 1912, application was made for two claims in the Coeur d'Alene district, Idaho. This entry was also held for cancellation by the Commissioner of the General Land Office, after a hearing. These claims were completely surrounded by patented placer claims, but were situated on a small mountain. The ground was covered by a stand of white pine having a stumpage value of $3,478. One of the claimants was interested in a saw mill situated near the claims.

Along in 1906, a mining company (long since defunct) did considerable development work on lode locations in western Montana, and located a large number of claims. The company failed to find any ore, and went out of existence. The first year after the company abandoned the ground, it was located by a man who did no more work on it. He relocated some of the claims for several years, but when no one else showed any desire to obtain the ground, he also abandoned it. In 1914, a lumber company secured timber on the river above these claims. After the lumber company cleared its right-of-way, the mineral locator came along, saw the situation and immediately reasserted his right to some of the ground that controlled the right-of-way. He demanded an excessive sum for the privilege of crossing his ground. His demands were refused and logging operations shut down for about a year. An examination of the mining claims was made by a mining engineer and it was found that the claims were invalid because no mineral had been discovered upon them. After a year's delay, the company constructed the logging road and commenced operations without any further trouble and without paying tribute to the locator. If these locations were valid without mineral discoveries, the logging operations could have been held up indefinitely unless the excessive demands of the locator were met. The claims are in a general mineral belt.

If valid locations of lode claims can be made in the future without a mineral discovery, many town sites are going to be held as mining claims. The following case is now before the Land Department for settlement. In 1909, a lode claim was located in such a manner as to take in all of the level portion of the canyon at the division point of a new railroad in Idaho. On June 27, 1913, application for patent was made. The section in which the claim is situated has been classified by the United

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States Geological Survey as non-mineral, and the land is in the Northe Pacific grant. The railroad has a station and some dwelling houses o the ground and part of the claim is also claimed by the heirs of a hom steader. If the mineral locator can prove a valid discovery on the clain he is entitled to the ground. The railroad began operating about th time the claim was located. Practically a whole town, with a popul tion of several hundred, is on the mining claim, and there is no doul that the ground is more valuable for town-site purposes than for an thing else. A large diabase dike, in which can be found traces of coppe runs through the claim, but there is no indication that paying ore w ever be found upon it. Sufficient development work has been performe If no discovery of mineral is going to be required, many similar cases ma be expected in the future.

The repeal of the mineral-discovery requirement would open th doors for blackmail of all irrigation companies situated like the one the following case, even when there is not a trace of mineral on th land. In 1910, an application for patent was made for a placer situate in a small park in western Montana. Previous to the mineral applic tion, an application was made to the Secretary of the Interior for the us of the park for a storage-reservoir site, for which purpose the ground well adapted. It was proposed to impound water during the winte months and high-water periods, to be used for reclaiming about 3,00 acres of arid land. The reservoir permit was approved by the Secretar but construction has not been begun on account of the placer clain The Commissioner of the General Land Office decided that sufficie mineral to constitute a discovery had not been made upon the claim an it is now before the Secretary of the Interior on appeal. The park an old lake bottom and contains sand, clay and gravel to an undetermine depth. This material all contains minute quantities of gold, but in place, as far as developed, in paying quantities. No serious effort h been made to mine the gravel, and for the last 20 years the ground h been used by the claimant as a ranch. If it is finally decided that th claim contains sufficient mineral for a discovery, the mineral claima will be legally entitled to the land, and, before the irrigation system ca be commenced, the land will either have to be obtained from him purchase or by condemnation proceedings.

Throughout the country, attempts have been made to obtain posse sion of numerous springs, mineral or otherwise, by taking up the grou as mining claims. On Aug. 26, 1886, the Margaret Mining Co. ma application for the Gray Eagle Lode mining claim (11 L. D., p. 563) King County, Washington. This claim embraced some springs known the Hot Springs of Green River which were supposed to have gre medicinal value. The entry was canceled by the Land Office because was shown at a hearing that the land had no value for mineral but w

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used for a health resort, and that the mining company was formed to acquire title to the Hot Springs.

By the Act of Jan. 31, 1901 (31 Stat., 745) Congress provided that "all unoccupied public lands of the United States containing salt springs or deposits of salt in any form and chiefly valuable therefor are hereby declared to be subject to location and purchase under the provisions of the law relating to placer mining claims." Under this Act many mineral springs having no value whatever for salt, but having supposedly medicinal properties, have been located as placer claims.

On March 20, 1905, Henry Lovely made application for patent for the Lovely placer claim in the Juneau land district in Alaska. It was shown at a hearing that the ground was used as a health resort and water from two springs was conducted to bath houses, the cost of this being applied as mining improvements. It was also shown that the ground was not mineral or saline in character. The claim was canceled Feb. 13, 1907, by the General Land Office (25 L. D., 426).

By decision of April 14, 1913, the Secretary of the Interior held that the Salt Creek placer claims were invalid (21 L. D., 745). These claims were located in February, 1911, under the Act of Jan. 31, 1901, so as to cover a group of hot springs whose waters carried in solution common salt, Epsom salts and Glaubers salts and other chemical substances. As compared to sea water, the waters of these springs were shown to contain about one-eighth as much common salt. It was held by the Secretary of the Interior, from facts determined at a hearing, that the ground had no value for the manufacture of common salt, but possessed distinct curative properties.

In central Montana is situated a mineral spring known as the Appollonaris. The ground at the small side gulch on which the spring is situated was located as a stone placer and the adjoining ground on the main creek as gold placers. No discovery of gold had been made on the gold placers, and the limestone on the stone placer was not valuable on account of its poor quality and the distance from any market or railroad. This spring has a local reputation for medicinal properties and Special Use Permits for erecting bottling works and hotel buildings on the area was desired by several people. The development of the spring was held up for 5 years, but finally the men holding the mining claims decided that they could not patent the ground and abandoned it. The spring is now being developed.

The abrogation of the mineral-discovery requirement would greatly facilitate such fraudulent attempts to obtain ground desired for purposes other than mining.

At numerous points desirable for summer residences on the shore of Lake Pend Oreille, mining locations have been posted, and in many cases there has been a pretense of doing the annual assessment work. In a

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few cases, application for patent has been made. On some of these claims sufficient mineral has been found to constitute a mineral discovery, but in the majority of cases no mineral has been found.

The Frances Placer, consisting of 20 acres on a bar on the Clearwater River in central Idaho, was applied for under the mining laws. The claimant had farmed the land for at least 10 years previous to making application. The mineral entry was contested and canceled in 1912, and the land was later taken as a homestead.

Mining claims have been patented, and others are being held as locations, on favorably located scenic points along the rim of the Grand Canyon of the Colorado. Two lode claims also have been patented which cross the Bright Angel Trail. Tourists who have gone down this trail and paid their dollar each to the owner of the lode claims can realize his farsightedness in finding his mine just in this particular location. It is of interest to note that no mineral commercially valuable has been found on any of these claims.

On the public domain there are numerous conflicts of mineral claims with homesteads. In all contests, the burden of proof is on the homesteader, as he has to prove the land non-mineral in character. If valid locations can be made without the necessity of a mineral discovery, the homesteader's lot in mountain valleys is going to be still harder to bear, and the good Lord knoweth it is bad enough now.

If the mineral-discovery requirements for the location of mining claims is abrogated, no form of activity can be safely undertaken on unpatented land, and it is difficult to anticipate the many abuses that can be perpetrated.

The uncertainty of ultimate possession of unpatented land in mineral belts could be remedied, I believe, by making provision by law for clearing title to the ground against all except the United States at any time, as is now done at the time of application for patent. If, after development, mineral is found, the ground could then be patented.

Perhaps the method now used in connection with homesteads on the public domain could be adapted to the making of mineral entries. In the case of homesteads on surveyed land, an applicant, at the time of going on the land, makes a filing in the U. S. Land Office, corresponding to the posting of a location notice on a mining claim, and, if no adverse claims are proven and the land is non-mineral, his entry is allowed. This excludes all other private claims to the ground, and, after he has complied with certain prescribed regulations, patent is issued to the homesteader.

If all men were honest, or this country were a Utopia, it would be perfectly feasible to repeal the law requiring the discovery of mineral, but, unfortunately, such is not the case. Under the present law some operators may have been unjustly obliged to defend their rights, but it has been found that laws made to restrain the unjust usually cause suffer-

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ing to some of the just. The repeal of this law would afford relief to some who are acting in good faith, but at the same time it would let down the bars, allowing an unlimited amount of fraud, and fostering blackmail. That the wrong application, if possible, would be made of any new mining laws, should be expected from the many frauds that have been perpetrated under the present laws. By far the greater number of the mining claims that have been desired for purposes other than mining have had no discoveries of mineral. I believe that by suitable laws it is quite possible to ameliorate the one condition without aggravating the other.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Possibility of Deep Sand Oil and Gas in the Appalachian Geo-Syncline of West Virginia

BY DAVID B. REGER,* A. B., B. S., C. E., MORGANTOWN, W. VA.

(Arizona Meeting, September, 1916)

Introduction

THE exhaustion of oil and gas in the United States is proceeding at a rapid pace. This is especially true in fields where the light oils that furnish the most fuel for internal-combustion engines are found, leading to a late estimate by Arnold that the petroleum resources are 36 per cent. exhausted.[1] The further remark by Dr. David T. Day at the New York meeting of the Institute, in February, 1916, that the operators of the country are now scarcely able to supply the ever-increasing demand for gasoline, leads the writer to believe that a determined effort will soon be made to secure deeper producing horizons in those regions that now furnish high-grade oil, and in which pump stations and pipe lines offer convenient means for the economical handling of new production. A study of Arnold's table shows further that the States that produce paraffin oil exclusively are 45 per cent. exhausted, this figure being much higher than the average of paraffin and asphalt States altogether. The most extensive of all the paraffin-oil-producing areas is the Appalachian Field, extending with the Appalachian Geo-Syncline from western New York southwestward through western Pennsylvania, western West Virginia, Ohio, Kentucky, and Tennessee, where oil of high gravity is always secured. Of these States, Pennsylvania and West Virginia rank highest in the quality of their product. In West Virginia, the oil sands dip to a lower level than in any other locality in the Appalachian Geo-Syncline, the Pittsburgh Coal of the Pennsylvanian being only 50 ft. above sea level along the Nineveh Syncline at Wileyville, Wetzel County.[2] The very significant fact that the sands of the Mississippian and Upper Devonian Measures furnished the richest oil pools of the State along this

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[1] Ralph Arnold: Petroleum Resources of the United States, *Economic Geology*, vol. 10, p. 710 (December, 1915).

[2] Ray V. Hennen: Marshall-Wetzel-Tyler Report, *West Virginia Geological Survey*, p. 66. 1909.

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deep basin, most of them being non-water-bearing, should give pertinen to such information as may be obtained regarding deeper sands in th region, in which there may be a possible duplication of conditions in t more shallow sands.

Authority for Data

Information has been secured primarily from the published repo of the Geological Surveys of West Virginia, Ohio, Pennsylvania, a Kentucky, supplemented by various special reports and maps of t United States Geological Survey. The southwestward extension of t Chestnut Ridge (Warfield, Campton) Anticline through Kentucky h been made on the authority of a recent publication by James H. Gardne Several important well records used have never been previously publishe having been lately secured by the West Virginia Geological Survey. D credit has been given in the table of well records to the operators w furnished the various logs, in all cases where this information is availab Special acknowledgment is made to Dr. I. C. White, State Geologist West Virginia, for many valuable suggestions from his extensive kno edge of the region under discussion.

Description of the Appalachian Geo-Syncline

The Appalachian Geo-Syncline, lying west of the Appalachian Mou tain System, and being roughly parallel to it, extends in a south-sout westward direction from New York State through Pennsylvania, We Virginia, Kentucky, Tennessee, and Alabama. As partially shown Plate 1 (in those regions where detailed structure maps are available) has a course almost straight through Pennsylvania from Brookville, Kittanning, Pittsburgh, and Washington, to the southwestern corner the State where it enters West Virginia and extends southwestward acr Wetzel, Tyler, and into Ritchie, where the Burning Springs Anticli cuts diagonally across it shifting the line of its axis nearly 20 miles farth west, to a point near Parkersburg. From this point its course is rough southwest through Wood, Jackson, Mason, Cabell, and Wayne Counti to the Kentucky State line 7 miles south of Catlettsburg. Its cou through Kentucky is apparently interrupted by the Chestnut Rid (Campton) Anticline which cuts across it, separating it from its southe extension.

Geology of the Appalachian Basin

The surface geology along the axis of the Appalachian Basin is pri cipally that of the Coal Measures, but at the eastern rim these rocks ri

[3] James H. Gardner: A Stratigraphic Disturbance through the Ohio Vall Running from the Appalachian Plateau in Pennsylvania to the Ozark Mountains Missouri. *Bulletin of the Geological Society of America*, vol. 26, pp. 477–483 (Decem 4, 1915).

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rapidly toward the Allegheny and other mountain ranges of the Appalachian System until they disappear above the tops and the rocks of the

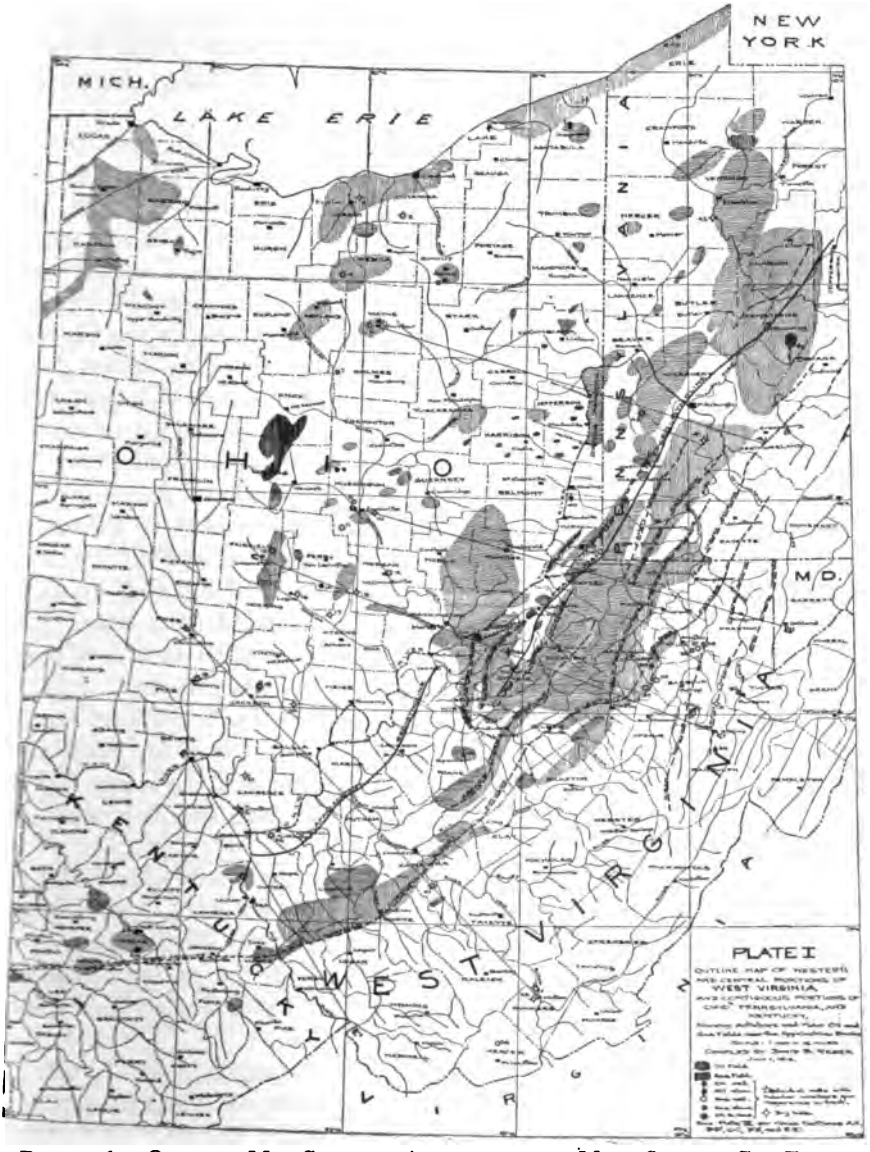

PLATE. 1.—OUTLINE MAP SHOWING ANTICLINES AND MAIN OIL AND GAS FIELDS NEAR THE APPALACHIAN BASIN.

Devonian, Silurian and lower formations come to the surface, the pitch in this region being abrupt and irregular. On the western side of the axis the measures rise gradually toward the Cincinnati Anticline, showing

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none of the spectacular structural disturbances visible on the east. Northeastward from West Virginia, through Pennsylvania, the strata rise gradually along the axis of the basin, but the Pennsylvanian still forms the surface rocks at the New York State line, near Olean. Southwestward from West Virginia through Kentucky, the rocks rise along the axis, but very gradually, so that the Pennsylvanian still makes the surface deposits at the Tennessee line.

Plate 2 shows a columnar section made for the locality of Marion, Harrison, Wetzel, and western Monongahela Counties, West Virginia, where the rocks of the Appalachian Basin dip to their lowest level, and where the oil reservoirs of the State are the most prolific. In this region the surface rocks are principally the transitional Permo-Carboniferous type represented by the Dunkard Series. Throughout this region numerous drillings are available that extend to the base of the Catskill Series, leaving no uncertainty as to the character of the sediments down to this level. Below the Catskill, the section is based on the deep wells drilled in Pennsylvania (Nos. 41 and 42 on Plate 1) and on numerous drillings in Morgan, Muskingum, and other Counties in Ohio where the Devonian shales have less thickness than in southern Pennsylvania and on the fairly well-known thickness and character of these Devonian and Ordovician beds in the Alleghany Mountain region of West Virginia farther east where these measures outcrop. This surrounding information makes the portion of the section below the Catskill more than hypothetical, the probability being that the intervals shown are fairly good while some of the individual formations may be locally absent.

Dunkard Series.—The Dunkard Series, composed of 1,150 ft. of sandy red, and variegated shales, gray sandstones, with a few thin coals and limestones, is barren of oil and gas in this region and the lower 300 to 400 ft. of the series found below the main drainage channels must be drilled through without result.

Monongahela Series.—The Monongahela Series, composed of 400 ft. of gray sandstones, gray shales, limestones and coal beds, is also practically barren of oil and gas, and is encountered by the drill everywhere along the deeper part of the basin. At its base is the great Pittsburgh Coal bed used widely as a key rock by the drillers of Pennsylvania and West Virginia.

Conemaugh Series.—The Conemaugh Series, comprising 500 to 600 ft. of gray sandstones, red and sandy shales and a few coals and limestones contains the Little Dunkard sand in the lower portion and the Big Dunkard at its base, both of which have produced oil at several localities in the State.

Allegheny Series.—The Allegheny Series, 250 ft. thick and composed of gray sandstones, gray shales and having a few coals and thin limestones contains the Burning Springs and Gas sands, that produce oil in great quantity along the Burning Springs Anticline.

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COLUMNAR SECTION FOR CENTER OF WEST VIRGINIA OIL FIELDS

(Marion and Surrounding Counties)

Vertical Scale: 9/16″ = 1000 Ft.

		Name of Series	Columnar Section	Thickness Feet	Total Feet	Description
Permo-Carboniferous		Dunkard		1150	1150	Variegated Shales and Gray Sandstones with a few Thin Coal Beds
Carboniferous	Pennsylvanian	Monongahela (Pittsburgh Coal at Base)		400	1550	Gray Sandstones, Gray Shales, Limestones, and Coal Beds
		Conemaugh		600	2150	Gray or Brown Sandstones, Gray and Red Shales, and Coal Beds
		Allegheny		250	2400	Gray Sandstones, Gray Shales, and Coal Beds
		Pottsville (Salt Sands of W.Va.)		300	2700	Gray Sandstones and Shales, with a few Coal Beds
	Mississippian	Mauch Chunk (Contains Marion Sand of W.Va.)		250	2950	Red Shales with a few Thin Sandstones
		Greenbrier ("Big Lime" of W.Va.)		100	3050	Limestone
		Pocono (Big Injun at Top; Berea Sand at Base)		500	3550	Gray Sandstones and Gray Shales
Devonian		Catskill (Gordon Group of Oil Sands)		800	4350	Brown Sandstones and Red Shales
		Chemung (No Productive Sands in W.Va.)		1500(?)	5850	Olive Brown, Shales with Sandstone Lentils
		Portage (No Productive Sands in W.Va.)		800(?)	6650	Gray Shales with Sandstone Lentils
		Hamilton (No Productive Sands in W.Va.)		700(?)	7350	Brown, Shales with Sandstone Lentils
		Marcellus or Romney (Gas in Ohio and Kentucky)		300(?)	7650	Brown or Black Bituminous Shales with Sandstone Lentils
		Corniferous Limestone (Bagdad Sand of Kentucky)		50(?)	7700	Dark Flinty Limestone
		Oriskany		150(?)	7850	Gray Sandstone
Silurian		Helderberg Salina, and Niagara ("Big Lime" of Ohio)		800(?)	8650	Limestone
		Clinton		200(?)	8850	Variegated Shales
		Medina White Sandstone ("Clinton" Oil Sand of Southern Ohio)		50(?)	8900	White Sandstone
		Medina Shales		500(?)	9400	Red Shales and Thin Sandstones
Ordovician		Martinsburg or Cincinnati Shale (Contains Hudson Sand of Kentucky)		500(?)	9900	Gray Shales with Sandstone Lentils
		Utica		300(?)	10200	Black Shales with Sandstone Lentils
		Trenton and other Limestones (Oil and Gas Horizon of Northern Ohio)		1200(?)	11400	Limestones

PLATE 2.—COLUMNAR SECTION FOR CENTER OF WEST VIRGINIA OIL FIELDS.

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Pottsville Series.—The Pottsville Series, composed of gray conglon eratic sandstones, gray shales, and having numerous coal beds, varies i thickness from about 200 ft. in the northern panhandle to nearly 4,0(ft. at the southern end next to Virginia. Along the axis of the App lachian Basin, its thickness at the Pennsylvania line is about 250 f and this increases gradually to nearly 1,000 ft. at the Kentucky lin Oil is often found in several of its sands, the upper one being called th Second Cow Run, and the lower ones being usually termed the Firs Second, and Third Salt sands. Copious flows of salt water are usual found in one or more of these horizons, giving rise to their names.

Mauch Chunk Series.—The Mauch Chunk Series, composed main of red and green shales, with a few flaggy green fine-grained sandston and impure limestones, varies in thickness from only a few feet in son of the western counties to about 3,500 ft. in Summers and Mercer. Alo the oil belt it is usually about 250 ft. thick, with the Maxton oil and g sand near the base.

Greenbrier Limestone.—The Greenbrier limestone, usually a solid ma of hard limestone, varies in thickness from 50 ft. in the northern pa handle to 1,500 ft. in Greenbrier and other southern Counties. Along th Appalachian Basin, it averages about 100 ft. and is known as th "Big Lime," and is the most dependable key rock below the Pittsbur Coal, as the intervals from it to the lower sands are fairly constant ov wide areas. It is seldom a producing horizon but makes some oil and g from sandy streaks in Lincoln, Wayne, and Mingo Counties.

Pocono Series.—The Pocono Series, composed of gray sandstones a gray shales, is usually about 500 ft. thick in all the counties along t Appalachian Basin, and contains the Keener, Big Injun, Squaw, We and Berea sands. Of these the Big Injun is the most prolific oil and g horizon of the State, possibly deriving its oil from the Big Lime whi often rests directly upon it; the Berea sand is a close second to the Inju being of widespread occurrence and often productive over wide areas.

Catskill Series.—The Catskill Series, composed of red or brown san stones, separated by red shales, varies from 500 to 800 ft. in thickness a contains the Gantz, Fifty-foot, Thirty-foot, Gordon Stray, Gordo Fourth, Fifth or McDonald, Sixth or Bayard, and Seventh or Elizabe sands. Of these the Gordon is the most valuable producer, ranking wi the Berea in the quantity of oil but not having such a wide occurren The sands of this series are the lowest that have been commercially pr ductive in the State. Southward toward Kentucky, the series thins o completely, being scarcely represented at all south of the Great Kanaw River.

Chemung, Portage and Hamilton Series.—The Chemung and Porta composed of green, gray and brown shales respectively, with lenticu sandstones, the Chemung being probably about 1,500 ft., the Porta

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about 800, and the Hamilton about 700, contain no productive sands in West Virginia, although the Speechley sand of the Chemung has made shows of oil and gas in various wells. In northern Pennsylvania, there are several productive horizons in this group. Westward in Ohio and southward toward Kentucky, these three series thin out almost completely, leaving the Berea sand only 700 to 1,000 ft. above the Corniferous limestone. These shales form a great barrier that has prevented any exploitation of the lower Devonian sands in northern West Virginia.

Marcellus Shale.—The Marcellus shale, known as the Ohio black shale in the State of that name and as the Chattanooga shale in Tennessee, and often called simply the Devonian black shale, is probably about 300 ft. thick along the Appalachian Basin in West Virginia. It is a black, bituminous horizon, and contains a few lenticular gas sands in Ohio and Kentucky.

Corniferous Limestone.—The Corniferous limestone, probably not more than 50 ft. thick in West Virginia, is the Ragland oil sand of Menefee and Morgan Counties, Kentucky. It has been penetrated in West Virginia by only three wells near the Kentucky line (Nos. 26, 26A, and 27 on Plate 1), and in two or three wells in Randolph and Tucker Counties in the Alleghany Mountains, the detailed records of only one of which (No. 32) has been obtained. It is probable that the most eastern oil counties of the State do not have this horizon, as its presence is not noted in the Alleghany Mountains where it should crop, and where it should be easily recognized from its well-known flints and fossils.

Oriskany Sandstone.—The Oriskany sandstone, gray in color and probably 150 ft. thick, and a producer of oil and gas in southern Indiana and central New York, was found in the Geary Well (No. 42 on Plate 1) in Pennsylvania, where it was 155 ft. thick, and in the Slaughter Creek Well (No. 27 on Plate 1) in Kanawha County, West Virginia, where it was 15 ft., and in the Parsons Well (No. 32 on Plate 1), where it was 80 ft. West of the basin it is not noted in the deep wells of Ohio but on the eastern side of the Alleghany Mountains it is the well-known glass sand of Morgan County, West Virginia, and it seems likely that it is represented along the basin in West Virginia.

Helderberg, Salina, and Niagara.—The Helderberg, Salina, and Niagara limestones, often comprising a solid limestone mass 800 ft. or more thick, and known in Ohio drilling parlance as the "Big Lime," has been penetrated in West Virginia only by the Bartram (No. 26A) and Slaughter Creek (No. 27) Wells, and was found in the Geary (No. 42) Well in Pennsylvania. Since this great limestone mass is found in these three wells and is universally present in Ohio, and is also found in much the same great development in West Virginia east of the Alleghanies, it seems safe to conclude that it is present along the Appalachian Basin with the same or greater thickness. It is possibly the parent reser-

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voir from which much of the White Medina sandstone, or "Clinton Sand" oil of Ohio is derived, so that its presence along the basin is of special interest.

Clinton Shales.—The Clinton variegated shales are usually found in Ohio between the "Big Lime" and the "Clinton" sand, varying in thickness from 50 to 200 ft., and probably occur in West Virginia along the Basin.

Medina White Sandstone.—The "Clinton" oil sand of southern Ohio, believed by I. C. White and others to represent the Medina white sandstone, is the great oil-and gas-producing stratum of Hocking, Fairfield, Perry, and other southern Ohio Counties, being usually 10 to 50 ft. thick. No drill has ever reached it in West Virginia, but it is present in the Alleghany Mountains east of the oil fields and probably occurs along the Basin.

Medina Shales.—The Medina variegated shales, with probable lenticular sandstone horizons, noted in many deep wells in southern Ohio, may have a thickness of 500 ft. along the Appalachian Basin.

Martinsburg Shale.—The Martinsburg shale, often called the Cincinnati or Hudson, is found in Ohio, Kentucky, and eastern West Virginia, and may therefore be taken for granted along the Appalachian Basin in West Virginia. According to Hoeing,[4] this series furnishes several oil and gas sands in Wolfe, Morgan and other Kentucky Counties. Its thickness is possibly 500 to 1,000 ft. along the Appalachian Basin.

Utica Shale.—The Utica formation, consisting of black shales with probable sandstone lentils, may have a thickness of 300 ft. along the basin, although the information regarding it in surrounding States is vague and uncertain.

Trenton Limestone.—The Trenton group, composed of several thick limestones, often coalesced into one thick mass 1,000 to 1,200 ft. thick, is the great oil and gas-producing zone of north-central Ohio and also is productive in Indiana, central New York, and in some parts of Kentucky. The fact that it occurs in these surrounding States and also crops in great thickness in West Virginia east of the Alleghanies seems to leave little doubt that it occurs also, with its usual great thickness, along the Appalachian Basin.

Cross-Sections

Based on the evidence of the numbered wells shown on Plate 1, many of which were very deep tests, and on many hundreds of oil and gas wells, the records of which are published in the West Virginia State Reports, and also on the outcropping rocks around the rim of the Appalachian Basin, five cross-sections, the geographic locations of which are shown

[4] J. B. Hoeing: Oil and Gas Sands of Kentucky, *Bulletin No.* 1, *Kentucky Geological Survey*, pp. 39–40 (1905).

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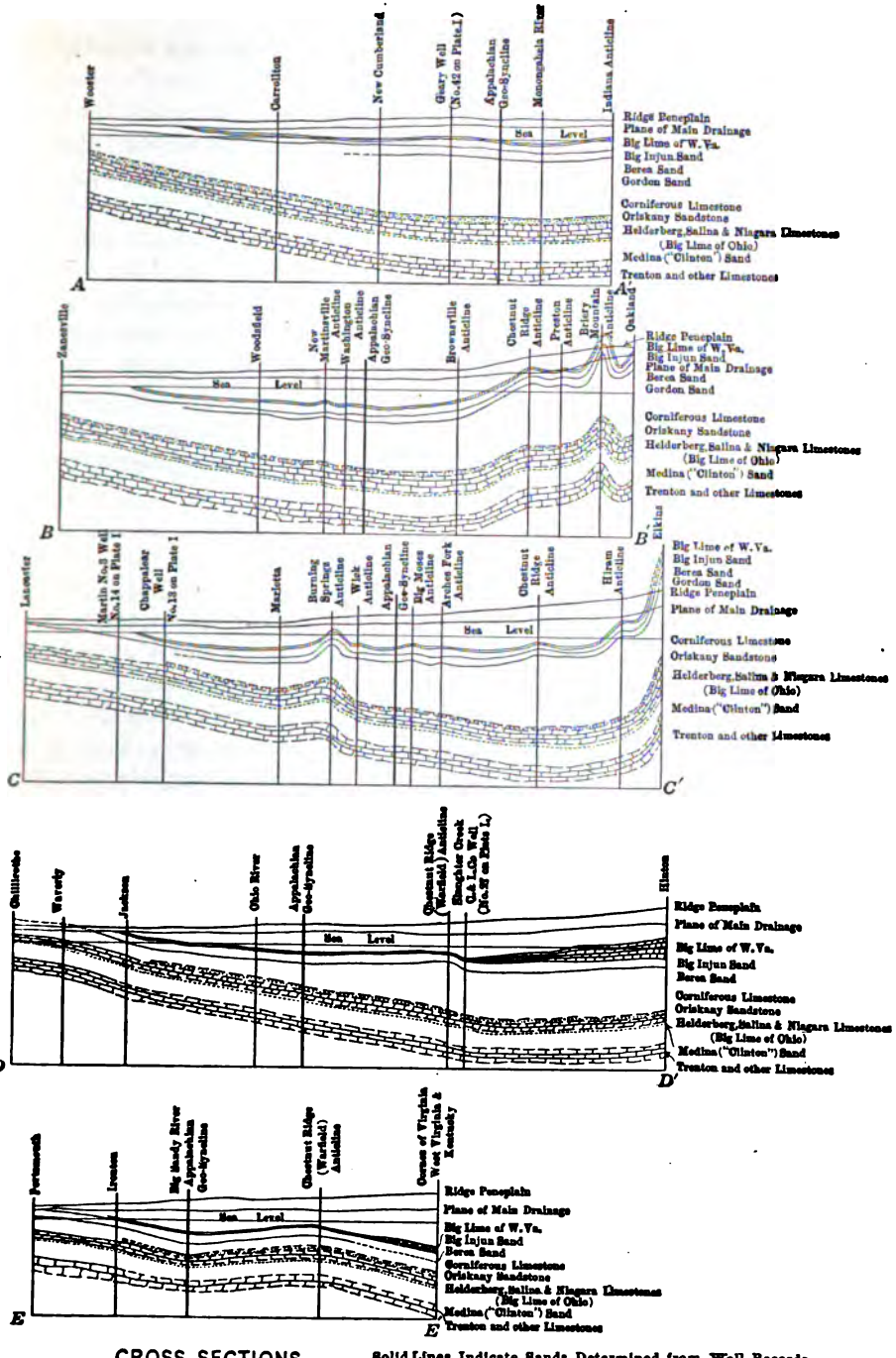

CROSS SECTIONS

Horizontal Scale: 11/32 = 16 Miles
Vertical Scale: 11/32 = 4000 Ft.

Solid Lines Indicate Sands Determined from Well Records
Dotted Lines Indicate Supposed Position of Deep Sands as Determined from a few Deep Wells and from Intervals East and West of Appalachian Basin

PLATE 3.—CROSS-SECTIONS AS INDICATED ON PLATE 1.

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on Plate 1, have been plotted across the basin in West Virginia (Plate 3). The scale of the plates does not permit the representation in detail of the surface hills and valleys but a solid line shows the ancient peneplain surface of the ridges, and another shows the plane of the main drainage channels with reference to sea level, from which the approximate interval from this latter plane to the principal sands may be determined, as all estimates for drilling must start from this drainage plane. Formations shown with solid lines are plotted from actual records but the gaps are filled in with dotted lines to represent the probable positions of the various deep sands and limestones as outlined above. Of the more shallow sands, only two or three are plotted for use as key rocks. These cross-sections reveal the very interesting probability that the well-known gradual eastward thickening of the Chemung, Portage, and Hamilton shales shifts the axis of the Appalachian Basin of the lower limestones and sands many miles farther east than the position it occupies when referred to the formations above these shales.

Deep-Well History

The accompanying table, containing the abbreviated records of a large number of wells, has been compiled from the various sources indicated to show the different deep tests that have been made in West Virginia and in neighboring States to find oil in the sands below the Catskill Series. In Ohio, where hundreds of productive wells have been drilled to the "Clinton" and Trenton, only those have been selected that lie nearest to the Appalachian Basin, being located along a fairly straight line reaching from Lake Erie to the Ohio River, and in Kentucky only a few wells are listed to show the typical formations and intervals. Each well is given a tabular number that corresponds to the same number placed beside the well symbol on Plate 1.

Of the wells tabulated for West Virginia, only four (Nos. 26, 26A, 27 and 32) were drilled as deep as the Corniferous limestone, and only three of these went through the Oriskany sandstone into the Niagara and Helderberg limestone. No well in the State has ever been drilled into the Medina White sandstone ("Clinton" oil sand of Ohio) or into the Trenton limestone. All the wells drilled to the Oriskany sandstone have been somewhat distant from the main oil fields of the State and are therefore indecisive as tests of its oil and gas possibilities. Briefly, then, the deep sands not yet tested in the main West Virginia oil fields are the Corniferous limestone ("Ragland" of Kentucky), Oriskany, Medina white sandstone ("Clinton" of Ohio), and the Trenton limestone, with a possibility of another sand horizon in the Martinsburg (Hudson) group, which produces oil in Kentucky.

The well most typical of the formation and conditions likely to be

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encountered in drilling for these deep sands in West Virginia is the Geary deep well (No. 42 on Plate 1) drilled near McDonald, Pa., and only about 50 miles due north of the main West Virginia fields. This well holds the unique distinction of being the deepest hole ever drilled in the United States, and its record down into the Ohio "Big Lime" was published by I. C. White.[5] Judging from the record of this well and that of the Slaughter Creek (No. 27) deep well, it seems likely that troublesome salt water may be expected in the Oriskany and Salina beds along the minor synclines that lie parallel to the main Appalachian Basin, and that oil and gas will more probably be found along the axes of the anticlines or at no great distance down their slopes.

This theory that oil may be found principally along the anticlines is contrary to the experience of operators who have exploited the sands of the Catskill Series in the same region, and who have learned through the drilling of many thousands of wells that these latter sands are usually not water-bearing and the oil is therefore almost universally found along the synclines.

Whether deep-sand oil and gas exist in commercial quantity along the Appalachian Basin in West Virginia will be known only through the ultimate test of the drill, but the fact that these sands are widely productive in Ohio, Kentucky, Indiana, and New York, in regions where structural relief is similar to that in West Virginia, and the further strong probability, and almost certainty, that these porous reservoirs actually exist in close proximity to the great limestones that, along with the Devonian black shales, are evidently their main source of oil in other localities, indicates that they will be found commercially productive along the Basin. The fact that the bituminous shales are undoubtedly much thicker in West Virginia than in Ohio, Kentucky and Tennessee indicates an even more hopeful possibility of oil.

If the doubt as to the presence of oil be dismissed as unreasonable, the problem of the operator resolves itself into the double question; first, as to the location of the pools of oil and gas; and, second, as to the mechanical possibility of drilling wells 5,000 to 10,000 ft. deep. As outlined above, the most favorable localities seem to be along or near the axes of the anticlines where the danger of finding salt water would be minimized and where the gas, if any, should most certainly collect. The search for oil would naturally be made on either side of the anticlines by making locations at increasing distances from the axes.

That drilling may be carried on successfully to the great depths indicated by the figures above seems open to little serious doubt. In 1860, when the search for oil and gas may be said to have been actively begun in the Appalachian Basin, few, if any, wells had been drilled to a

[5] I. C. White: Notes on a Very Deep Well Near McDonald, Pennsylvania, *Bulletin of the Geological Society of America*, vol. 24, p. 275 (June 10, 1913).

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Summarized Record of Deep

No. on Plate 1	Farm Name and Number and Oil Company	County and State	Elevation above Tide, Feet	Big Lime of W. Va.		Berea Sand		Corniferous Limestone		Oriskany Sandstone	
				Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet
1	J. A. Giddings lot, Jefferson O. & G. Co.	Ashtabula, Ohio.	...	...	...	...	...	...	...	...	...
2	Garfield & Caine, East Ohio G. Co.	Cuyahoga, Ohio.	780	...	...	10	80	...	...	...	...
3	Citizens No 1.	Lorain, Ohio.	...	...	...	...	...	...	...	...	...
4	E. S. Albert No. 1.	Medina, Ohio.	920	...	...	386	10	...	...	...	...
5	Barberton Chem. Co. No. 1.	Summit, Ohio.	...	...	...	460	10	2,180	...	...	...
6	Wooster well, H. B. Odenkirk.	Wayne, Ohio.	860	...	...	465	30	...	...	...	...
7	Kaylor.	...	...	...	...	615	10	...	...	...	...
8	Fairall.	Muskingum, Ohio.	...	...	...	834	16	...	...	...	...
9	Dillon Falls well, Chicago-Zanesville O. & G.	Muskingum, Ohio.	...	...	...	895	12	...	...	...	...
10	George Handehsy, Zanesville G. & O. Co.	Muskingum, Ohio.	...	...	...	1,010	23	...	...	...	...
11	W. J. Roberts.	Muskingum, Ohio.	...	250	38	847	14	...	...	...	...
12	McConnellsville Fair Grounds, McConnellsville City.	Morgan, Ohio.	725	625	44	1,279	66	3,082	2	3,125	8
13	T. J. Chappalear, J. P. Fishel.	Morgan, Ohio.	750	...	...	1,150	20	...	...	...	...
14	Martin No. 3. E. S. Martin.	Perry, Ohio.	...	...	...	920	20	...	...	...	...
15	Crouse No. 1. B. S. Stretton.	Fairfield, Ohio.	...	...	...	...	...	...	...	...	...
16	Logan town, Logan town.	Hocking, Ohio.	748	...	...	683	45	...	...	...	...
17	Columbus Hocking C. & I. Columbus Hocking C. & I.	Athens, Ohio.	...	...	...	...	...	...	...	...	...
18	Waverly O. & Gas. Co. Waverly O. & Gas Co.	Pike, Ohio.	575	...	...	...	...	...	...	...	...
19	Buckeye Coal Co. No. 2. Buckeye Coal Co.	Jackson, Ohio.	...	...	...	640	20	...	...	...	...
20	At Vinton village.	Gallia, Ohio.	...	...	...	...	...	...	...	...	...
21	At Mt. Vernon village.	Lawrence, Ohio.	...	...	...	1,095	25	...	...	...	...
22	At Ironton town.	Lawrence, Ohio.	500	...	...	1,010	47	...	...	...	...
23	Burns well.	Morgan, Kentucky,	...	368	106	1,095	10	...	...	...	...

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Wells Shown on Plate 1

Helderberg, Salina and Niagara, "Big Lime of Ohio"		Medina ("Clinton") Sand		Trenton Limestone		Total Depth, Feet	Reference to Detailed Record	Remarks
Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet			
1,810	288	2,127	13	...	...	2,140	Ohio G. S. 4th series, *Bull.* 1, p. 303.	Oil well in "Clinton" sand.
1,210	1,455	2,680	...	...	...	2,770	Ohio G. S. 4th series, *Bull.* 12, p. 65.	Heavy flow of water and oil show in Niagara lime. Gas in "Clinton" sand.
440	850	...	...	2,645	...	....	Ohio G. S. 4th series, *Bull.* 12, p. 64.	No "Clinton" sand.
1,512	1,127	2,749	14	...	...	2,777	Ohio G. S. 4th series, *Bull.* 12, p. 62.	
2,180	350	2,500?	30	...	...	3,006	Ohio G. S. 4th series, *Bull.* 1, p. 296.	Gas show(?) in "Clinton" sand. Rock salt and lime in lower 300 ft. of hole.
1,830	1,085	3,104	31	...	...	3,140	Ohio G. S. 4th series, *Bull.* 12, p. 60.	1,440,000 cu. ft. of gas daily and oil show in "Clinton" sand. Heavy flow of brine at 2,055 ft.
1,725	960	2,851	28	...	...	2,957	Ohio G. S. 4th series, *Bull.* 12, p. 59.	Gas show in "Clinton" sand.
2,035	934	3,141	21	...	...	3,194	Ohio G. S. 4th series, *Bull.* 12, p. 37.	"Clinton" sand oil well.
2,120	1,275	3,543	21	...	...	3,567	Ohio G. S. 4th series, *Bull.* 12, p. 35.	Salt water 2,560 ft. and 3,205 ft. Shows of oil and gas in "Clinton" sand.
2,640	1,005	3,709	38	...	...	3,805	Ohio G. S. 4th series, *Bull.* 12, p. 36.	Dry hole.
2,195	970	3,296	...	...	...	3,314	.....	Gas in "Clinton" sand.
...	...	...	...	...	...	3,186	Ohio G. S. 4th series, *Bull.* 1, p. 145.	Gas in Berea sand.
2,700	900	3,935	9	...	...	3,947	Ohio G. S. 4th series, *Bull.* 12, p. 38.	No oil or gas; 1 bailer of water per hour in Niagara lime.
2,170	1,170	3,175	17	...	...	3,213	Ohio G. S. 4th series, *Bull.* 12, p. 23.	Oil well in "Clinton" sand.
1,378	...	2,180	5	3,470	...	3,569	Ohio G. S. 4th series, *Bull.* 12, p. 28.	No oil or gas noted in record, but was probably a "Clinton" gasser.
1,715	715	2,627	18	...	...	2,694	Ohio G. S. 4th series, *Bull.* 12, p. 39.	Dry hole.
...	...	...	...	...	...	....	Ohio G. S. 4th series, *Bull.* 12, p. 41.	Drilled to "Clinton" sand. No oil or gas.
485	490	...	...	1,960	808	3,335	Ohio G. S. 4th series, *Bull.* 12, p. 48	Some salt water and gas in Niagara lime; oil and gas show in Trenton.
660	1,454	2,255	8	...	...	2,315	Ohio G. S. 4th series, *Bull.* 12, p. 44.	4,000,000 cu. ft. of gas daily in "Clinton" sand.
...	...	...	...	...	...	3,200	Ohio G. S. 4th series, *Bull.* 12, p. 44.	Dry hole. No "Clinton" sand.
1,810	620	...	...	...	...	2,730	Ohio G. S. 4th series, *Bull.* 12, p. 50.	Dry hole. No "Clinton" sand.
1,827	584	...	...	3,440?	...	3,600	Ohio G. S. 4th series, *Bull.* 12, p. 49.	Dry hole. No "Clinton" sand.
...	...	1,408	94	...	...	1,508	Ky. G. S. *Bull.* 1, p. 159.	"Clinton" sand oil well.

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No. on Plate 1	Farm Name and Number and Oil Company	County and State	Elevation above Tide, Feet	Big Lime of W. Va.		Berea Sand		Corniferous Limestone		Oriskany Sandstone	
				Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet
24	Caney Creek well, G. M. Sullivan.	Morgan, Kentucky	...	453	69	1,035	24	1,370	...	...	...
25	Blaine Creek well.	Lawrence, Kentucky	...	...	...	1,660	2	2,592	−5	...	...
26	Central City well, T. H. Harvey.	Cabell, W. Va.	530	970	150	1,730	25	2,760	10	...	...
26 A	David Bartram No. 483, United Fuel Gas Co.	Wayne, W. Va.	595	1,220	80	1,875	20	2,760	410	...	...
27	Slaughter Creek C. & L. Co. W. S. Edwards Oil Co.	Kanawha, W. Va.	640	1,453	207	2,093	12	4,945	90	5,035	15
28	Griffith No. 1, Rich Creek O. & G. Co.	Mercer, W. Va.	2,090	...	...	...	...	...	...	...	...
29	B. T. Baker No. 1, Summers O. & G. Co.	Summers, W. Va.	1,586	1,020	1,057	2,600	25	...	...	...	...
30	P. C. Daniels No. 1, Tygarts Valley Oil Co.	Randolph W. Va.	1,990	...	...	...	...	...	...	...	...
31	Alonso W. Murphy No. 1. Mead *et al.*	Randolph, W. Va.	1,995	...	...	...	...	...	...	...	...
32	Parsons Pulp & Lumber Co. Parsons Pulp & Lumber Co.	Tucker, W. Va.	1,655	...	...	...	...	3,825	20	3,980	80
33	J. C. Benson No. 3,612 Hope Natural Gas Co.	Barbour, W. Va.	1,026	1,235	90	1,615	138	...	...	...	...
34	Porter Maxwell No. 3,869, Hope Natural Gas Co.	Harrison, W. Va.	1,050	1,531	...	...	...	...	...	...	...
35	M. J. Coplin No. 1, Tri-State Gas Co.	Harrison, W. Va.	1,030	1,288	77	1,711	13	.....	...	...	...
36	Absalom Knotts No. 1, Greensboro Gas Co.	Taylor, W. Va.	1,460	960	50	1,460	120	...	...	...	...
37	R. S. Hauser No. 1, Greensboro Gas Co.	Taylor, W. Va.	1,480	950	40	...	...	...	...	...	...
38	Russell McCartney No. 1, Greensboro Gas Co.	Taylor, W. Va.	1,580	1,055	35	1,500	90	...	...	...	...
38 A	T. E. Dye No. 1, Am. Hydroscope Co.	Ritchie, W. Va.	...	...	...	...	...	...	...	...	...
39	J. F. Dobbs No. 1, South Penn Oil Co.	Marshall, W. Va.	980	1,760	30	...	...	...	...	...	...
40	Boggs Run well, Wheeling Dev. Co.	Marshall, W. Va.	900	...	...	...	...	...	...	...	...
41	Wm. Bedell, Forest Oil Co.	Allegheny Penna.	...	1,050	50	1,515	50	...	...	...	...
42	R. A. Geary No. 1,770, Peoples Nat. Gas Co.	Washington, Penna.	1,050	953	29	1,610	12	6,008	37	6,045	155
	Witherop, Conway Bros.	Venango, Penna.	955	...	...	...	...	3,870	10	...	...

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Helderberg, Salina and Niagara, "Big Lime of Ohio"		Medina ("Clinton") Sand		Trenton Limestone		Total Depth, Feet	Reference to Detailed Record	Remarks
Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet			
...	...	1,408	...	...	...	2,021	Ky. G. S. *Bull.* 1, p. 72.	Oil and gas in "Clinton" and Caney Sands. Salt water in Clinton. Probably stopped just short of Trenton.
...	...	...	...	...	...	2,597	Ky. G. S. *Bull.* 1, p. 78.	Gas in Pottsville Conglomerate and Berea sand.
...	...	...	...	...	...	2,770	W. Va. G. S. Vol. I (A), p. 495.	Oil and gas in Berea sand; gas in bastard limestone at 2,485 ft.
2,760	410	...	...	...	...	3,130	Unpublished.	Dry hole; oil show in Weir sand at 1,790 ft.; Corniferous and Niagara recorded together.
5,050	545	...	...	...	...	5,595	W. Va. G. S. Kanawha Report, p. xix.	Squaw sand gas; water in Niagara Lime at 5,592 ft.
...	...	...	...	...	...	4,105	W. Va. G. S. Raleigh Report.	Still drilling; base of Pottsville, 105 ft.
...	...	...	...	...	...	2,664	W. Va. G. S. Raleigh Report.	Dry hole.
...	...	...	...	...	...	2,000?	W. Va. G. S. Barbour-Upshur-Randolph Report.	Starts near base of Chemung Series. Shows of oil and gas.
...	...	...	...	...	...	2,385	W. Va. G. S. Barbour-Upshur-Randolph Report.	Starts near base of Chemung Series. Dry hole.
...	...	...	...	...	...	4,250	Not published.	Starts near base of Chemung Series. Pockets of gas at 825 to 835 ft., 1,053 and 3,830 ft. Strong salt water at 3,990 ft. to 4,000 ft.
...	...	...	...	...	...	4,570	W. Va. G. S. Barbour-Upshur-Randolph Report.	Gas in Big Lime, Big Injun, Berea, and Kane(?) sands.
...	...	...	...	...	...	4,386	W. Va. G. S. Barbour-Upshur-Randolph Report.	
...	...	...	...	...	...	4,028	W. Va. G. S. Doddridge-Harrison Report, p. 120.	Little gas in lime and shells at 2,950 ft.
...	...	...	...	...	...	3,955	W. Va. G. S. Barbour-Upshur-Randolph Report.	Gas show at 1,210 ft.
...	...	...	...	...	...	4,437	W. Va. G. S. Barbour-Upshur-Randolph Report.	Light gas in Big Injun sand; Kane(?) sand, 4,108 to 4,162 ft.
...	...	...	...	...	...	3,506	W. Va. G. S. Barbour-Upshur-Randolph Report.	Gas show, 2,425 ft.
...	...	...	...	...	...	4,450	W. Va. G. S. Doddridge-Harrison Report, p. 284.	Flow of oil in Speechley sand at 3,125 ft.
...	...	...	...	...	...	3,411	W. Va. G. S. Marshall-Wetzel-Tyler Report, p. 390.	Gordon sand oil; Speechley sand, 3,344 to 3,376 ft. with some dark oil.
...	...	...	...	...	...	4,500	W. Va. G. S. Vol. I, p. 364.	Base of Big Injun at 1,570 ft.; show of oil at 2,955 ft.
...	...	...	...	...	...	5,575	W. Va. G. S. Vol. I (A), p. 104.	Speechley sand, 3,150 to 3,165 ft.; Bradford sand, 3,500 to 3,520 ft. with trace of oil; hole stopped in Marcellus shale.
...	...	...	...	...	...	6,299+	G. S. A., *Bull.* Vol. 24, p. 275, 1913.	Gas, 4,850 ft., 4,870, 5,900, 5,905, 5,910, 5,915, 6,060 ft. Not yet completed.
...	...	...	...	...	...	3,880	W. Va. G. S. Vol. I (A), p. 86.	Slight shows of oil and gas.

depth of 1,000 ft. In 1890, when development was at its height, 3,000 ft. was considered an unusual depth, but by 1910 several holes had been drilled more than 5,000 ft. and at the present time at least two wells in the world have been drilled more than 7,000 ft. The advance in drilling has been just as rapid as in any other line of human endeavor where mechanical ingenuity and skill must be used and in these qualifications the Scotch-American well driller stands second to none. To say that drilling cannot be continued to greater depths than the Geary and Slaughter Creek wells would call for a total cessation of the enterprising experimentation that has characterized oil-well drilling from its very beginning.

Suggestions for Drilling

A comparison of the cross-sections on Plate 3, in conjunction with the data exhibited on Plates 1 and 2, shows that ideal conditions may be found along several of the anticlines east and west of the main Appalachian Basin, among which may be mentioned the New Martinsville, Washington, Brownsville, Arches Fork, Big Moses, Chestnut Ridge (known as the Warfield in southern West Virginia), and last and best of all, the Burning Springs, which stretches in a north and south direction for more than 30 miles through the very heart of the oil fields and in a region where the Devonian shales are probably much thinner than at points farther east. A well drilled at the point where this anticline crosses Walker Creek, near Sandhill, 10 miles southward from St. Marys, and 5 miles north of Petroleum on the Baltimore and Ohio Railroad, and 5½ miles south of the point where cross-section *CC'* crosses the anticline, would have a tidal elevation of 920 ft. Being at the summit of the great Volcano dome, it would start at an approximate interval of 1,000 ft. below the Pittsburgh Coal and should find the Big Lime of West Virginia at about 100 ft., the Berea sand at about 600 ft., the Corniferous limestone at about 3,500 ft., the Oriskany sandstone at about 3,550 ft., the Medina white sandstone ("Clinton") at about 4,700 ft., and should reach the Trenton limestone at about 6,000 ft. The dome-shaped structure at this point would almost insure the absence of water in the sands, at the same time making the financial risk much less than at any other point in the well-developed oil fields of the State. Oil or gas secured at this point would immediately open a prospective field for 30 miles along the crest of this great anticline where conditions are similar all the way from St. Marys on the Ohio River to Burning Springs near the corner of Wirt, Roane and Calhoun Counties.

Should this region prove to be commercially productive from the lower sands, attention would naturally turn to the other anticlines mentioned, where drilling would be more expensive but the possibility of finding oil and gas reasonably good.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Petrography of the Mount Morgan Mine, Queensland.

BY W. E. GABY, B. S., M. A.,* BUTTE, MONT.

(Arizona Meeting, September, 1916)

INTRODUCTION

SINCE the time of their discovery, the genesis of the ores at Mount Morgan, and the nature of the changes which have affected the surrounding rocks, have been the subject of investigation and speculation by geologists and mining engineers. Developments at this mine, for a long period the greatest gold producer of the world, show that with depth the orebody grades into copper, of which metal it now furnishes a large output, exceeding in value the gold.

In earlier times, the purity of the gold and the first extremely heterogeneous character of the ore made it an interesting problem. Now, the probable depth of the copper enrichment, genesis, and true relations of the neighboring rocks furnish equally profitable material for investigation, and the present somewhat detailed description of these ores and rocks, from microscopic study, may prove of interest.

The mine occupies a highly silicified portion of a belt of quartz-porphyry, which is believed to be the oldest rock of the region. This belt of quartz-porphyry and allied rocks runs north and south between two areas of a later intrusive granite, as shown on the geologic map,[1] Fig. 1. Near the orebody the quartz-porphyry has been intruded by a dense basaltic mass called the "Old Basic." Cutting all three of these igneous masses are two sets of basaltic dikes, one running northwest-southeast, and the other, cross dikes, northeast-southwest. These dikes may be contemporaneous with, instead of later than, the ore. Their number has caused peculiar difficulties in extracting it, as the clay selvage of decomposed material affords no adhesion between dike and ore, causing great blocks to loosen and come down.

The investigation was based on specimens furnished by B. Magnus, former general manager of the mine, and notes on the areal geology by

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* Assistant Geologist, Anaconda Copper Mining Co.

[1] Reproduced by permission of the Australasian Institute of Mining Engineers from the plate accompanying the paper of J. M. Newman and G. F. Campbell Brown, *Transactions, Australasian Institute of Mining Engineers*, vol. 15, part 2 (1910).

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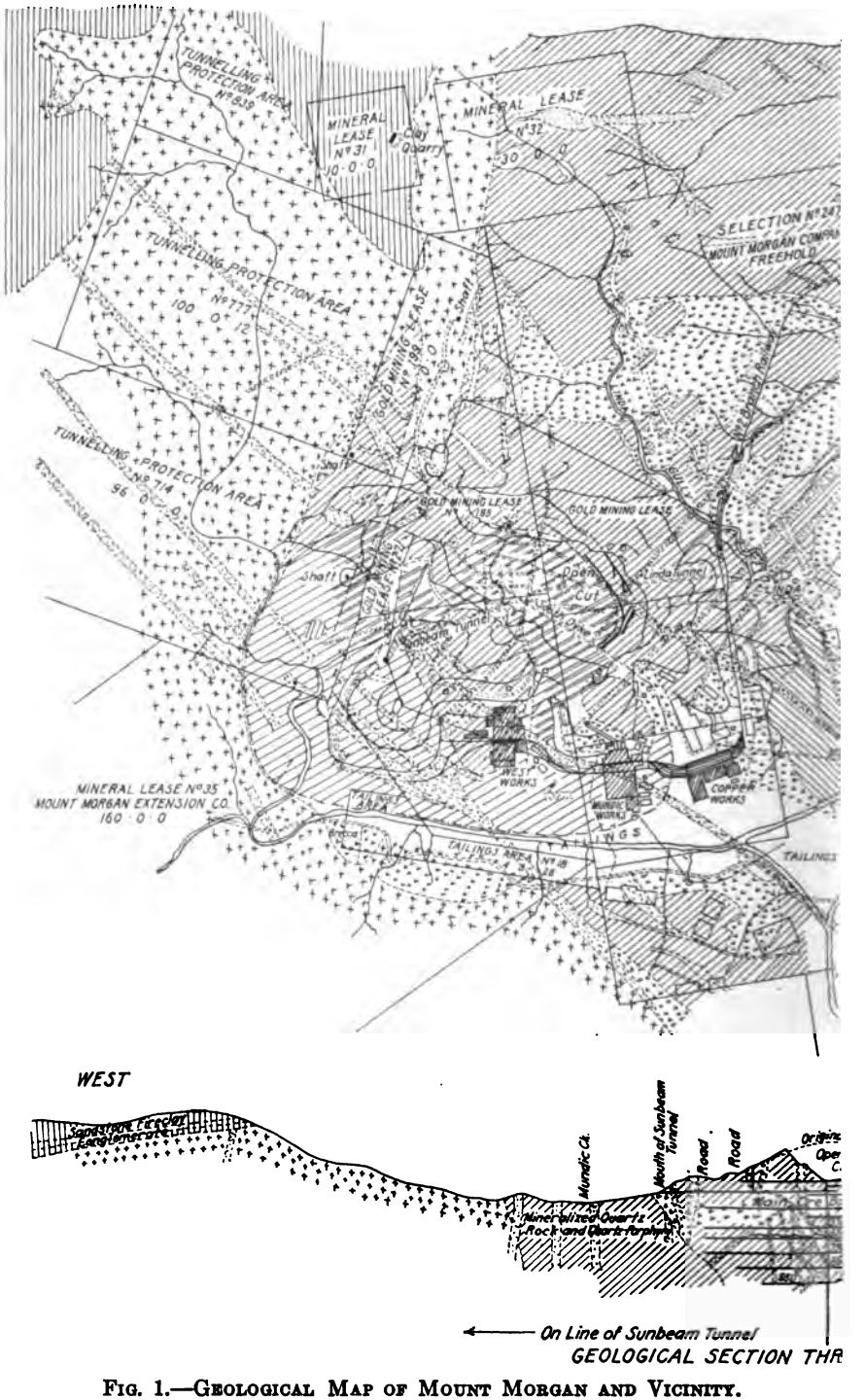

FIG. 1.—GEOLOGICAL MAP OF MOUNT MORGAN AND VICINITY.

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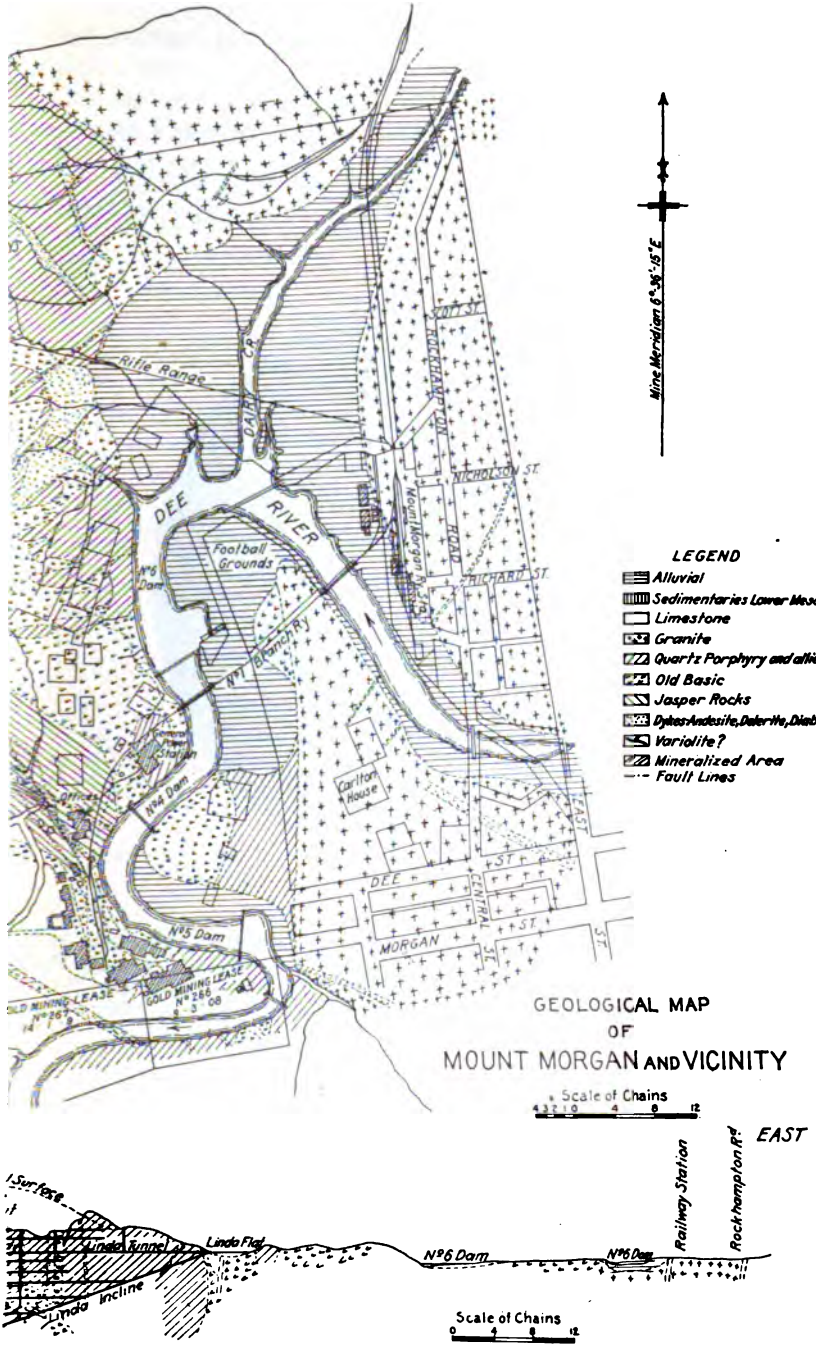

FIG. 1A.—GEOLOGICAL MAP OF MOUNT MORGAN AND VICINITY.

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J. M. Newman and G. F. Campbell Brown.[2] Reference to the literature shows a wide divergence of opinion concerning its geology to have existed at different times.

Summary of Opinions on Genesis of Mount Morgan Orebody

The first examination was made by R. L. Jack, as geologist for the Queensland Government, in 1884, and he came to a conclusion, from the

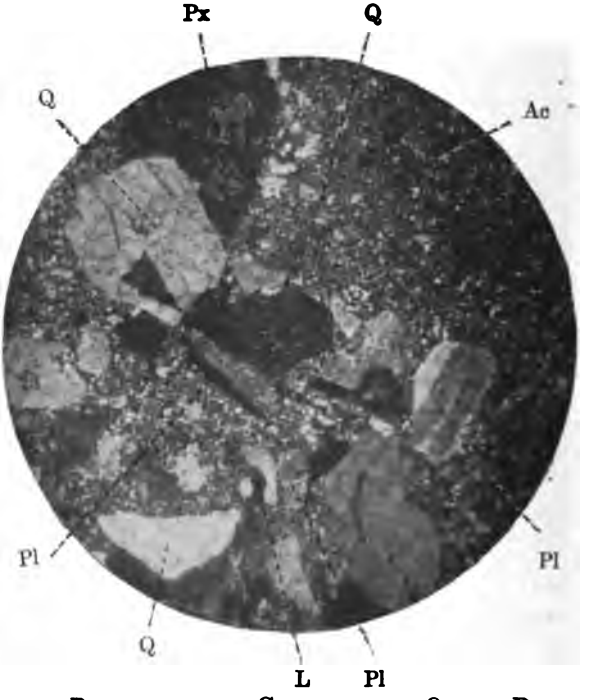

FIG. 2.—QUARTZ-PORPHYRY OF THE COMPOSITION OF QUARTZ-DIORITE OR QUARTZ-MONZONITE, WITH A GROUND-MASS OF FINE GRANULAR SECONDARY QUARTZ, CHLORITE, LEUCOXENE, AND HEMATITE. PLAGIOCLASE AND QUARTZ MAKE UP THE PHENOCRYSTS, AND PATCHES OF PYROXENE ARE COMMON. THE LATTER HAS BEEN PARTLY ALTERED TO CHLORITE AND EPIDOTE. CROSSED NICOLS. × 24.

evidence then available, that nothing but a thermal spring in the open air, or geyser, could account for the formation of the orebody. In 1891, Walter H. Weed of the United States Geological Survey was asked by Mr. Jack to examine and compare a suite of specimens from the mine with the siliceous sinters of the hot spring region of Yellowstone Park. Weed came to a similar conclusion, citing the Steamboat Springs of Nevada, which have long been known to be surrounded by deposits of sinter in fissures of which ore deposition is taking place.

T. A. Rickard, after an examination of the mine the same year, sum-

[2] *Loc. cit.*, pp. 439 to 470.

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marized the various views held up to this time as follows: 1. That the deposit is one of a geyser (R. L. Jack, 1884). 2. That it is an auriferous zone traversed by a series of quartz veins of auriferous mundic (gold-bearing pyrite) (J. MacDonald Cameron, 1887). 3. That it is the decomposed cap of a large pyrite lode. (The view held by several local and other mining engineers at the time of his examination.)

Mr. Rickard dismissed the geyser theory as an altogether local appearance, based as it was on a small open cut which showed a fan-shaped

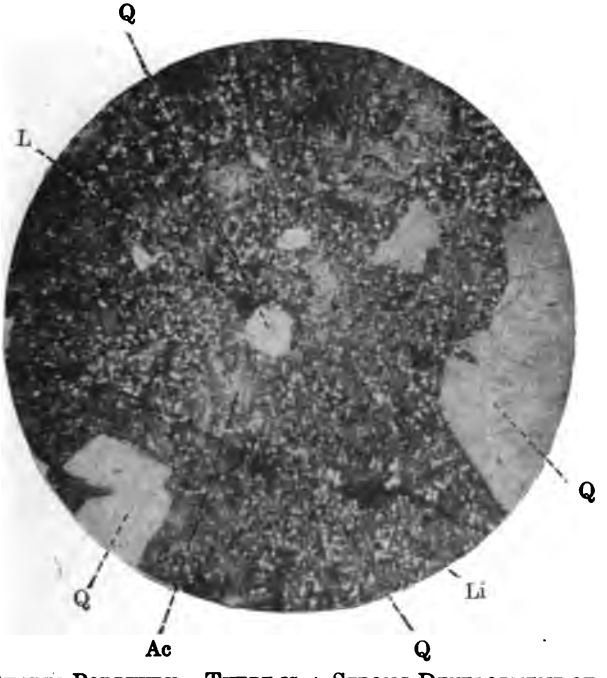

FIG. 3.—QUARTZ-PORPHYRY. THERE IS A STRONG DEVELOPMENT OF ACTINOLITE AMONG THE SMALL GRAINS OF SECONDARY QUARTZ WHICH MAKES UP THE GROUND-MASS, PROBABLY A RESULT OF PROXIMITY TO THE BASALTIC ROCKS. LEUCOXENE (WHITE AND OPAQUE, HENCE DARK) IS ABUNDANT. PHENOCRYSTS OF QUARTZ, AND GROUND-MASS LARGELY SECONDARY QUARTZ. LIMONITE, AS EVIDENCE OF WEATHERING, OCCURS DISSEMINATED AND ALONG FRACTURES. CROSSED NICOLS. × 24.

arrangement of ore, and his own explanation of Mount Morgan was, that it "represents an altered portion of shattered country rock, which, by reason of its crushed condition, was readily acted upon by mineral solutions, and that these solutions replaced the basic and feldspathic with acidic and quartzose material which was also gold-bearing."

The geyser and replacement theories were later reviewed by Dr. Frank D. Adams of McGill University with an interesting account of the discovery and subsequent history of the mine. But by 1899, the reaching of depths below the zone of oxidation revealed data which did not agree with some of these earlier conceptions.

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Many different names and origins have also been assigned to the rocks of the district. Metamorphism was supposed by some to have transformed original sedimentary rocks into the present ones, which are, as is generally stated, difficult to distinguish from igneous rocks. The supposed graywacke, quartzites, and shales of sedimentary origin are now known to be igneous or derived from igneous rocks. There are no true sediments near the mine except those to the northwest of it, which overlie

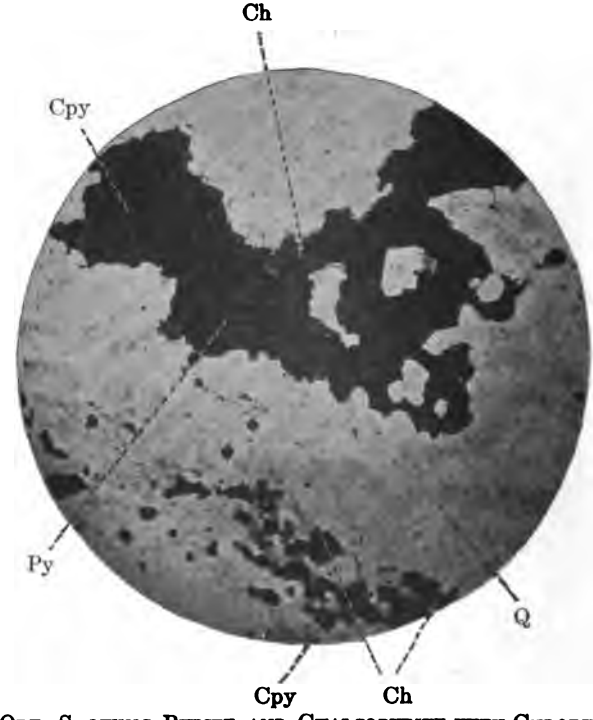

FIG. 4.—ORE, SHOWING PYRITE AND CHALCOPYRITE WITH CHLORITE IN GANGUE OF QUARTZ, THE CHLORITE WITHIN AND AROUND THE SULPHIDES, ALL THREE APPEARING TO OCCUPY THE SPACES AND CHANNELS BETWEEN THE QUARTZ GRAINS. SOME SMALL GREEN NEEDLE-LIKE CRYSTALS, NOT PLAINLY SEEN, ARE PROBABLY ACTINOLITE. PLANE LIGHT. × 24.

the very latest of the intrusive masses. These sediments are Lower Mesozoic in age and consist of conglomerates, fireclays, and sandstones, with a possible bed of volcanic ash or tuff. No specimen of the latter material was available, so this point could not be determined.

The notes by Newman and Brown, and by G. S. Hart, in the *Transactions* of the Australasian Institute of Mining Engineers, from whose maps and descriptions the field relations are taken, represent the most recent published work on the mine.

It is hoped the additional facts here set forth, from microscopic study, may be of assistance in solving the problem of the origin of its ores, as well as give additional information concerning the associated rocks.

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In some cases the names of rocks have been changed to more nearly fit their mineralogy and textures as revealed in thin section, the names here given according with the American usage. Acknowledgments are due the authors of the above papers, to Mr. Magnus for specimens and notes, and to Dr. Charles P. Berkey of Columbia University for his kindly assistance in the laboratory.

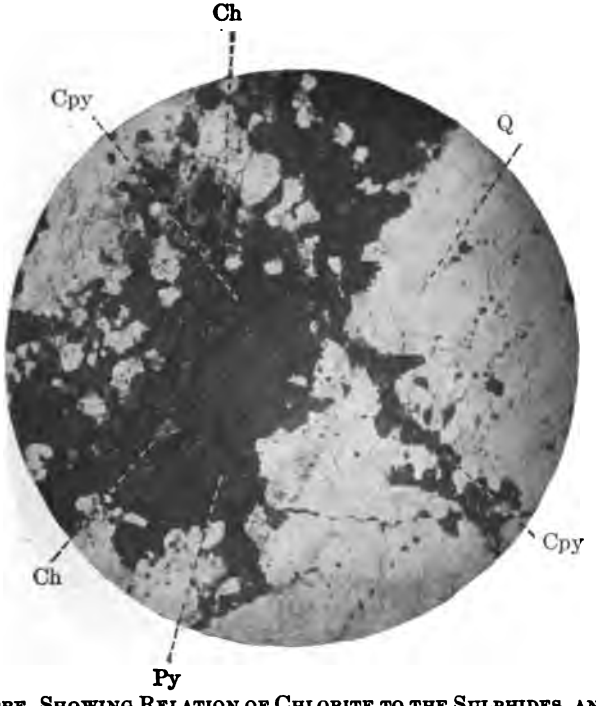

FIG. 5.—ORE, SHOWING RELATION OF CHLORITE TO THE SULPHIDES, AND DISTRIBUTION OF THE PYRITE AND CHALCOPYRITE WITH THE CHLORITE ALONG THE VEINLETS AND SPACES IN THE QUARTZ. PLANE LIGHT. × 24.

DESCRIPTION OF MICROGRAPHS

The accompanying geologic map, Fig. 1, will make clear the general relationship of the Mount Morgan rocks, and with this in mind, the descriptions follow in the order of their relative geologic age.

Key to Minerals in Photomicrographs

Ac	= Actinolite	L	= Leucoxene
Ap	= Apatite	Li	= Limonite
Aug	= Augite	M	= Magnetite
C	= Carbonate	Pl	= Plagioclase
Ch	= Chlorite	Px	= Pyroxene
Cpy	= Chalcopyrite	Py	= Pyrite
Hb	= Hornblende	Q	= Quartz
Il	= Ilmenite		

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Quartz-Porphyries and Ore

These rocks are grayish green and porphyritic with a fine-grained ground-mass and scattered irregular-shaped crystals of quartz. Two photomicrographs, Figs. 2 and 3, show them in thin section. Plagioclase is the predominant feldspar and ilmenite is the chief accessory mineral. Secondary development of quartz, actinolite, chlorite, epidote, kaolin, leucoxene, and hematite is a very pronounced characteristic. Later

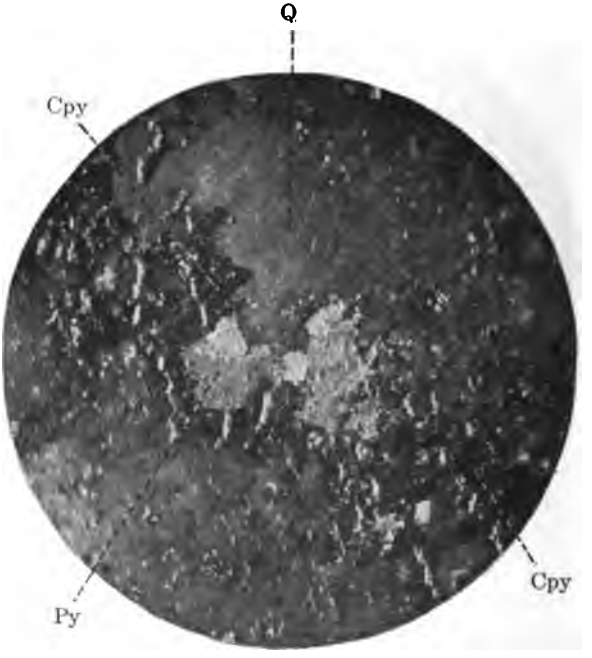

FIG. 6.—ORE. POLISHED SECTION SHOWING DISTRIBUTION OF PYRITE AND CHALCOPYRITE IN THE QUARTZ GANGUE. SILVER NITRATE HAS BEEN USED TO STAIN THE CHALCOPYRITE. THE MINERALS ARE DISSEMINATED AS REPLACEMENTS IN THE QUARTZ, AS WELL AS DISTRIBUTED ALONG THE VEINLETS AND CHANNEL BETWEEN THE QUARTZ GRAINS. BOTH SULPHIDES APPEAR TO HAVE BEEN DEPOSITED AT THE SAME TIME. REFLECTED LIGHT. × 24.

changes have developed some limonite, which is probably derived from pyrite, though occurring chiefly along fractures. Other fractures appear in the slides which are still more recent than these.

The belt of quartz-porphyry, which is the oldest of the rocks, is peculiar on account of the fact that the phenocrysts in it seem to have been subjected to a certain amount of primary fracturing. This would indicate that their original matrix had become somewhat viscous before movement within the mass entirely ceased, although all evidence of flow structure in the matrix or ground-mass has since been obliterated, if ever developed. Specimens of the unaltered rock would undoubtedly

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tell this. Nevertheless, its general heterogeneous nature suggests that it was originally a surface flow or large extrusive mass.

The quartz-porphyry within the boundaries of the ore deposit has been thoroughly silicified, and this gray quartzose material is streaked with veinlets and bands of pyrite and chalcopyrite. The original structure of the rock is only slightly preserved and Figs. 4 and 5 will show it to be replaced by aggregates of quartz grains of irregular size and shape.

FIG. 7.—SO-CALLED VARIOLITE, A MARGINAL PHASE OF THE QUARTZ-PORPHYRY, MADE UP OF SECONDARY QUARTZ AND ACTINOLITE. THE APPARENT PECULIAR SPHERULITIC STRUCTURE SHOWN IN THE PHOTOGRAPH IS A SECONDARY DEVELOPMENT. A, ACTINOLITE IN FIBROUS PATCHES BORDERING THE FINE QUARTZ AGGREGATES. B, FINE SECONDARY QUARTZ. C, COARSER-GRAINED NUCLEUS OF SECONDARY QUARTZ. CROSSED NICOLS. × 24.

Veinlets occur which include chlorite and the sulphides. The banded nature of the secondary quartz is also apparent. The interior structure of the quartz grains, their outline and irregular shape, features which are not brought out clearly except under crossed nicols, suggest the replacement of porphyritic material.

The photograph of a polished section of the ore, Fig. 6, shows the distribution of the pyrite and chalcopyrite. The minerals occur partly disseminated as small crystalline aggregates in the quartz mass and partly concentrated along veinlets and channels between the quartz grains. They appear to have been deposited simultaneously, though somewhat later than the period of silicification of the quartz-porphyry.

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The chlorite occurs within and around the sulphides as patches, and possibly actinolite along with it as needles, occupying with the sulphides the spaces and channels between the quartz grains. The chlorite is believed to have formed from the ferromagnesians of the original rock, incident with its silicification, and to have had some influence on the precipitation of the sulphides. As a mineral, it is too universal an alteration product to have a more direct significance. The actinolite will be discussed later in connection with the associated rocks.

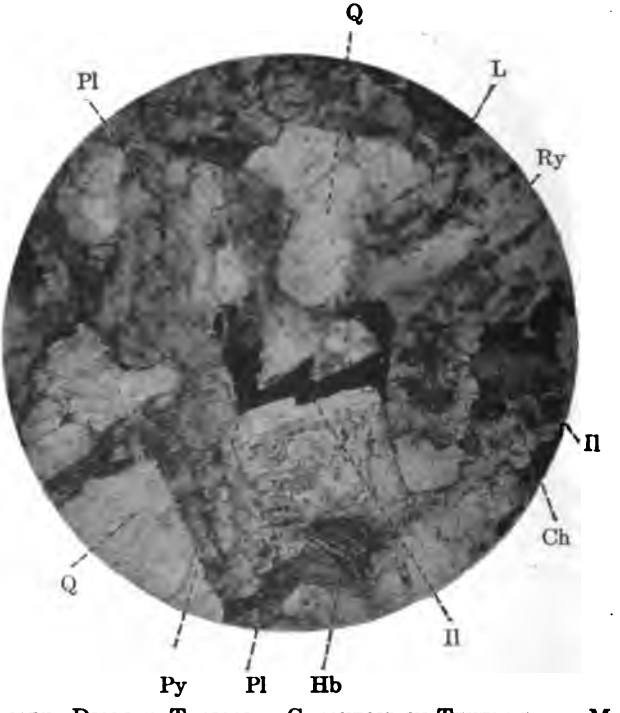

FIG. 8.—QUARTZ-DIORITE, TYPICALLY GRANITOID IN TEXTURE AND MADE UP OF INTERLOCKING GRAINS OF PLAGIOCLASE, QUARTZ, AND HORNBLENDE. ILMENITE AND APATITE ARE ACCESSORY MINERALS. PYRITE OCCURS IN THE ROCK AND MAY POSSIBLY BE PRIMARY, AS IT DOES NOT APPEAR TO CUT THE OTHER CRYSTALS. PLANE LIGHT. × 24.

A peculiar marginal phase of the quartz-porphyry has been developed at the northeast edge of the orebody near the dikes for which the name variolite, see Fig. 7, has been suggested on account of its peculiar spherulitic structure. But variolite is a variety of basaltic glass. The apparent spherulitic structure here is merely a peculiar secondary development of quartz and actinolite.

The Quartz-Diorite

The granite, on account of the predominance of plagioclase feldspar, is best termed a grano-diorite or quartz-diorite. It is a speckled, greenish-

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gray and white rock showing abundant hornblende and plagioclase, with quartz, in large interlocking grains. It shows some slight fracturing, and zonal growth of plagioclose. The feldspar is basic in composition, probably labradorite, and comparatively free from alteration. Some carbonization is observable in the feldspar, and the hornblende is largely changed to chlorite. Ilmenite and apatite are accessory minerals, some of the apatites being 2 to 3 mm. in length. Pyrite occurs in the rock and may possibly be primary, since it does not appear to cut structures in the other minerals to any considerable extent (see Figs. 8, 9, and 10).

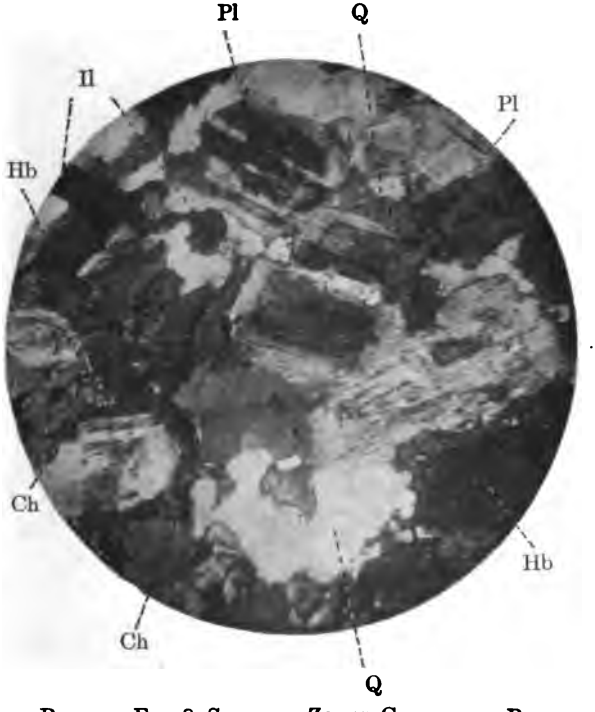

FIG. 9.—SAME ROCK AS FIG. 8, SHOWING ZONAL GROWTH OF PLAGIOCLASE, ALSO THE DEVELOPMENT OF CHLORITE FROM HORNBLENDE. CROSSED NICOLS. × 24.

The "Old Basic"

This body is intrusive into the quartz-porphyry as laccoliths and numerous steep and inclined dikes. It is a dark gray felsitic rock, weathering on its surface to a whitish limonite coating. Though felsitic, it does have a few very small feldspar phenocrysts (see Figs. 11 and 12). The rock is composed of small criss-crossed columnar crystals of plagioclase with some larger irregular grains of the same mineral. All are surrounded by pale green actinolite and chlorite, derived from altered pyroxene, and small grains of magnetite are scattered through the mass.

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Quartz is present in small grains but is very largely of secondary origin. Small and sparse crystals of pyrite occur associated with patches of chloritic material and actinolite. The heavy iron content in magnetite attests to the basic character of the rock, the magnetite having also resulted from the alteration of the pyroxene. Evidently the ferromagnesian minerals were unstable under conditions which did not affect the feldspars. The rock is an unusual type, though composed of common minerals, and may be called an andesine basalt.

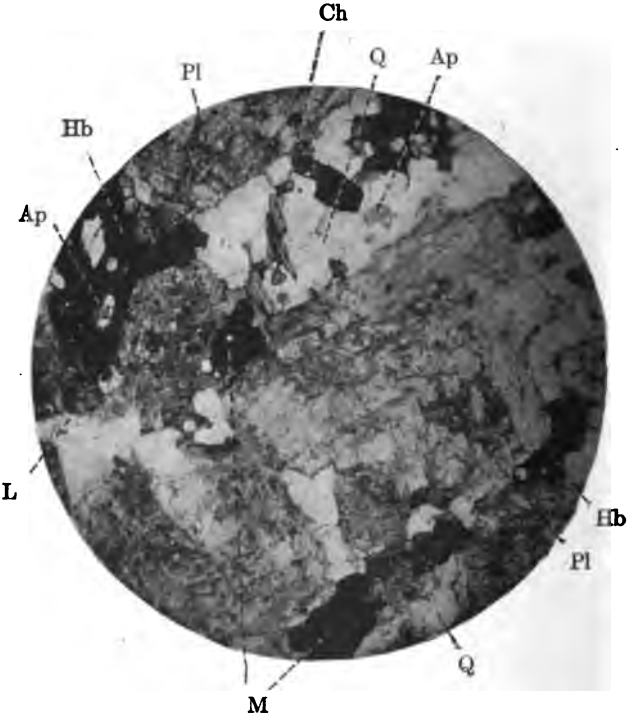

FIG. 10.—THIS PHOTOMICROGRAPH IS SIMILAR TO FIG. 8. IT SHOWS ABUNDANT PLAGIOCLASE, QUARTZ, AND HORNBLENDE. MAGNETITE, POSSIBLY ILMENITE, IS A PROMINENT ACCESSORY MINERAL, AND APATITE CRYSTALS OF FAIRLY GOOD SIZE ARE SCATTERED THROUGH THE ROCK. ALTERATION HAS AFFECTED MOST OF THE FELDSPARS TO SOME EXTENT, ATTACKING THEM ALONG STRUCTURE AND FRACTURE LINES. PLANE LIGHT. × 24.

Fig. 13 illustrates one of the thin sections under high power, and shows very clearly the small fibrous crystals of actinolite associated with the magnetite. As above stated, the actinolite, and probably most of the magnetite, is regarded as a metamorphic derivation from the original pyroxene in the rock, pseudomorphs of which are common through it. Pyrite also occurs in places penetrated by crystals of the actinolite, tending to the belief that this mineral, as well as the pyrite, is made up partly

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of introduced material. However, the pyrite may have formed around the actinolite at a later stage without replacing it.

Stope Dike

This rock, Fig. 14, belongs to the older dike series. It is dense, black and felsitic, very much resembling the "Old Basic," if indeed it is not the same rock. It includes an occasional fine patch of minute pyrite crystals, and weathers with a reddish iron stain. The structure of the

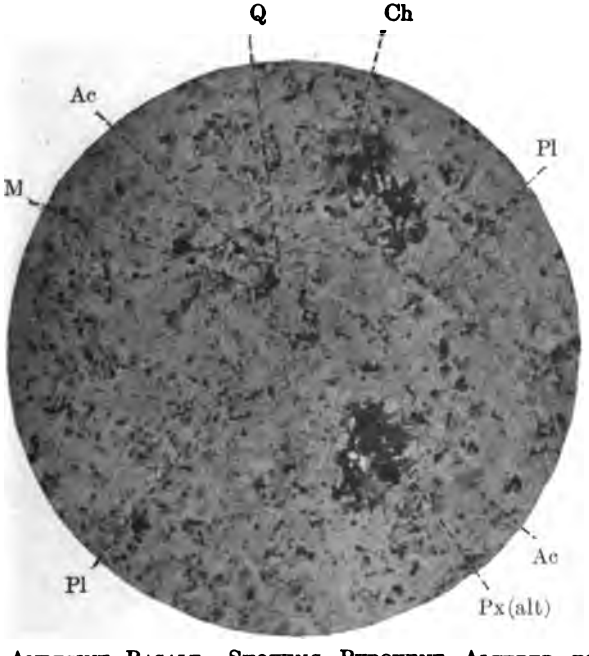

FIG. 11.—ANDESINE BASALT, SHOWING PYROXENE ALTERED TO SCATTERED AGGREGATES OF MAGNETITE WITH ACTINOLITE AND CHLORITE. FELDSPAR IS PROMINENT AND APPEARS TO HAVE BEEN THE FIRST MINERAL TO CRYSTALLIZE. ITS VARIETY IS ANDESINE. QUARTZ, LARGELY SECONDARY, IS PRESENT AS SMALL GRAINS THROUGHOUT THE MASS. THOUGH COMPOSED OF COMMON MINERALS, THIS IS AN UNUSUAL TYPE OF ROCK. PLANE LIGHT. × 24.

rock may be described as a network of small columnar crystals of plagioclase and fibers of actinolite, filled in with irregular grains of quartz and magnetite. A few fine fractures appear which are healed with chlorite. Pyroxene was an original mineral, which, together with the feldspar, has been largely altered to quartz and actinolite.

Dolerite Dikes

The material of these dikes varies slightly in composition and greatly in texture, some being diabasic and others basaltic. All the material

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is dark, massive and more or less felsitic. Fig. 15 shows the diabasic type. It is made up of rounded irregular grains of augite in a plexus of lath-shaped crystals of feldspar. The feldspar is plagioclase. The augite grains are small and irregular in outline, and appear to fill the spaces between the crossed crystals of plagioclase. Alteration has changed a considerable portion of it to chlorite and actinolite, part of which has replaced occasional lines and twinning bands in the plagioclase. Alteration has also changed some of the plagioclase to quartz. This

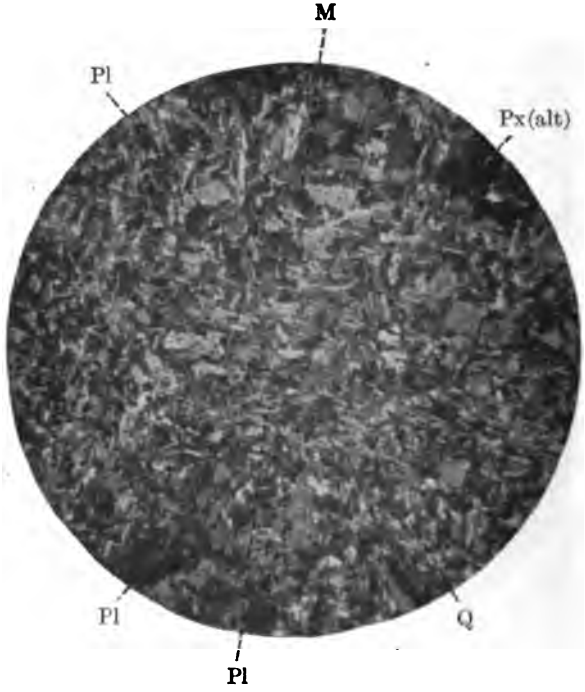

FIG. 12.—SAME ROCK AS IN FIG. 11. THE TEXTURE IS COARSELY FELSITIC TO PORPHYRITIC, AND THE CRISS-CROSSED ARRANGEMENT OF THE COLUMNAR PLAGIOCLASE CYSTALS IS APPARENT. THERE ARE A FEW PLAGIOCLASE PHENOCRYSTS VISIBLE, AND MAGNETITE IS DISTRIBUTED THROUGHOUT. CROSSED NICHOLS. × 24.

quartz seems to have been derived from both pyroxene and feldspar, being penetrated by fibrous actinolite crystals of simultaneous growth. However, it can not be said that all the quartz is secondary, for in the case of many interstitial grains, the traces of a replaced mineral are too slight to make it possible to say it existed.

The term Dolerite, as used on the map, while indicating rock of this general character, does not sufficiently emphasize the important structural feature shown in the arrangement of the feldspars in this particular specimen, and it would, therefore, be more properly called Diabase. The idiomorphism of the feldspars with respect to the pyroxene, as shown by

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Fig. 15, is also a distinguishing characteristic of diabase, along with their columnar structure and crisscross arrangement.

Similarly, for textural reasons, the dolerite shown in Fig. 16 is more properly called basalt. This material is a more felsitic variety of the rock composing this group of dikes. It seems to possess the same general alteration effects except that the quartz, if present, is not visible, which might argue that the quartz of the other basic rocks is entirely secondary. In the ground-mass are many fine rods of plagioclase, but as a whole it

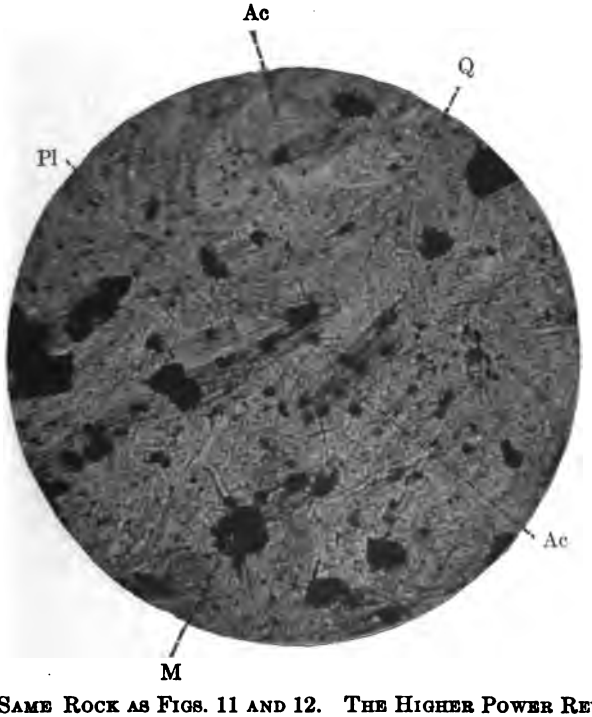

FIG. 13.—SAME ROCK AS FIGS. 11 AND 12. THE HIGHER POWER REVEALS SMALL FIBROUS CRYSTALS OF ACTINOLITE ASSOCIATED WITH THE MAGNETITE. THE QUARTZ AND PLAGIOCLASE ARE RATHER INDISTINCT. THE ACTINOLITE, AND PROBABLY MOST OF THE MAGNETITE, IS BELIEVED TO BE DERIVED FROM THE BREAKING DOWN OF ORIGINAL PYROXENE UNDER THE ACTION OF SILICEOUS MAGMATIC WATERS. PLANE LIGHT. × 257.

is finely felsitic and the rock a typical basalt. It contains some pyrite, which is generally taken to be an introduced substance.

The rock of the East Dike is a dark massive diabase, shown in Fig. 17. Quartz occurs in much the same relation to the crisscrossed columnar plagioclase as the pyroxene, and because this quartz may be partly primary the rock might well be called a quartz-diabase, which is a rather rare rock. In this original, the augite-pyroxene and plagioclase have been largely changed to chlorite, actinolite, and secondary quartz. The quartz and the unaltered plagioclase are penetrated to some extent by

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fibers of actinolite. The scattered magnetite is probably also secondary, and pyrite is present.

The variation in texture of these dike rocks is mainly due to different rates of cooling, occasioned either by the influence of wall rock, which may have been hot or cold, or by the thickness of the dikes themselves. Their chemical composition must also have exerted a strong influence on the order of crystallization of the minerals, producing the tendency to idiomorphic development of the feldspar.

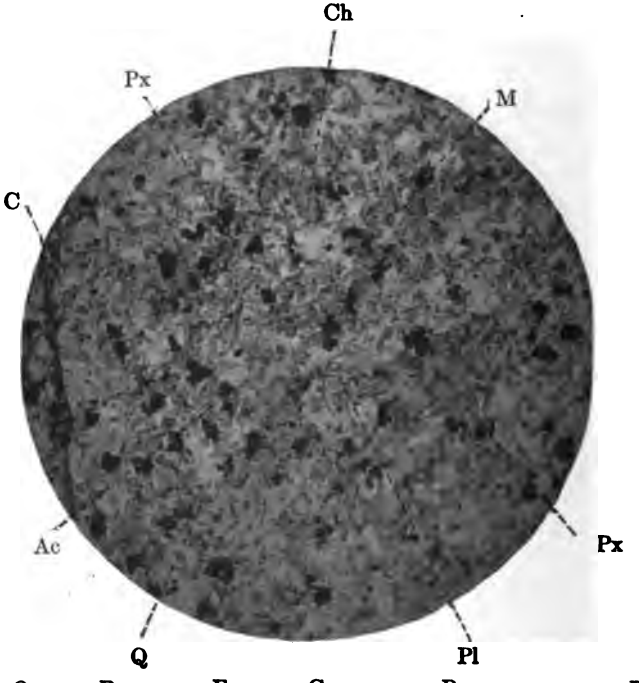

FIG. 14.—QUARTZ-BASALT, A FELSITIC COMPLEX OF PLAGIOCLASE AND PYROXENE WHICH HAS BEEN PARTLY ALTERED TO QUARTZ, ACTINOLITE, AND MAGNETITE, LEAVING ONLY A SMALL PART OF THE PLAGIOCLASE INTACT. PART OF THE QUARTZ, HOWEVER, APPEARS TO BE ORIGINAL. ABUNDANCE OF MAGNETITE, PART OF WHICH IS ALSO PRIMARY. SMALL CARBONATE VEIN. PLANE LIGHT. × 24.

SUMMARY

It will be seen from the descriptions above that, aside from the quartz-porphyry, the principal rocks associated directly with the ore are basaltic in character. They contain quartz, part of which I judge to be primary and part secondary. Magnetite is widely distributed and appears to have resulted in large part from the breaking down of the original ferromagnesian minerals. The rocks have undergone great change, but do not appear to have been much affected by superficial weathering.

The strong development of actinolite in all the specimens indicates

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either the alteration of pyroxene and plagioclase by solutions, or their change to a more stable mineral under conditions of regional metamorphism. The first is more probable, since there is not the slightest evidence of metamorphism in the surrounding grano-diorite. The pyrite in the basic rocks appears to be later than the actinolite, but as a whole is probably associated with the same period of alteration by hot solutions, the phenomena of expiring vulcanism. It is not known whether this pyrite contains gold or copper. In respect to the actinolite, it is not

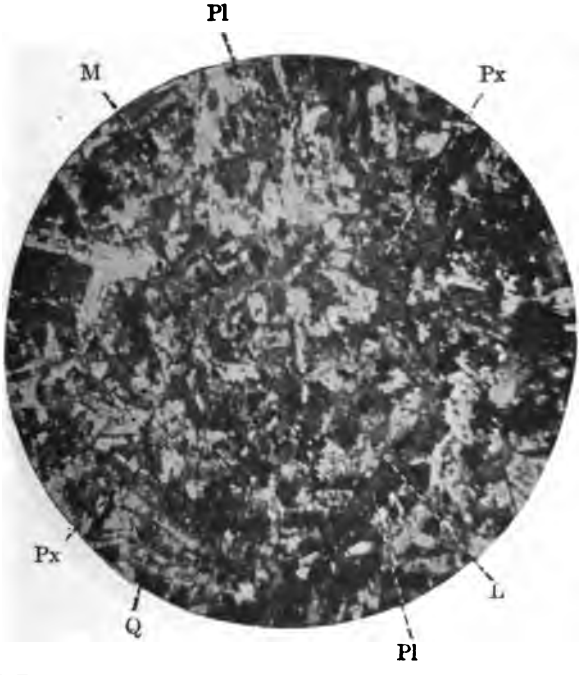

FIG. 15.—A DIABASE WITH THE CHARACTERISTIC TEXTURE. THE AUGITE GRAINS ARE SMALL AND IRREGULAR IN OUTLINE. MUCH OF THE PYROXENE HAS BEEN CHANGED TO CHLORITE AND SOME ACTINOLITE. TWINNING OF THE COLUMNAR PLAGIOCLASE CRYSTALS CAN BE SEEN. THE QUARTZ IS PROBABLY DERIVED FROM BOTH THE PYROXENE AND FELDSPAR, BEING PENETRATED BY FIBROUS CRYSTALS OF ACTINOLITE OF SIMULTANEOUS GROWTH. THE NAME DOLERITE IS AS GOOD A ONE AS ANY, BUT THE USAGE VARIES AMONG PETROGRAPHERS. CROSSED NICOLS. × 24.

possible to say that the alteration is due to direct igneous influence; but because actinolite is generally regarded as a high-temperature mineral there is the possibility of regarding it in much the same light as tourmaline, on the presence of which in the volcanic copper deposit at Braden, Chile, the primary character of the deposit was regarded as certain.

The quartz in general seems also to be a result of metamorphic action on the plagioclase and pyroxene, penetrated as it is by the contemporaneous crystals of actinolite. Part of the quartz in the dikes, however, must be primary, as the sharp outline of some of the interstitial grains

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involve difficulty in conceiving the replaced original if these particular quartz grains were secondary. Hence the basic rocks might well be called quartz-basalts and quartz-diabases, which rocks are rather rare occurrences.

The grano-diorite or quartz-diorite intrusion shows some evidence of superficial weathering, but otherwise no alteration, and is considered a fairly fresh igneous rock. Pyrite is present in it, and the more altered character of the rock near the quartz-porphyries has led to the belief that its intrusion had something to do with the formation of the ores.

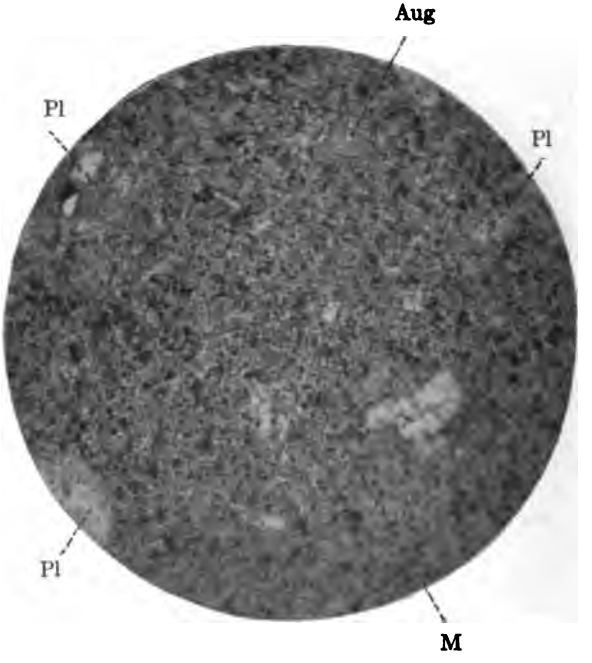

FIG. 16.—BASALT, A MORE FINELY FELSITIC VARIETY OF THE FOREGOING ROCKS, BUT CONTAINING SMALL SCATTERED PHENOCRYSTS OF PLAGIOCLASE. IN THE GROUND-MASS THERE ARE MANY LITTLE FINE RODS OF PLAGIOCLASE. QUARTZ, IF PRESENT, IS NOT VISIBLE. PLANE LIGHT. ×24.

The quartz-porphyries are thought to be such, rather than volcanic tuffs, because they contain no material foreign to the composition of a crystalline igneous rock, except the secondary quartz. By this quartz, however, these rocks are almost entirely replaced, the ground-mass completely so, and on this account their appearance under the microscope does somewhat resemble that of siliceous sinter. From the specimens examined and the mapping of this mass as quartz-porphyry, jasper, etc., its heterogeneous character, as of acid extrusive rock in general, is apparent. One of the specimens is a rhyolitic porphyry and another dacitic, and the structure of both, as already described, also suggests that the mass is an

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ancient surface flow of large extent. The later intrusions of quartz-diorite and the basic rocks have further altered its local character.

In regard to the sequence of the different igneous masses, it seems most reasonable from this study to put the "Old Basic" later than the grano-diorite and of the same age as the Stope Dike. No very strong structural evidence to the contrary is shown on the map, as the two apparent intrusions of grano-diorite into the "Old Basic" on the western and southern borders of the mineralized area may just as well be interpreted as

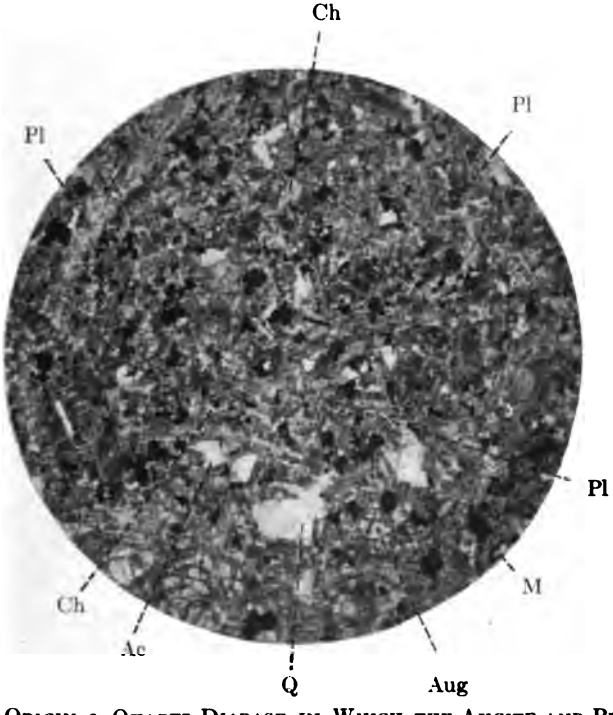

FIG. 17.—ORIGINAL QUARTZ-DIABASE IN WHICH THE AUGITE AND PLAGIOCLASE HAVE BEEN LARGELY CHANGED TO CHLORITE, ACTINOLITE, AND QUARTZ. THE DIABASIC TEXTURE IS EVIDENT, PYROXENE AND CHLORITE FORMING A FILLING BETWEEN THE CROSSED COLUMNAR CRYSTALS OF PLAGIOCLASE. ACTINOLITE NEEDLES PENETRATE THE QUARTZ WHICH, HOWEVER, IN SUCH A BASIC ROCK, MUST BE LARGELY SECONDARY. PLANE LIGHT. × 24.

intrusions of "Old Basic" around spurs of the grano-diorite, and the petrographic evidence we have in addition is very strongly in favor of placing the "Old Basic" in the same age relation to the grano-diorite as the two systems of basic dikes which cut it. All the basic-rock types show too marked a mineralogical and textural similarity, even in the character of their alteration, to be divided into two periods of intrusion by another rock of opposite character, the grano-diorite. However, such occurrences are not unknown in the history of rock magmas.

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According to this interpretation, the relative ages of the different masses are as follows:

Formation	Rock Name
9. Sedimentaries—Mesozoic.	
8. Faults.	
7. "East" Dike (Dolerite).	Quartz-diabase.
6. "North" and "South" Dikes (Diabase).	Quartz-basalt.
5. Andesite Dikes.	
4. "Old Basic," "Stope," "East and West" and "Cross" Dikes.	Andesine- and quartz-basalt.
3. "Flat" Dike.(?)	
2. Granite.	Quartz-diorite.
1. Porphyry.	Quartz-porphyry.

The ore is of granular secondary quartz containing pyrite and chalcopyrite with chlorite and actinolite, a highly silicified phase of the quartz-porphyry. No evidence of a brecciated or crushed zone appears in any of the slides studied. The enrichment has been derived from magmatic solutions, and there is no reason to believe that the pyrite and chalcopyrite were deposited at different times. The more abundant occurrence of the sulphides along spaces or channels between the grains of quartz, and in bands or streaks through the gangue, seems to indicate that they were deposited shortly after the quartz replacement, rather than at the same time. The associated chlorite and actinolite seem to have been derived mainly from the ferromagnesians of the original quartz-porphyry during the period of replacement by quartz.

The chlorite seen in the ore is, as stated before, believed to have no important significance, but referring to Figs. 2 and 3, the strong development of actinolite in the surrounding quartz-porphyry is believed to be closely associated with the sulphide mineralization. If such is the case, the presence of this mineral in the basic dikes might imply that they had to do with the enriching solutions rather than the earlier grano-diorite intrusion, as actinolite is a metamorphic mineral formed either as a result of hot water action or of direct igneous metamorphism. It is not uncommon to find it at the contact of basic with quartzose rocks, as, for instance, near the periodotite dikes in the Franciscan sandstone of California, which formation itself is highly metamorphosed.

It might be noted here also that many gold deposits are associated with basic rocks, and the further structural fact can not escape attention that the ore deposit here occurs near the intersection of two systems of basic dikes. Hence, to say the mineralization was contemporaneous with these dikes seems a closer correlation of the data than to associate it with the earlier grano-diorite, for there are numberless other points in the quartz-porphyry near the grano-diorite contact where enrichment might have occurred and did not do so.

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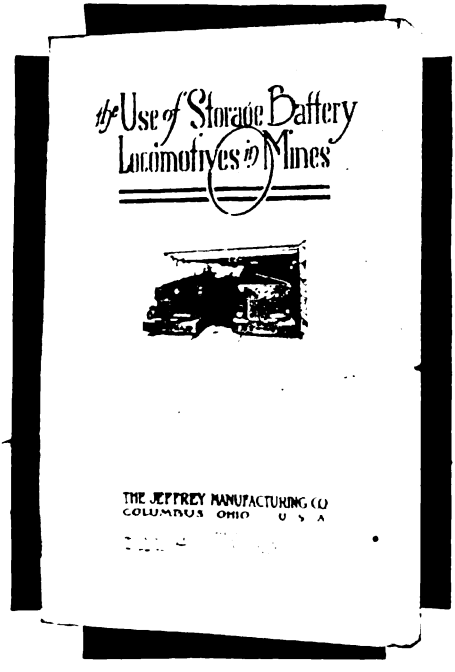

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OCTOBER, 1916

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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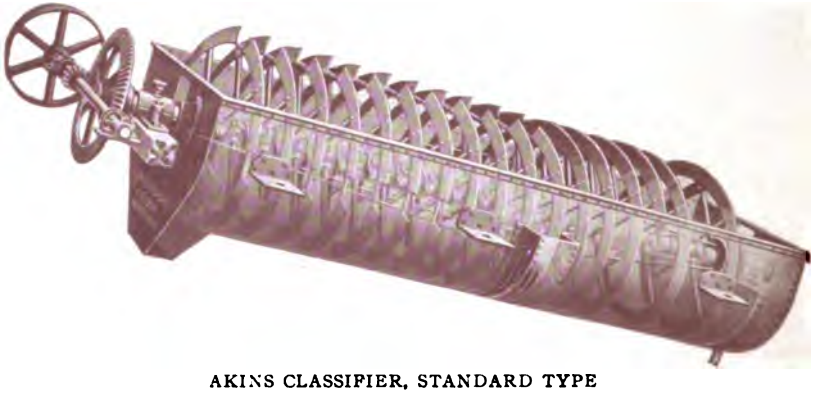

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BULLETINS WANTED

The Institute is still in need of additional copies of the *Bulletin* for January, February and March, and will be glad to pay members the regular price of fifty cents per copy. Copies may be sent express collect, or postage may be used and the amount will be refunded in addition to the fifty cents.

TRANSACTIONS FOR 1916

Vols. 52 and 53 have been shipped to all who have paid their dues for 1916. Members who do not receive them within a reasonable time are asked to notify the Secretary.

MANUSCRIPT CLOSING DATE FOR THE NEW YORK MEETING, 1917

The 114th (New York) Meeting of the Institute will be held in the third week of February, 1917. The Committee on Papers and Publications has set Dec. 1, 1916, as the closing date for the receipt of manuscripts intended for this meeting.

Manuscripts must be received by the Secretary on or before this date. If they are sent through members of the technical committees they must be sent to those committees enough in advance of the time to be forwarded and in the hands of the Secretary by the time specified.

While Dec. 1, 1916, is the final closing date, it is a great advantage to get papers in as much before that time as possible as it enables them to be submitted in proof form to their authors. Papers received on or before

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Nov. 1, 1916, will, as far as possible, be published in the January *Bulletin* or earlier, thus giving good distribution among the members in advance of the meeting.

The Institute's records show that the discussion of papers is very much better when they are distributed somewhat in advance of the meeting instead of in the *Bulletin* which goes out immediately preceding the meeting.

AMERICAN MINING CONGRESS

The American Mining Congress, of which the President is Carl Scholz, and the Secretary is J. F. Callbreath, and of which many of the officers and directors are prominent in Institute affairs, has extended to the members of the Institute a cordial invitation to attend the nineteenth Annual Session of the Congress, to be held at the La Salle Hotel, Chicago, Nov. 13 to 16, 1916, and to take part in its deliberations.

The morning sessions will be general meetings at which the attendance of all members of the Convention is hoped for, and which will be devoted to a discussion of subjects related to the three several heads, "safety," "efficiency" and "conservation." The afternoon meetings will be divided into several sections, each devoted to the discussion of subjects of interest to its special branch of the industry. It is planned to have all papers printed and distributed, allowing the author a limited time in which to present the salient features of his address and giving up the greater part of the time to a discussion of the subject presented by members on the floor. By this plan it is hoped to have a free expression of opinion and to provide ample time for reaching conclusions to be embodied in resolutions representing the best thought of the mining industry.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period August 1, 1916 to September 10, 1916:

C. L. Cantley, New Glasgow, N. S.
D. A. Cowell, Columbus, Ohio.
Alexander Deussen, Houston, Texas.
Chas. E. Dutton, Goldfield, Nev.
H. S. Emlaw, Grand Haven, Mich.
J. M. Gillette, Swatsburg, N. Y.
E. Giroult, San Antonio, Texas.
Bertram Grant, Kingman, Arizona.
Ruger W. Hay, Bisbee, Arizona.
C. F. Hendricks, Cobalt, Ontario.
Stanley C. Herold, San Jose, Cal.
Lloyd Hoffman, Pottersville, N. J.
Charles Hunter, London, England.
G. Jansen, Benkoelen, Sumatra.
T. H. Jenks, Oatman, Arizona.
Warren Jenny, Chanaral, Chile.
H. M. La Follette, La Follette, Tenn.
Montrose L. Lee, Montevideo, Uruguay.
V. H. McNutt, Billings, Mont.
J. W. Maller, New York, N. Y.
Edwin W. Mills, Tul. Mi Chung, Chosen, Korea.
Francis R. Pyne, Elizabeth, N. J.
Frank A. Ray, Columbus, Ohio.
John A. Rice, El Paso, Texas.
W. E. Rice, Ringwood Manor, N. J.
W. H. Staver, Lynchburg, Va.
Ralph H. Sweetser, Easton, Pa.
Kirby Thomas, New York, N. Y.
Henry Traphagen, Union Hill.
M. M. Valerius, Tulsa, Okla.
Wilfrid B. Wainewright, London, Eng.

Walter A. Barrows, Jr., of Brainerd, Minn., has been elected president and general manager of the Thomas Iron Co., Easton, Pa., to succeed Ralph H. Sweetser, whose resignation of several months ago becomes

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effective Aug. 31. Mr. Barrows has had an extended experience in blast-furnace and iron-mining work. He was for several years general manager of the Shenango Furnace Co., and for 5 years past has been engaged in iron mining and exploration in Minnesota.

Edwin S. Berry has become associated with Pope Yeatman and will engage in consulting mining engineering with offices at 111 Broadway, New York, N. Y.

Dr. G. H. Cox and **D. H. Radcliff** have opened offices in Tulsa, Okla., as consulting geologists.

W. R. Hamilton has withdrawn from the management of the Montobello Oil Co. He has opened offices in the Hobart Building, San Francisco, Cal., and will devote his time to general engineering work in petroleum and metal mining.

George B. Holderer, recently manager of the Furlough Development Co., in Arizona, is now with the General Chemical Co., 25 Broad Street, New York.

Joseph Jensen, mineral inspector in the Field Service of the U. S. General Land Office, has been appointed Assistant Professor of Mining at the Carnegie Institute of Technology, Pittsburgh, Pa.

Robert Livermore has resigned as manager of the Kerr Lake Mine, Cobalt, and is succeeded by H. A. Kee of the Nipissing Mining Co., Cobalt.

James MacNaughton, general manager and vice-president of the Calumet & Hecla Mining Co., is to take up his residence in Boston on Dec. 1. He will continue to manage the company. **John Knox,** superintendent, will take care of a considerable part of the work at the mine. Mr. MacNaughton will be a frequent visitor to the copper country and there will be no change in the titles of the two men.

Howard C. Parmelee, formerly Western editor of *Metallurgical & Chemical Engineering* of New York, has accepted the presidency of the Colorado School of Mines.

H. W. Ross has been appointed assistant manager for Backus & Johnston Co., Casapulca, Peru.

Lyon Smith has resigned as metallurgist with the Snyder Electric Furnace Co. of Chicago, and has gone to Florence, Colo., to be Assistant Superintendent with the River Smelting and Refining Co.

Thomas A. Stroup has left the engineering staff of the Tennessee Copper Co., and is now engineering assistant with the Utah Copper Co., at Salt Lake City, Utah.

William Thomlinson, managing director of the Seaton Carew Iron Co. Ltd., West Hartlepool, England, and the Carlton Iron Co. Ltd., of Ferry Hill, and Lieutenant Colonel in the Durham Light Infantry, raised and trained for Kitchener's army the 19th Durham Light Infantry (Bantams) and the 22d Durham Light Infantry (Pioneers). He was at one time at South Pittsburgh, Tenn.

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ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, Cornell M. E., graduate, aged 31, married. Will soon complete 2 years' engagement in the Russian oil fields. Eight years' practical experience in drilling for and producing oil. Can speak Russian fluently and some French and German. Open for engagement. Interview after August 15, in the United States. No. 299.

Member, graduate mining engineer. Married. Aged 38. For past 6 years, general superintendent of large silver mine in Mexico. Desires connection with responsible mining company, preferably in the United States. Has had 15 years' experience in the United States and Mexico, including mine engineering, sampling, examinations, milling and mine superintendence. No. 312.

Metallurgist, 4 years' experience in electric furnace operation on steel, ferro-alloys and smelting of ores, both practical and research, desires position as metallurgist or superintendent. No. 314.

Member, technical mining graduate. 5 years' experience in coal and metal mines and construction work. Married. Aged 27. Employed at present. Interview in Salt Lake City or Denver. No. 316.

Member, American, aged 39, married, desires position as superintendent or mine superintendent. 14 years' experience in opencut and underground work. 12 years' Spanish, Portugese, Cuban and Mexican labor. Fluent Spanish. Good training in first aid and safety first. Specialty of high explosives and steam-shovel work. Good costs. Go anywhere, but tropics preferred. References. Interview, New York or Philadelphia. No. 318.

Member, aged 29, unmarried, technical graduate. 7 years' varied experience in metal mining: 3 years' as draftsman, surveyor and assayer; 4 years' operating experience as foreman, superintendent, and manager. Good health and habits. References. At present employed, available on short notice. Can handle men efficiently and get results. Employment with reputable company desired, preferably as foreman or superintendent. No. 319.

Member. Graduate Columbia School of Mines. Single. 30 years old. Six years' practical experience, 5 years in Mexico and Central America. Fluent Spanish. Good organizer, able to handle men. Desires position as superintendent or assistant in metal mine in U. S. or foreign country. No. 320.

Young technical graduate who has had 1 year's post graduate work in geology, would like to get into a geological department or to work as a mining engineer in engineering or mining departments. Willing to go anywhere. No. 321.

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LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

COAL MINERS' POCKETBOOK. Ed. 11. New York, 1916.

CONSTRUCTION AND OPERATION OF A SINGLE TUBE CRACKING FURNACE FOR MAKING GASOLINE. Technical Paper No. 161, U. S. Bureau of Mines, Washington, 1916.

THE NEW ANACONDA. July, 1916. New York, Eugene Mayer, Jr. & Co., 1916. (Gift of publisher).

ELEMENTS OF MINING. By George J. Young. McGraw-Hill Book Co., Inc., New York, 1916. Price $5. (Gift of publishers.)

[NOTE.—This exceedingly useful little book will be found very valuable as a textbook and reference book for those who are taking up the subject of mining. It is written in a way which will commend itself to teachers and to those beginning the subject. It covers the fundamental principles and from them elaborates on different details of mining practice; the relation of geology to mining, and other related topics. The scope of the book is shown by the following list of titles of chapters:

Preface, Introductory, Prospecting, Boring, Drilling for Blasting Purposes, Rock Breaking, Transportation and Hoisting, Mine Drainage, Ventilation and Illumination, Support of Mine Workings, Open-Pit Mining, Alluvial Mining, Development, Underground Methods, Mine Organization and Operation, Mine Accounting, Accidents and Miners' Diseases, Examination of Mineral Deposits, Index. B. S.]

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Geology and Mineral Resources

AMERICAN FERTILIZER HANDBOOK, 1916. Philadelphia, Ware Bros. Company, 1916. (Gift of publishers.)

COALS OF LETCHER COUNTY. Ser. IV, Vol. 4, pt. 1, Kentucky Geological Survey, Frankfort, 1916.

GEOLOGY OF THE SHAWNEETOWN QUADRANGLE IN KENTUCKY. Kentucky Geological Survey, Frankfort, 1916.

OIL SHALES IN THE PORT CURTIS DISTRICT. Publication No. 249, Queensland Geological Survey.

PETROLEUM PROSPECTS ON BRUNY ISLAND. Tasmania, 1916. Report by Arthur Wade.

PETROLEUM WITHDRAWALS AND RESTORATIONS AFFECTING THE PUBLIC DOMAIN. Bull. No. 623, U. S. Geological Survey, Washington, 1916.

RAPPORT SUR LES GISEMENTS DE CUIVRE DES CANTONS DE L'EST DE LA PROVINCE DE QUÉBEC. By J. A. Bancroft. Quebec, 1916.

THE ADELONG GOLDFIELD. Mineral Resources, No. 21, New South Wales. Department of Mines, Sydney, 1916.

THE SOUTH HEMMSKIRK TIN FIELD, with maps. Bull. No. 21, Tasmania. Geological Survey, Tasmania, 1916.

WOLFRAM MINES OF MOUNT CARBINE. Publication No. 251, Queensland. Geological Survey, Brisbane, 1915.

DETAILED REPORT on Raleigh County, Summers West of New River, and the Coal Area of Mercer County, by C. E. Krebs, assisted by D. D. Teets, Jr., with a chapter on Kanawha Marine Fossils by Wm. Armstrong Price, issued under date of August 1, 1916, with topographic and geologic maps. Eastern Raleigh, Summers and Mercer lie within the great New River and Pocahontas smokeless coal districts, while western Raleigh holds immense deposits of Kanawha Splint and gas coals. Price of Report with case of maps, including soil report and map, delivery charges paid by the Survey, $2.50. West Virginia Geological Survey, P. O. Box 848, Morgantown, W. Va.

General

CHART SHOWING PRICES FOR NON-FERROUS METALS FOR 1898–MARCH, 1916. (Gift of Daily Iron Trade.)

THE ENGINEER IN WAR. By P. S. Bond. New York, 1916.

METHODS OF SAMPLING DELIVERED COAL. Bull. No. 116, U. S. Bureau of Mines. Washington, 1916.

PRACTICAL LUBRICATION. By Lieut. G. S. Bryan. (Reprinted from Jour. Amer. Soc. of Naval Engineers, Vol. 27, Nov. 1915). (Gift of The Texas Company.)

TABLE OF THE EIGHTY-FOUR RESEARCH PAPERS AND SCIENTIFIC ADDRESSES BY SIR ROBERT HADFIELD, 1888–1912. (Gift of Sir Robert Hadfield.)

THEORY AND CALCULATION OF ALTERNATING CURRENT PHENOMENA. Ed. 5. By C. P. Steinmets. New York, 1916.

TRAINING OF OUR CAPTAINS OF INDUSTRY. By Sir Robert Hadfield. (Reprinted from Iron and Coal Trades Review, June 9, 1916.) (Gift of Sir Robert Hadfield.)

Trade Catalogs

CHICAGO PNEUMATIC TOOL CO., Chicago-New York. Bull. 34-N. Single compressors steam and power driven.

DENVER FIRE CLAY CO., Denver, Colo. Catalog D. Case Furnaces, 1916.

E. I. DU PONT DE NEMOURS & CO., Wilmington, Del. Du Pont Magazine. Aug., 1916.

PELTON WATER WHEEL CO., New York, N. Y. Bull. No. 9. Pelton-Doble Centrifugal Pumps.

ROBERTSON-CATARACT ELECTRIC CO., Buffalo, N. Y. Illustrated brochure of buildings of company. pp. 21.

SMITH, H. COLLIER, Detroit, Mich. Catalog No. 50. Quickwork machinery.

SPARTA IRON WORKS CO., Sparta, Wis. Sludge bucket. Aug., 1916.

TEXAS CO., New York, N. Y. Lubrication. v. 2, No. 3; v. 3, No. 1–10. Nov., 1914; Nov., 1915. Aug., 1916.

UNDER-FEED STOKER CO., Chicago, Ill. Publicity Magazine. Aug., 1916.

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MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Aug. 1, 1916, to Sept. 10, 1916:

ALLEN, LOUIS M., JR., Chief Mine Sampler, Old Dominion Min. & Smelt. Co., Globe, Ariz.
ATKINSON, CHESTER, Min. Engr., Asst. Supt., Smelt. Dept., Cerro de Pasco Mining Co., La Fundicion, Peru, South America.
BAKER, CLAIR LINCOLN, Genl. Mgr., Mexican Dept., Amer. Smelt. & Ref. Co., 1108 Mills Bldg., El Paso, Tex.
BARROWS, WALTER A., 3d, Asst. Supt., Blast Furnaces, Thomas Iron Co., Hellertown, Pa.
BARTHOLOMEW, GEORGE POMEROY, Asst. Cons. Engr., Guggenheim Bros., 120 Broadway, New York, N. Y.
BILLOW, ELMER E., Pres., National Supply Co., 416 West Grand Ave., Chicago, Ill.
BROWN, SAMUEL ROLLINS, Mill Supt., The Plymouth Cons. Gold Mines, Ltd., Plymouth, Cal.
BUCHANAN, R. H., Chief Engr. of Mines, Delaware & Hudson Co., Scranton, Pa.
BURNS, WILLIAM.....Min. Engr., Phelps, Dodge & Co., Morenci, Ariz.
CAMPBELL, ARTHUR ROY, Genl. Supt., Zinc Dept., United States Smelt. Co., 413 Republic Bldg., Kansas City, Mo.
CARLISLE, GEORGE LISTER, JR., Mining.....14 Wall St., New York, N. Y.
CHAPMAN, RAYMOND M., Chemist and Met., Owner, Indiana Laboratories Co., Ruff Building, Hammond, Ind.
CLEVELAND, CURTIS, Chemist, International Lead Ref. Co., East Chicago, Ind.
CODY, BENJAMIN H., Chief Chem., The Arizona Copper Co., Ltd., P. O. Box 1003, Clifton, Ariz.
COE, HARRISON STREETER, Min. Engr., The Dorr Company, 810 Cooper Bldg., Denver, Colo.
COLE, CARL H., Asst. Met., Smelter Dept., Calumet & Arizona Mining Co., Douglas, Ariz.
CONNER, WILLIAM S., Min. Engr., Braden Copper Co., Rancagua, Chile, South America.
COOKE, COSBY CRITTENDEN, Supt. of Coal Mines, The Low Moor Iron Co. of Va., Kay Moor, W. Va.
CORDES, FRANK.....Prest., Hubbard Steel Foundry Co., East Chicago, Ind.
CRAWFORD, WILLIAM ELMER, Mill Supt., Compania Beneficiadora de Pachuca, S. A., Apt. No. 1, Pachuca, Hidalgo, Mexico.
CRERAR, GEORGE.....Hammond Mining Company, Glacier, Wash.
CULLEN, JOSEPH F., Research Chemist, United States Smelt. Co., Midvale, Utah.
DEUSSEN, ALEXANDER, Geol., Humble Associates—Oil Producers, Room 504, Stewart Bldg., Houston, Tex.
DICK, JAMES E.....Mgr., Akron Mine, Whitepine, Colo.
DICKSON, ROBERT HENRY, Min. Engr., Calumet and Arizona Min. Co., Bisbee, Ariz.
DUNCAN, HAROLD S., Min. Engr., Old Dominion Copper Mining & Smelting Co., P. O. Box 815, Globe, Ariz.
EATON, ARTHUR.....Cons. Geol., 1641 Euclid Ave., Berkeley, Cal.
EDGAR, SELWYN C., JR., Pres., Edgar Zinc Co., 910 Boatmen's Bank Bldg., St. Louis, Mo.
EDWARDS, HUGH ROBERT.....Cons. Engr., 433 California St., San Francisco, Cal.
FOLLOWILL, DEXTER BEMIS, Supt. of Sulphate Dept., St. Louis Smelt. & Ref. Co., Collinsville, Ill.
FRY, CARL HERBERT, Supt., Yellow Aster Mining & Milling Co., Randsburg, Kern Co., Cal.
GRENFELL, DONALD S.....Chemical Engr., Mineral Point Zinc Co., Depue, Ill.
GRUGAN, JUSTICE, Min. Engr., Mgr., Northern Ore Co., Edwards, St. Lawrence Co., N. Y.
GUYER, RAYMOND, Mining and examination, Guyer Hot Springs, Ketchum, Idaho.
HAMM, KARL G. C.....Chemist, Arizona Copper Co., Ltd., Clifton, Ariz.

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HAMPTON, FRANK HOWARD FRANKLIN, Engr., Braden Copper Co., Rancagua, Chile, South America.
HARTMANN, MARIANO R., Owner and Mgr., Challacollo Silver Lixiviation Plant, and Mines, Cerro Gordo, La Granga, Prov. de Tarapaca, via Iquique, Chile, South America.
HAY, RUGER W.....Min. Engr., Calumet & Arizona Min. Co., Warren, Ariz.
HENDRICKS, W. HOMER, Genl. Sales Engr., New Jersey Zinc Co., 55 Wall St., New York, N. Y.
HYSLOP, JOHN M.....Met., Hubbard Steel Foundry, East Chicago, Ind.
JOHNSTONE, JAMES O., Chemist, International Lead Refining Co., 151st & McCook Sts., East Chicago, Ind.
KIRBY, ALFRED GEORGE, Min. & Met. Engr., 121 Howard Park Ave., Toronto, Ont., Canada.
KIRK, MILTON D., Asst. to Prest., The Davis Coal & Coke Co., Cumberland, Md.
KOBBE, WILLIAM H., Petroleum Engr., Pierce Oil Corpn., 420 Olive St., St. Louis, Mo.
LEMKE, CARL A., Asst. Supt. in Charge of Concentrating Dept., U. S. Smelt. Co., Midvale, Utah.
LINDMUELLER, CHARLES, Supt., Goldschmidt Detinning Co., East Chicago, Ind.
LLEWELLYN, PAUL, Works Mgr., Interstate Iron & Steel Co., East Chicago, Ind.
LOVE, ROBERT G., District Min. Engr., Delaware, Lackawanna and Western R. R. Co., Scranton, Pa.
McRAE, AUSTIN LEE .....Director, Missouri School of Mines, Rolla, Mo.
MACMICHAEL, HUGH RENWICK, Chief Engr., Southern Dept., American Smelt. & Ref. Co., 1108 Mills Bldg., El Paso, Tex.
MARSHALL, ALEXANDER FULTON, Chief Engr., Red Jacket Cons. Coal & Coke Co., Red Jacket, W. Va.
MARTINEZ, FIDEL C.....Asst. Met. Engr., Chino Copper Co., Hurley, N. Mex.
MINOR, CYRUS EDWARD.....Engr., Mining Associates, Ltd., Bonanza, Colo.
MORGAN, DAVID.....Supt., United Verde Extension Mining Co., Jerome, Ariz.
MORSE, GILBERT LIVINGSTON, Asst. to Mine Captain, New Jersey Zinc Co., Franklin, N. J.
MOTHERWELL, WILLIAM, Met..... Colorado Springs, Colo.
OLMSTEAD, SEYMOUR G,..Chief Sampler, Utah Copper Co., Bingham Canyon, Utah.
OSBORN, WALTER X., Cons. Engr., United Verde Extension Min. Co., Douglas, Ariz.
PORTER, RUDOLPH.....Mgr., Federal Lead Co., Federal, Ill.
PROCTOR, ISRAEL O.....Min. Engr., 1450 Dewey Ave., Butte, Mont.
RIDER, EZRA B., Min. Engr., Copper Queen Cons. Mining Co., P. O. Box 1224, Bisbee, Ariz.
ROBERTS, T. C., Chief Engr.....United Verde Copper Co., Jerome, Ariz.
ROBERTSON, JOHN H., Min. Engr.....Box 1405, Miami, Ariz.
ROSS, CLYDE POLHEMUS, Min. Geol., Asst., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
SANDFORD, JOHN J., Engr., Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
SAUNDERS, HOWARD P., Chief Engr. and Supt. Maintenance and Construction, St. Joseph Lead Co., Herculaneum, Mo.
SEAMON, WILLIAM H., JR., Mine Engr., Jos. S. Qualey & Co., Care Harding & McKee, 404 Roberts-Banner Bldg., El Paso, Tex.
SHEFFIELD, EUGENE S., Min. Engr., Departmental Mgr., Care Standard Oil Co. of New York, Producing Dept., Peking, China.
SPENCER, FRANK NORTON, Asst. to Genl. Mgr., The New Jersey Zinc Co., 55 Wall St., New York, N. Y.
SPRAGUE, CHARLES S., Pres. and Genl. Mgr., Jumbo Extension Mining Co., Goldfield, Nev.
STEWART, LEIGHTON, Min. Engr., Care Hayden Stone & Co., 25 Broad St., New York, N. Y.
VARLEY, THOMAS..... Federal Lead Co., Flat River, Mo.
WAHL, ROY LESSLIE, Chief Min. Engr.....Inland Steel Co., Crosby, Minn.
WALKER, HENRY Y., Mgr.....Tacoma Smelt. Co., Tacoma, Wash.
WAUCHOPE, ALBERT, Mine Supt., (Gold) Sons of Gwalia, Ltd., Gwalia, Western Australia.
WEBSTER, ERNEST B., Engr., with Highway Commission, Regina, Sask., Canada.
WEMPLE, LELAND E., MET.....American Zinc Lead & Smelt. Co., Hillsboro, Ill.
WYSOR, DAVIDSON CHARLTON, Geol., General Chemical Co., 25 Broad St., New York, N. Y.
ZIESEMER, HARRY M., Min. Engr., Copper Queen Cons. Min. Co., Bisbee, Ariz.

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Associate Member

COOMBE, A. PALMER, Care The Standard Oil Co., East Ohio Gas Bldg., Cleveland, O.

Junior Members

BRENNEN, GEORGE K. ..... Scottdale, Pa.
HARROD, WAYNE A., Flotation Operator, Engels Copper Mining Co., Keddie, Cal.
HIRAOKA, MICHIYA, Prof. of Min. Engrg. & Met., Osaka Higher Technical College, Osaka, Japan.
HOLMES, GEORGE F., Student ..... Michigan College of Mines, Houghton, Mich.
KOMAR-OF-SKYE, GEORGE, Min. Engr., Care American Institute of Mining Engineers, 29 W. 39th St., New York, N. Y.
KUPFERSTEIN, JOSEPH T. ..... Care Producers Oil Co., Tulsa, Okla.
LAU, KAAN ..... 201 Bryant Ave., Ithaca, N. Y.
MCDONALD, CARLTON K., Laborer, American Smelt. & Ref. Co., Omaha Plant, Omaha, Nebr.
OVENS, JAMES MASON, Student, Michigan College of Mines, Houghton, Mich.
REPETTI, GEORGE W., Min. & Met. Engr., The Dorr Company, 812 Cooper Bldg., Denver, Colo.
SCHOLL, LOUIS A., JR., Geol. and Min. Engr., Producers Oil Co., Mayo Bldg., Tulsa, Okla.
SPURNY, EMIL ..... 802 Second Ave., Astoria, L. I.
STICKNEY, WILLIAM HERZOG, Miner, Tramway Mine, Anaconda Mining Co., Butte, Mont.
WILSON, HARRY R., Chemist, Care Burma Mines, Ltd., Nam Tu, Northern Shan States, Burma.
WOO, WAI KYI ..... 120 East St., Houghton, Mich.
YOST, HAROLD W., Min. Engr., C. H. Fulton, Experimental Zinc Smelter, Boatmen's Bank Bldg., St. Louis, Mo.

Total Membership, Sept. 10, 1916 5,748

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period Aug. 1, 1916, to Sept. 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Arthur Abraham Arluck, Cleveland, O.
Proposed by J. Burns Read, F. A. Fahrenwald, Frank R. Van Horn.
Born 1886, Russia. 1907, Cleveland College Preparatory School. 1909, Spencerian Business College. 1913, Case School of Applied Science. 1913–14, Miner, Anaconda Copper Min. Co., North Butte Min. Co., Butte, Mont. 1914–15, Leaser,

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Miner, Easton Pacific Mine, Virginia City, Mont. 1915–16, Const. Engr. and Surface Foreman, Cons. Interstate-Callahan Min. Co., Wallace, Idaho.
Present position: City Water Works Tunnel, Cleveland, O.

Theodore Newton Barnsdall, 2d, Bradford, Pa.
Proposed by W. L. Bell, Frank A. Ross, L. K. Armstrong.
Born, 1878, Titusville, Pa. Public Schools in Titusville, Pa., and Bradford, Pa. 1899–1912, Director, Manufacturers Gas Co., Bradford, Pa. 1906–08, Vice-Pres., Ardizzone Construction Co. actively engaged laying gas lines in Kansas. 1908–12, Pres., Barnsdall Construction Co. laying gas lines in Ohio, Kansas and Okla. 1910 to date, Treas., Highland Gas Co., Bradford, Pa. 1909 to date, Director, Potter Gas Co., Pittsburgh, Pa. 1910 to date, Oil Producer in Pennsylvania and Oklahoma.
Present position: Treas. and Genl. Mgr., Highland Gas Co.

Carl H. Beal, Muskogee, Okla.
Proposed by W. R. Hamilton, W. A. Williams, J. C. Branner.
Born 1889, Western Kansas. 1905–09, High School, Ukiah, Cal., and Palo Alto, Cal., 1909–12, Leland Stanford Jr. University, A. B. in Geology. 1914–15, Leland Stanford Jr. University, A. M. Geology. 1913, Asst. Geol., Kern Trading & Oil Co., Coalinga. 1913–14, Petroleum Geol., with Harry R. Johnson, Los Angeles, Cal., making geology reports on oil properties in developed and prospective fields of California, Montana, British Columbia and Alberta. 1914–15, Asst. Geol. and Mining, Stanford University, and during summer of 1915 in charge field work of summer class. 1915–16, Oil and Gas Inspector, U. S. Bureau of Mines, Muskogee, Okla.
Present position: Petroleum Technologist, U. S. Bureau of Mines, Washington, D. C.

Paul Thomas Benson, Thane, Alaska.
Proposed by G. T. Jackson, Joseph Daniels, E. V. Daveler.
Born 1889, Pueblo, Colo. 1908–11, Colorado School of Mines. 1911–12, Univ. of Washington, B. S. in Min. Engrg. 1912–13, Assayer and Asst. Chemist, Tacoma Smelt. Co. 1913–14, Assayer, Alaska Gastineau Min. Co., Juneau, Alaska. 1915, Foreman of Roll Dept., Alaska Gastineau Min. Co., Thane, Alaska. 1915–16, Foreman of Concentrating Dept., Alaska Gastineau Min. Co., Thane, Alaska.
Present position: Foreman of Concentrating Dept., Alaska Gastineau Min. Co.

Thomas Townsend Brewster, St. Louis, Mo.
Proposed by B. F. Bush, W. J. Jenkins, H. A. Wheeler.
Born, 1867, Saco, Me. Attended Common School. Experience and study. 1882–83, Blair & Truesdell, Bankers, Syracuse. 1883–86, Salt Springs National Bank, Syracuse. 1886–88, Acting Treas., Sweets Mfg. Co. (Rolling Mills), Syracuse. 1888–96, in business for myself, promoting, representing security holders in various corporations, specializing in coal propositions from 1890. 1896–98, Mgr., Thomas Carmichael, Trustee, Rio Grande Coal & Irrigation Co., Laredo, Texas. 1898–1912, Vice Pres. and Genl. Mgr., Rio Grande Coal Co. a reorganization of next above, property was sold in 1912. 1902 to date, Vice-Pres. and Genl. Mgr., Mt. Olive & Staunton Coal Co. 1910–12, Vice-Pres., Coal Operators' Assn. 5th and 9th Dist. of Illinois. 1912 to date; Pres., Coal Operators' Assn.
Present position: Vice-Pres. and Genl. Mgr., Mt. Olive & Staunton Coal Co.

Cecil John Brown, Wallaroo, South Australia.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1888, Moonta, South Australia. 1893–1901, State Public School. 1901–03, District High School. 1903–09, Technical, Mechanical Draftsman, Survey & Chemical Courses, Mechanical and Electrical Engineering. 1910–15, Civ. Engineering Diploma, International Correspondence Schools. 1903–05, Junior Draftsman's Dept., Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1905–08, Mining Engineering Draftsman (Wallaroo Mines) Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1908–10, Surveyor, Surface & Underground (Wallaroo Mines) Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1910–16, Engineering Draftsman & Surveyor (Wallaroo Smelt. Works) Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1907–16, Instructor, Art & Science Subjects, Moonta School of Mines, evening classes.
Present position: Chief Draftsman and Surveyor, Wallaroo Smelt. Works.

Godfrey L. Cabot, Boston, Mass.
Proposed by R. B. Woodworth, John Hays Hammond, I. C. White.
Born 1861, Boston, Mass. 1882, Harvard College, A. B. Magna Cum Laude in Chemistry. Studied in Zurich in 1883–84 with Dr. George Lunge and Prof. Victor

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Meyer. Took graduate course in Harvard. 1882, Employed by Samuel Cabot, Jr. 1883, in experimental work in Natural Gas. 1886, in partnership with L. Cabot. 1886–87, made gas regulators. From then to present time busied in selling gas, building carbon black factories and operating them. Owning, operating, drilling gas and oil wells, laying pipe lines distributing and selling gas on an increasing scale, and am now busied also in liquefying Natural Gas.

Present position: Owner of five carbon black plants. Controlling owner of two others. Pres., Liquid Fuel & Gas Co.

Harry Elmer Bullock, Hazard, Ky.

Proposed by O. G. Petersen, William H. Shearman, Thomas T. Read.

Born 1883, Packard, Ky. 1903, educated at Lincoln Memorial Univ., Cumberland Gap, Tenn. 1904, Brown Univ., Providence, R. I. Made specialty of chemistry and mining engineering. No degree. 1906, Asst. to Genl. Mgr., Old Sterling Iron & Min. Co., Antwerp, N. Y. 1907, Supt., Bennett Jellico Coal Co., Trosper, Ky. 1908, Pres. and Genl. Mgr., Trosper Coal Co., Trosper, Ky. 1912, Mgr., Poteau Coal and Mercantile Co., Poteau, Okla. Also Genl. Mgr., Fort Smith Poteau and Western R. R.

Present position: 1914 to date; Pres. and Genl. Mgr., Kentucky Jewel Coal Co. and Kentucky Block Coal Co.

Norman L. Calder, Matahambre, Pinar del Rio, Cuba.

Proposed by J. Mark Smith, J. H. Jowett, L. D. Albin.

Born 1887, Hobart, Tasmania. 1906, Univ. of Tasmania. 1909–11, 1913–15, Columbia Univ., M. E. 1902–06, Tasmanian Smelt. Co., Tasmania. 1907–09, Transvaal Bischoff Tin Mine, Transvaal. 1911–12, Nevada Cons., McGill, Nev. 1912–13, Goldfield Cons., Goldfield, Nev. 1912–13, Ray Cons., Ray, Ariz. 1915, Minas de Matahambre, Matahambre, Cuba.

Present position: Asst. Engr., Minas de Matahambre.

Everett Carpenter, Bartlesville, Okla.

Proposed by Alfred J. Diescher, J. C. McDowell, Henry L. Doherty.

Born 1884, Holton, Kans. 1911, University of Oklahoma, A. B. 1908 and 1909, Asst. Geol., Oklahoma Geol. Survey. 1910, Junior Geol., U. S. Geol. Survey. 1911–16, Chief Geol., Wichita Natural Gas Co. and allied concerns.

Present position: Chief Geol., Empire Gas & Fuel Co.

Cairy Clyde Conover, Springfield, Ill.

Proposed by G. H. Cox, D. H. Radcliffe, M. M. Valerius.

Born 1890, Emporia, Kans. 1896–1908, Grammar School and High School, Carrollton, Mo. 1908–12, Missouri School of Mines and Metallurgy, Rolla, Mo., B. S. 1912, Mill man in wet mill and magnetic mill, Vinegar Hill Zinc Co., Platteville, Wis. 1913, Chemist, Chief Chemist, and later Asst. Supt., National Zinc Co., Springfield, Ill.

Present position: Asst. Supt., National Zinc Co.

Ralph Augustus Conrads, Garfield, Utah.

Proposed by R. C. Gemmell, H. C. Goodrich, R. H. Hawley.

Born 1880, Trenton, Mo. 1897, Grad., High School, Trenton, Mo. 1904, Grad., Missouri School of Mines, B. S. in Min. Engrg. 1902, Rodman, on R. R. Construction for Arkansas & Choctaw Ry. in Indian Territory. 1904–05, Concentration and Assaying, Utah Copper Co., Bingham Canyon, Utah. 1905–06, Engr., Balaklala Cons. Copper Co., Shasta Co., Cal. 1906–07, Engrg. and Cyanide mill, The Annie Laurie Gold Min. Co., Kimberly, Utah. 1907, Engr., Cia. Carbonifera Agujita, of South America, Sabinas, Coah., Mex. 1907–11, Engr., for 8 months, remainder of period, Asst. Mgr., Esperanza Min. Co., El Oro, Mex. 1911–12, Mining in interest of myself and associates in Ter. de Tepic and Oaxaca, Mex. 1912–13, Asst. Mgr., Cia. Minera Las Dos Estrellas, El Oro, Mex. 1913–14, Mgr., Cia. Minera Santa Ana Esperanza, Zac., Mex.

Present position: Met. Engr., Magna Plant, Utah Copper Co.

John Vipond Davies, New York, N. Y.

Proposed by W. L. Saunders, Alexander C. Humphreys, B. F. Cresson, Jr.

Born 1862, Swansea, South Wales. Wesleyan College, Taunton, England. Matriculated—University of London. 1880–84, Articled Apprentice, Parfitt & Jenkins of Cardiff, South Wales, Contracting Engineers. Principal work, Junction Dry Dock-Dock Gates, Bule Docks, Cardiff—Shaft 2,500 ft. deep, 25 ft. dia. Ynysybwl

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Colliery and machine shop practice. 1884–85, Mine Surveyor, Collieries, John Vipond & Co. 1885–88, Asst. Engr. & Chief Asst. Blaenavon Coal & Iron Co., South Wales, in charge erection and construction blast furnaces, stoves, coke ovens, regen. furnaces, rolling and tire mills—Bessemer convertors and steel plants—Limestone quarry operations and brick making plants. 1888–89, Sea-going Engr., S/S "Argus" of Melbourne. 1889–90, Engr. & Mgr., Anthracite Coal Briquetting plant, Mahanoy City, Pa. 1891 to date; Charles M. Jacobs as Cons. Engr. Principal works: East River Gas Tunnel from 70th St., New York to Ravenswood; Chief Engr., designed and built, W. Va. Short Line R. R.; Chief Engr., designed and built, Kanawha & Pocahontas R. R.; Chief Engr., designed and built, Atlantic Ave., Improvement, L. I. R. R.; Chief Engr., designed and built, Entire tunnel and subway system of Hudson & Manhattan R. R., New York and New Jersey. 1906–07, Two years in charge for Contractors of Grand Central Yards Improvements. 1904, Cons. Engr., Board of Water Supply Detroit on Intake Tunnel. Engineers in charge of construction Hales Bar Dam across Tennessee River near Chattanooga involving pneumatic pressure construction for foundations. Engineers in charge of construction of 19 miles Aqueduct Tunnels for Mexico City Tramways Co. to divert waters Laxaxalpam River to Necaxa Dam. Engineer of design and actually in charge Construction Astoria Tunnel under East River, N. Y., for Astoria L. H. & Power Co., Tunnel about 20 ft. dia., 4,700 ft. long, Shaft 250 and 280 ft. deep. Consulting Engineer N. Y. Municipal Rw. Co. (B. R. T.) on Subway Railway developments in City of New York. Member Engineer Board Moffat Tunnel Commission, Colorado, reporting on Moffat Tunnel through Rocky Mountain Divide (other members, D. W. Brunton & Finch).

Present position: Vice-pres., Jacobs & Davies, Inc.

Graham Barclay Dennis, Spokane, Wash.

Proposed by L. K. Armstrong, Charles H. Goodsell, F. A. Ross.

Born 1856, Southampton, England. 1865–75, Cincinnati Public Schools. Cincinnati Pharmacy School, Bethany College, West Va. 1875–78, Asso. Editor, Business Mgr., Journal, Dayton, O. 1879–80, Editor and Publisher, Farmers' Home, Dayton, O. 1880–85, Investment Bankers, G. B. Dennis & Co., Dayton, O. 1885, Editor and Publisher, Spokane Miner, Spokane, Wash. 1886–88, Pres. and Mgr., Muscovita Mica Co., Spokane, Wash. 1888–91, Pres. and Mgr., Ross Park Elec. Ry. Co., Spokane, Wash. 1891–1916, Pres., Old Dominion Min. & Mill. Co., Spokane, Wash. 1893–96, Pres., Northwest Min. Asso., Spokane, Wash. 1899–1916, Pres. and Mgr., Insurgent Gold Min. Co., Spokane, Wash. 1908–16, Pres., Warehouse & Realty Co., Spokane, Wash. 1916, Pres., Spokane Mining Mens' Club, Spokane, Wash. 1916, Secy. & Treas., Spokane Railway & Terminal Co., Spokane, Wash. 1890, Vice-Pres., Northwestern Industrial Exp. Co., Spokane, Wash. 1906–16, Director, Exchange National Bank, Spokane, Wash. 1912–13, Vice-Pres. for Washington, American Mining Congress, Spokane, Wash. Have held like positions, within the years named, and many others of responsibility and trust. Trustee and Treas., Jenkins Univ., Spokane, Wash. Member, Com. Revision Federal Mining Laws. Organizer and Director and Pres., Spokane Publicity Bureau, for the years 1904–08. Some experience and Real Estate.

Present position: A number of above, and Pres., Spokane Mining Mens' Club.

Alfred F. Duggleby, Santiago de Cuba, Cuba.

Proposed by Max Roesler, S. B. Patterson, Jr., J. T. Singewald, Jr.

Born 1893, Brisbane, Aust. 1915, Colorado School of Mines, E. M. 1910–11, Draughtsman, American Pneumatic Action Co., Davenport, Ia. 1911, Transitman, A. R. Boudinot, Davenport, Ia. 1912, Miner, with E. P. Arthur, Jr., Cripple Creek, Colo. 1913, London Mines Co., Apex, Colo. 1914, Transitman, City Eng. Office, Davenport, Ia. 1915, Research in concentration Titanium Ores, Titanium Alloys Co. in connection with F. W. Traphagen, Golden, Colo. 1915–16, Mgr., Gold Reef Mines Co.

Present position: Mine Supt., Juragua Iron Co.

David L. Dunn, Joplin, Mo.

Proposed by H. A. Buehler, G. B. Corless, A. F. Truex.

Born 1881, Braidwood, Ill. 1897, Grad., High School, WeirCity, Kans. 1902–05, Asst. Engr., Central Coal & Coke Co., Weir City, Kans. 1905–06, Resident Engr., Central Coal & Coke Co., Bevier, Mo. 1907–09, Resident Engr., C. E. & C. Co., Huntington, Ark. 1910–11, Engr., and Asst. Supt., C. C. & C. Co., Rock Springs, Wyo. 1912, Asst. Chief Engr., C. C. & C. Co., Kansas City, Mo. 1913, Resident

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Engr., C. C. & C. Co., Pittsburg, Kans. 1913–15, Supt., C. C. & C. Co. Mines, Pittsburg, Kans.

Present position: Stockholder and Manager, Spengeon Tiger M. Co. and Bumble Bee M. Co.

Harold Clinton Eddy, San Francisco, Cal.

Proposed by Arthur F. L. Bell, Albert Burch, M. E. Lombardi.

Born 1882, Middleboro, Mass. 1902–04, Colorado School of Mines. 1905, Mass. Inst. of Technology. 1907–08, Colorado School of Mines, E. M. 1909–10, Supt. Nevada Belle Helen Mines Co. 1910–11, Supt., Orleans Petroleum Co. 1911–13, Victor Electric Co.

Present position: 1913 to date; Engr., Petroleum Rectifying Co. of California.

Jay L. Emrich, Durango, Colo.

Proposed by F. C. Gilbert, L. R. Clapp, P. A. Mosman.

Born 1888, New York, N. Y. 1908–12, Colorado School of Mines, E. M. 1912, San Pedro Gold and Copper Min. Co., San Pedro, N. Mex. 1913, American Smelt. & Ref. Co., Leadville, Colo. 1913–16, American Smelt. & Ref. Co., Denver, Colo.

Present position: Chief Chemist, American Smelt. & Ref. Co.

Fred A. Fair, Boulder, Colo.

Proposed by M. S. Brandt, R. D. George, George W. Teal.

Born 1878, Ottumwa, Ia. 1898, Grad., East Denver High School. 1899–1902, School of Mines, Golden. 1905, classified minerals, as assistant, Univ. of Colo., Boulder. 1903–04, Asst. Engr., George Evans Gold Pan Placer, Breckenridge, Colo. 1905, U. S. Deputy Mineral Surveyor, Colo. 1906, Charge of construction city water system of the City of Boulder $500,000 plant. 1907–08, City Engr., City of Boulder Colo. 1909–10, County Surveyor, Boulder Co., Colo. 1911, Engr., Water users District #6, Colo. 1912, Engr., Boston Colorado Power Co., Colo. 1913, Engr., Welington Mines, Nevada. 1914, Engr., Nona Mines, Leadville. 1915, Engr., Yellow Pine and Logan Mines, Boulder. 1916, Engr., Chief Engr. for Boulder Tungsten Production Co. Cons. Engr., Tungsten mountain mine, Logan. Yellow Pine and Nona group. Also County Surveyor, Boulder County and U. S. Mineral Surveyor.

Present position: Pres. and Genl. Mgr., The Fred A. Fair Engrg. Assn.

T. H. Powers Farr, Jr., Jackson, Cal.

Proposed by Bradley Stoughton, Benjamin B. Lawrence, Charles F. Rand.

Born 1885, Orange, N. J. 1903–07, Princeton Univ., A. B. 1907–09, Columbia School of Mines. 1909, Aug. and Sept., American Smelters Securities Co., Velardina, Mexico. 1909–10, Engr., American Smelters Securities Co., Matehuala, San Luis Potosi. 1911–14, Examination and development work for private individuals in Ecuador and Southern Colombia, South America.

Present position: Argonaut Mining Co.

John Brainard Fidlar, Tulsa, Okla.

Proposed by H. F. Wright, W. E. Hopper, Irving Perrine.

Born 1892, Davenport, Iowa. 1916, Iowa State College, B. S. Besides the four-year course in mining engineering, I received a year of graduate work in geology but did not write a thesis necessary for a Master's degree. 1915, Iowa Geological Survey.

Present position: Geol., Gypsy Oil Co., working in the Oklahoma fields.

Royce Albert Field, Newcastle, N. S. W., Aust.

Proposed by David Baker, G. D. Delprat, E. J. Horwood.

Born 1889, Lebanon, Pa. Public Schools, Lebanon, Pa. 1905–09, Lebanon High School. 1909–13, Pennsylvania State College, B. S. in Engrg. 1908–12, Summer work, Lebanon Furnaces, Pennsylvania Steel Co. 1913–15, Blast Furnace Dept., Youngstown Sheet & Tube Co.

Present position: 1915 to date; Furnace Foreman, Blast Furnace Dept., Broken Hill Proprietary Co., Ltd.

Martin Fishback, El Paso, Texas.

Proposed by Kirby Thomas, W. G. Swart, D. W. Shanks.

Born 1870, Gudum, Struer, Denmark. 1876–90, Grade and High Schools in Denmark and special instruction in polytechnics. 1893–96, worked in the mines in Colorado while making a special study of geology and mineralogy. 1896–99, Asst. Mgr., Modoc Cons. Gold Min. Co., Boulder Co., Colo. 1899–1900, operated in Crip-

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ple Creek, Colo. 1900–10, Did field and examination work, Mexico for French and British clients principally; among those were the Tharsis Sulphur & Copper Co., Glasgow. 1910–12, examination and field work in New Mexico and Arizona. 1912–13, operated Excelsior Copper Mine, Organ, N. Mexico. 1913–15, examination and field work in Arizona and California.

Present position: 1915 to date; Mgr., Merrimac Mine, Organ, N. Mex.

Howard A. Fitch, Kansas City, Mo.

Proposed by L. D. Ricketts, David Cole, A. G. McGregor.

Born 1868, Missouri. 1889, Finished Sophomore year, De Paw University, Greencastle, Ind. 1890–96, Structural Engr., Gillette-Herzog Mfg. Co., Minneapolis, Minn. 1897–1900, Contracting Engr., designing and selling steel structures for mining and smelting purposes, Gillette-Herzog Mfg. Co., office at Butte, Mont. 1900–02, Contracting Engr., American Bridge Co., Salt Lake City, Utah, same class of work as above. 1902–07, Chief Engr., Minneapolis Steel and Machinery Co., Minneapolis, Minn.

Present position: 1907 to date; Pres., Kansas City Structural Steel Co. Manufacturing steel structures for mining, milling and smelting purposes.

Giles Morton Fritch, Waterbury, Conn.

Proposed by William H. Bassett, George C. Stone, William H. Shearman.

Born 1887, Wichita, Kans. 1894–1905, Grade Schools and High School, St. Louis, Mo. 1905–07, 1911–13, Univ. of Michigan, B. S., Met. Chemistry. 1907–11, Salesman, New St. Louis Business College, Southwestern Business College, American Real Estate Co. of N. Y.; Dist. Sales Mgr., Terry-Swain Publishing Co. 1912–13, Teaching Asst. in General Chemistry of University of Michigan.

Present position: Asst. Met., American Brass Co.

Aubrey Hamilton Garner, Astoria, Ore.

Proposed by Ralph Arnold, Marion L. Thomas, A. Faison Dixon.

Born 1889, Ishpeming, Mich. 1912, Leland Stanford Jr. Univ., B. A. 1912 to date; 2 years as geologist, 1 year as field manager of an oil district, The Caribbean Petroleum Co., Venezuela.

Present position: Field Manager, The Caribbean Petroleum Co.

Earl Graham Gaylord, San Francisco, Cal.

Proposed by M. E. Lombardi, E. T. Dumble, J. A. Taff.

Born 1888, Redland, Cal. 1906, Grad., High School, Redlands, Cal. 1907–11, Univ. of Cal., B. S. 1906–07, Engrg. Dept., City of Los Angeles. 1911–14, Field Geol., Southern Pacific Co. 1914–16, Geol., Kern Trading and Oil Co.

Present position: Geol., Kern Trading and Oil Co.

Martin Butler Gentry, Santiago, Chile, South America.

Proposed by William Braden, B. B. Thayer, George E. Montandon.

Born 1886, Kansas City, Mo. 1903–06, Sheffield Scientific School, Yale Univ., Ph. B. 1906–07, School of Mines, Columbia University. 1908–09, School of Mines, Columbia University, E. M. 1908, Mucker, miner, timber man, Tombstone Cons. Mines Co., Tombstone, Ariz.; Miner and shift boss, Imperial Copper Co., Silverbell, Ariz. 1910–11, Shift boss, engineer in charge of porphyry development and drilling, Imperial Copper Co., Silverbell, Ariz. 1911, sampling department, Nevada Cons. Copper Co. mill. 1912, Drilling examination in Chile for William Braden and M. Guggenheim Sons Cons. 1912–15, Drilling examination and development, Chile Exploration Co., Chuquicamata, Chile. Scout examinations in Chile, Bolivia and Peru for Chile Exploration Co. 1914, Scouting in Northern Brazil. 1914–15, personal interests coal mining in Arkansas.

Present position: Scouting and examination, Andes Exploration Co.

Philip Raymond Gleason, Cerro de Pasco, Peru, South America.

Proposed by W. J. Turner, W. S. Bishop, R. K. Stockwell.

Born 1891, New Canaan, Conn. 1897–1904, Primary education, New Canaan, Grammar School and Franklin Grammar School, South Norwalk, Conn. 1904–09, High School training at Stamford, Conn. 1909–12, General Met. Engrg., Sheffield Scientific School, Yale Univ., Ph. B. 1912–14, 1 year as Chemist, 1 year as Blast Furnace Foreman, Braden Copper Co., Rancagua, Chile.

Present position: 1914 to date; Furnace Foreman, Braden Copper Co.

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Alfonce Henry Heller, Keddie, Cal.
Proposed by Thomas T. Read, Donald Dyrenforth, E. R. Ramsey.
Born 1892, Newman, Cal. Public Schools, Santa Maria, Cal. 1906–07, Gymnasium—Ulelsass, Germany. 1907–11, Santa Maria High School. 1912–13, Colorado School of Mines, full credits. 1913–14, Univ. of Cal., full credits. High School vacation periods, oil field work. 1912, Asst. to Supt., Bradley Canyon Oil Co., Santa Maria, Cal. 1914–15, Bayer Chemical Co., San Francisco, Cal. 1915–16, Flotation operator, Engels Copper Min. Co., Keddie, Cal.
Present position: Asst. to Mill Supt., Engels Copper Min. Co.

Emil Reinhard Reo Hennebach, Garfield, Utah.
Proposed by Guy M. Kerr, R. W. Sengler, F. K. Brunton.
Born 1887, Leipzig, Germany. 1912, Grad., University of California, Berkeley, Cal., B. S. 1912 to date; Foreman, Chemist and Met., American Smelt. & Ref. Co.
Present position: In charge of Sampling Dept., Garfield Smelt. Co.

Arthur Houle, Bisbee, Ariz.
Proposed by Arthur Notman, Y. S. Bonillas, Gerald Sherman.
Born 1877, Negaunee, Mich. 1899, Grad., Michigan College of Mines, B. S. and E. M. 1899–1902, Chemist and later Met., Old Dominion Copper Min. & Smelt. Co. 1902–04, Supt., Emcampment Smelt. Co. 1904–07, Met., Calumet & Arizona Min. Co. 1908–12, Cons. Engr.
Present position: 1912 to date; Supt., Shattuck-Arizona Copper Co.

Charles Taylor Jobes, Gila Bend, Ariz.
Proposed by Henry A. Tobelmann, J. Owen Ambler, Carl H. Cole.
Born 1886, Kosciusko, Miss. 1906–09, Missouri School of Mines and Metallurgy, Rolla, Mo. 1909–10, New Mexico School of Mines, Socorro, N. Mex. 1906, Utah Copper Co. 1907, Chief Engr., Lucky Tiger Gold Min. Co., Kansas City, Mo. 1910, Supt., Gold Dollar Mines, Sierra Co., N. Mex. 1911, Universal Smelt. & Ref. Co. 1912, Asst. Engr., Phelps, Dodge & Co., Nacozari, Son., Mex. 1913–14, Private Mining Engineering practice.
Present position: 1915 to date; Chief Engr., Rowley Copper Mines Co.

G. J. Kapteyn, Breckenridge, Colo.
Proposed by P. M. McHugh, Charles A. Chase, A. L. Blomfield.
Born 1883, Groningen, Holland. 1907, Freiberg, Saxony, Civ. Engr. and Min. Engr. 1908–10, Asst. Supt., Stratton Independence, Ltd., Victor, Colo. 1910–11, Cyanide Foreman, Camp Bird, Ltd., Ouray, Colo. 1911, Engr., Crown Chartered, Ltd., Porcupine, Ont. 1912, Engr., Swastika Min. Co., Swastika, Ont., Engr., Rheingold Min. Co., New Year, Mont. 1913, Mill foreman, Beck Min. Co., Atlantic City, Wyo., Mgr., Linda Ventura Mines Corpn., Wawa, Nicaragua. 1915, Mill Supt., Bar Principal Min. Co., Colombia.
Present position: Mgr., Jesse Cons. Mines.

Burwell Newton Kilbourn, Murray, Utah.
Proposed by A. H. Richards, C. A. Adams, Jr., A. L. Labbe.
Born 1891, Pueblo, Colo. 1909, Grad., Centennial High School, Pueblo, Colo. 1913, Colorado State School of Mines, Golden, Colo., E. M. 1913–14, Asst. in experimental works, International Smelt. & Ref. Co., Tooele, Utah. 1914 to date; Asst. Chemist, Asst. Assayer, Night Supt., American Smelt. & Ref. Co., Murray Plant, Murray, Utah.
Present position: Night Supt., Murray Plant, American Smelt. & Ref. Co.

Carl H. Kithil, Denver, Colo.
Proposed by David T. Day, Albert H. Fay, Max W. Ball.
Born 1878, Germany. Studied in Nürnberg and Munich. Mining engineer making specialty of the economics of graphite and monzite and other rare minerals. Cons. Engineer in Brasil and Africa for the Berlin Rare Minerals Co. and subsequently managing director of that corporation.
Present position: Mineral Technologist, Bureau of Mines.

Paul Henry Kleinhammer, Webb City, Mo.
Proposed by H. A. Buehler, A. F. Truex, W. C. Hogoboom.
Born 1890, Platteville, Wis. 1905, Graded Schools, Platteville, Wis. 1905–11, Northwestern College, Watertown, Wis. 1913–14, Wisconsin School of Mines, Platteville, Wis. 1913, Surveyor's Helper, Wisconsin Zinc Co., Platteville, Wis. 1913,

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underground experience, Vinegar Hill Zinc Co. 1914 to date, underground and surface workings, Century Zinc Co., Joplin, Mo.
Present position: Asst. Supt., Century Zinc Co.

J. Gary Lindley, Douglas, Ariz.
Proposed by J. Owen Ambler, H. A. Clark, Carl H. Cole.
Born 1888, Moberly, Mo. 1903–06, Moberly High School, Moberly, Mo. 1906–07, Hemet High School, Hemet, Cal. 1908–09, Univ. of Oregon, Eugene, Ore. 1910–13, Univ. of Arizona, Tucson, Ariz., B. S. in Metallurgy. 1910, summer, Chainman, U. S. General Law Office Survey. 1911, summer, Custom Assayer, E. A. Jacobs, Tucson, Ariz. 1912, summer, Mine Assayer with Fred Tatum, Oro Blanco, Ariz. 1913, Conveyor Foreman, C. & A. Smelter, Douglas, Ariz. 1914, Asst. Chemist, C. & A. Smelter, Douglas, Ariz. 1914–16, Engr., National Mines & Smelters Co., Magistral, Dur., Mexico. 1916, Chemist, Douglas Assay Co., Douglas, Ariz.
Present position: Met. Acct., Calumet and Arizona Smelter.

Thayer Lindsley, New York, N. Y.
Proposed by Robert Livermore, Halstead Lindsley, C. Q. Payne.
Born 1882, Yokohama, Japan. 1895–1900, Milton Academy, Milton, Mass. 1901–04, Harvard, A. B. & S. B. 1904–05, Rodman, Rapid Transit Commission, N. Y. 1905–09, Engaged in scouting, and employed in mines, mills, and cyanide plants in Idaho, Colorado, Nevada and Mexico. 1910–12, Exploration and scouting for mines. 1913, Scouting Engr. for the Tonopah Min. Co. 1914–16, Examination and geological work, Goodrich Lockhart Co., New York.
Present position: Min. Geol., Goodrich Lockhart Co.

Robert Davis Longyear, Minneapolis, Minn.
Proposed by E. J. Longyear, John E. Hodge, W. J. Mead.
Born 1892, Petoskey, Mich. 1910–14, Williams College, A. B. 1914–15, Univ. of Wisconsin, M. A. 1915–16, Standford Univ. 1912–14, summer, field geology, E. J. Longyear in Michigan. 1915, Geol., E. J. Longyear Co., Sudbury, Ont.
Present position: E. J. Longyear Co.

Alexander Neil Mackay, Barranquilla, Rep. of Colombia.
Proposed by A. L. Blomfield, T. A. Rickard, Arthur W. Allen.
Born 1877, Aberystwith, England. 1892–95, General education at Eastbourne College, Sussex, England. 1895–99, Technical education at The Royal School of Mines, London, England, graduating as Associate R.S.M. in the division of Mining. 1900, Asst. Assayer, Lake View Cons., Ltd., Kalgoorlie, W. Aust. 1902–04, Head Assayer & Cyanide Mgr., Cosmopolitan Proprietary, Ltd., Kookyine, W. Aust. 1905–07, Asst. Sampler, Asst. Survey Office, Head Sampler, successively, on East Rand Proprietary Mines, Ltd., Transvaal, S. Africa. 1908–11, Asst. Mgr., Cherokee Goldfields, Ltd., San Julian, Chih., Mex. 1912, 1915, Surveyor, Underground Mgr., Genl. Supt., Frontino & Bolivia (S. A.) Gold Min. Co., Ltd., La Salada, Antisquia, Rep. of Colombia, S. Amer.
Present position: Genl. Supt., Frontino & Bolivia (S. A.) Gold Min. Co., Ltd.

Bartlett F. Martin, La Harpe, Kans.
Proposed by E. List, H. A. Buehler, Herbert Taylor.
Born 1881, Shelbyville, Ill. 1898, Common schools. 1899–1903, High School, Shelbyville, Ill. 1904–06, Barnes Med., St. Louis, Mo. 1906–08, University of Missouri. 1908–09, Winona Technical Inst. 1909–11, Chemist, Illinois Steel Co. 1911, Met., Black Hills Smelt. Co. 1911–14, Chief Chem., Hegeler Bros. Zinc Co. 1915, Chemist and Met., U. S. Smelt. Co.
Present position: Chief Chemist, La Harpe, Kans., Zinc Plant, U. S. Smelt. Co.

Sylvester Edward Martin, Wallaroo, So. Aust.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1887, Moonta, So. Aust. 1892–1901, Primary & State School Education. 1901–07, Technical, Mechanical and Electrical Engineering, Surveying, Draftsman's and Chemical Courses, Moonta School of Mines, So. Aust. 1910–14, Technical, Mining Engineering Diploma, International Correspondence Schools. 1901–03, Mechanical Engineering (Wallaroo Mines), Wallaroo & Moonta Min. & Smelt. Co.,

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Ltd. 1903–06, Mechanical and Electrical Engineering, Wallaroo & Moonta Min. & Smelt. Co., Ltd. (Wallaroo Smelt. Works). 1906–09, Engineering Draftsman & Surveyor (Wallaroo Smelt. Works), Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1909–12, Engineering Draftsman & Surveyor (Wallaroo Mines), Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1912–14, Engineering Draftsman and Surveyor (Wallaroo Smelt. Works), Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1907–16, Instructor Mechanical and Machine Drawing, Moonta School of Mines, evening classes. 1915 to date: Metallurgical Staff, Wallaroo Smelt. Works.

Present position: Asst. Met., Wallaroo Smelt. Works.

Harry Auld Bell Motherwell, Clifton, Ariz.

Proposed by W. F. Geiger, M. S. Mazany, C. E. Arnold.

Born 1872, Glasgow, Scotland. 1885, Glasgow High School. 1887–89, 1897–98, Glasgow and West of Scotland Technical College, 1890, Mill worker and Chemist, Glengowan Dyeing and Bleaching Works, Scotland. 1899, Chemist, Lake George Copper Mines, N. S. W., Aust. 1901, Chemist, Hercules Gold and Silver Mines, Tasmania. 1902, Chemist, Great Cobar Copper Mines, N. S. W., Aust. 1909, Chemist and Genl. Asst., Scottish-Cardigan Lead Mines, Wales. 1910, Genl. Asst. to Chief Engr., Angola Exploration Syndicate, Angola, S. W., Africa. (Messrs. Farrar Bros., London). 1911, Assayer and Cyanider, Taquah Gold Mine & Exploration Co., Gold Coast, W. Africa. 1912–13, Asst. Engr. and Chemist, Western Orufu Mines, Northern Nigeria, W. Africa. 1914–15, Met. clerk, and mill operator, Inspiration Cons. Copper Co., Ariz. 1915–16, Chemist, International Smelt. Co., Ariz.

Present position: Smelter Chemist, Arizona Copper Co.

Robert Aldridge Nowlin, Elkhorn, W. Va.

Proposed by John J. Lincoln, H. D. Smith, Howard N. Eavenson.

Born 1888, Lynchburg, Va. 1910, Virginia Mil. Inst., B. S. in Civ. Engrg. 1911–12, Student, Civ. Engrg. course, Mass. Inst. of Tech. Several summers prior to 1906, Rodman and Chainman, with Maurice Garland, Lynchburg, Va. 1910, surveying, mapping, building coal washery, reconstructing tipple, etc., responsible for work during absence of Chief Engr., Rodman and Transitman, Freeburn Colliery, Edgarton, W. Va. 1910–11, general mining construction, surveying, office work, plotting surveys, designing small trestles, etc., Transitman and Draftsman, Vest & Cooper, Welch, W. Va. 1911, Draftsman, Pocahontas (Va.) Cons. Collieries Co., principally mine work. 1912, summer, Transitman and Draftsman, C. L. DeMott, Lynchburg, Va. 1913–14, general mining engineering, Glen Alum Coal Co., Glenalum, W. Va.; Transitman for this company and Gilliam, Arlington Shawnee, and Black Wolf Coal & Coke Companies, next to Chief Engr. for these companies; construction work at Glen Alum Coal Co. included opening a new piece of territory, tramway, 830-ft. steel conveyor, steel tipple, railroad tracks, etc., at Gilliam Coal & Coke Co. a picking table was added to old tipple, tracks changed, and walls built.

Present position: 1914 to date; Min. Engr., Crozer-Land Assn.

Harry Fordham Noyes, Newcastle, N. S. W., Aust.

Proposed by H. D. Hibbard, G. D. Delprat, David Baker.

Born 1881, Newton. 1900, Newton High School. 1904, Mass. Inst. of Tech., Dept. of Min. & Met., S. B. 1904–06, Blast Furnace and Bessemer Dept., Maryland Steel Co. 1906–10, Night Supt., New Jersey Zinc Co., Palmerton, Pa. 1910–11, Supt. of Blast Furnace, Princess Furnace Co., Glen Wilton, Va. 1911–14, Genl. Supt., Dayton Coal and Iron Co., Dayton, Tenn.

Present position: Supt. of Blast Furnaces, Broken Hill Prop. Co., Ltd.

Walter S. Palmer, Reno, Nev.

Proposed by George J. Young, J. C. Jones, Bradley Stoughton.

Born 1884, Waterville, Me. 1905, Univ. of Nevada, B. S. in Min and. Met. 1907, Columbia School of Mines, E. M. 1905, Greene Cons. Copper Co., Cananea, Mex. 1906, Mining and Metallurgical practice in the field, summer months, Virginia City, Nev., and Wilkes-Barre, Pa. 1907–08, Mine Surveyor, U. S. Smelt., Ref. and Min. Co., Mammoth, Cal. 1908–09, Asst. Chemist, Balaklala Cons. Copper Co., Coram, Cal. 1909–10, Min. Engr., U. S. General Land Office. 1910–13, Instructor in Mining and Metallurgy, Univ. of Nev. 1910, Chemist and Assayer, Nevada State Mining Laboratory. 1913, Asst. Prof. of Mining and Metallurgy, Univ. of Nev.

Present position: Chemist and Assayer, Nevada State Mining Laboratory and Asst. Prof., Mining and Metallurgy, Mackay School of Mines.

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George S. Patterson, Edwards, N. Y.
Proposed by Homer L. Carr, William Campbell, E. J. Hall.
Born 1892, Denver, Colo. 1914, Columbia School of Mines, E. M. 1914–16, underground mining operations and surveying, New Jersey Zinc Co.
Present position: Northern Ore Co.

John Lathern Pearson, Wallaroo Mines, So. Aust.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1871, Alston, Cumberland, England. 1877–84, Attended Church of England Day School and Public Schools, England. 1885–87, Studied Mathematics at a Night School, Moonta Mines, S. Amer. 1891–94, 1898–1901, Attended classes at Moonta School of Mines in Mine Surveying, Mineralogy, Metallurgy, Assaying, Mechanical Drawing, Steam and Practical Chemistry. 1885–89, Employed in Moonta Mining Company's Workshops at Moonta Mines, S. A. 1889–93, underground work at Moonta Mines. 1893–95, Asst. in Survey Dept., Wallaroo & Moonta Min. & Smelt. Co., Wallaroo and Moonta Mines. 1895–98, In charge of prospecting operations at Kalgoorlie, Kanowna and Menzies, Western Aust. 1898–1902, Underground survey work and sectional supervision of general mining work for Wallaroo & Moonta Min. & Smelt. Co. 1902–03, Asst. Mine Mgr., Nth. Lyell Mine, Tasmania. 1903–09, Mine Mgr., Queensland Copper Co., Mt. Perry, Queensland. 1910, Six months as Mine Mgr. at Mt. Jasper Mine, Heazlewood, Tasmania. 1910–15, Asst. underground officer at Wallaroo Mines.
Present position: 1915 to date; Chief underground officer, Wallaroo Mines.

Fred Hough Perkins, Glendale, Ariz.
Proposed by Thomas Armstrong, W. E. Defty, F. W. Libbey.
Born 1876, Kansas City, Mo. 1886–92, Ward School, Kansas City, Mo. 1893–96, High School, Kansas City, Mo. 1897–99, Missouri School of Mines, B. S. in General Science. 1900, Missouri School of Mines, B. S. in Min. Engrg. 1898, summer, L. X. Smith Assay Office, Cripple Creek, Colo., also Rio Grande Sampler of that place. 1899, summer, Perkins & Perkins, Assayers, also Supt., Kopy Kat Lead & Zinc Min. Co., Joplin, Mo. 1900, Asst. Min. Engr., Diamondville Coal and Coke Co., Diamondville, Wyo. 1901, Asst. Engr. and Assayer, Utah Cons., Bingham Canyon, Utah. 1902–03, Cons. Engr., Red Wing and Utah Apex Min. Cos., Bingham Canyon, Utah. 1904, General Min. Engrg. specializing in mine litigation. 1905, Min. Engr., Blue Ledge Min. Co., Jacksonville, Ore. 1907–08, County Engr., Jackson Co., Ore. 1909–15, General practice of Mine Engrg. in Arizona.
Present position: 1915 to date; Genl. Mgr. Shamrock Cons. Min. Co., Castle Hot Springs, Ariz.

Ocha Potter, Houghton, Mich.
Proposed by A. H. Wohlrab, F. W. Paine, Reginald E. Hore.
Born 1878, Sherwood, Wis. 1911, Michigan College of Mines, B. S. 1905–09, Mgr., Houghton-Alaska Exploration Co.
Present position: 1909 to date; Supt., Superior Copper Co. and Chief Efficiency Engr., Calumet & Hecla and associated mines.

Walter C. Rea, Ruth, Nev.
Proposed by H. L. Handley, George R. Hilby, W. S. Larsh.
Born 1893, Gallatin, Mo. 1913, Preparatory work, Wentworth. 1914, Preparatory work, Military Academy, Lexington, Mo. 1914, Mackay School of Mines, Univ. of Nevada, Reno, Nev. 1914–15, Surveyor, Buckhorn Mines Co., Buckhorn, Nev. 1915, Met. Experiments, Golden Cycle Mill, Colorado Springs, Colo. 1915 to present, General Engineering and Chemist, Nevada Cons. Copper Co., Ruth, Nev.
Present position: Engr., Ruth Mine.

Donald Duryea Reed, Kadina, South Australia.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1890, Wallaroo Mines, South Australia. 1895–1904, Public School, Wallaroo Mines. 1904–06, District High School, Moonta. 1906–10, Technical—School of Mines, Moonta, Depts. of Drawing, Chemistry, Assaying & Mineralogy. 1906–11, Met. Assay Dept. (Wallaroo Smelt. Works) & Chemical Laboratory (Wallaroo Mines), Wallaroo & Moonta Min. & Smelt Co., Ltd. 1911–16, Assisting in Supervising at Converter & Electrolytic Departments. 1915 to date: Mathematical Instructor, Moonta School of Mines, evening classes.
Present position: Asst. Met., Wallaroo Smelt. Works.

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Charles Ramsay Rinehart, Allentown, Pa.
Proposed by F. L. Antisell, S. Skowronski, Edward Heller.
Born 1875, Uniontown, N. J. 1899, Grad., Lafayette College, Civ. Engr. Tippett & Wood, Phillipsburg, N. J. Great Northern Portland Cement Co., Marlboro, Mich. Hudson Portland Cement Co., Hudson, N. Y. Seaboard Portland Cement Co., Alsen, N. Y. Lehigh Car, Wheel and Axle Works, Catasauqua, Pa. Fuller Engineering Co., Allentown, Pa. Lehigh Foundry and Lehigh Stoke Co., Catasauqua, Pa.
Present position: New York Representative, Lehigh Car, Wheel and Axle Works, Catasauqua, Pa.; Fuller Engineering Co., Allentown, Pa.; Lehigh Foundry and Lehigh Stoke Co., Catasauqua, Pa.

Bernard T. Rocca, Mayer, Ariz.
Proposed by J. L. White, J. N. D. Gray, A. F. Bassett.
Born 1892, Great Western Q. S. Mine, Lake Co., Cal. 1915, Univ. of Cal., B. S., College of Mining. 1914, General work, surface and underground, Helen Quicksilver Mine, Lake Co., Cal. 1915 to date, surveying, assaying, draughting, Cons. Arizona Smelt. Co., Humboldt, Ariz.
Present position: Engr., Bluebell Mine, Cons. Ariz. Smelt. Co.

C. Floyd Sapper, Herculaneum, Mo.
Proposed by William Allen Smith, Franz Cazin, S. Paul Lindau.
Born 1886, St. Louis, Mo. Grammar School, St. Louis. 1908–10, Univ. of Missouri, Mech. Engrg. 1910–15, Univ. of Missouri, specialized in chemistry and allied subjects. Substituted credit from Univ. of Missouri for high school credit. 1904–06, Mill Machine and Tool Co., St. Louis. 1906–08, Engrg. Dept., Century Electric Co., St. Louis. 1911, summer, Mech. Dept., Curtis & Co. Mfg. Co., St. Louis. 1912–13, summers, Drafting, Herculaneum plant, Engrg. Dept., St. Joseph Lead Co.
Present position: 1914 to date, Engrg. Staff, St. Joseph Lead Co.

Matthew Johnston Scammell, Sparrows Point, Md.
Proposed by Quincy Bent, Charles F. Rand, Edgar C. Felton.
Born 1883, Trenton, N. J. 1900, High School, Trenton, N. J. 1905, Lafayette College, Easton, Pa., B. Sc. Roebling Sons' Co., Trenton, N. J. Republic Iron & Steel Co., Youngstown, O. Carnegie Steel Co., Youngstown, O. Maryland Steel Co., Sparrows Point, Md.
Present position: Genl. Supt., Maryland Steel Co.

Anton Schneider, Brewster, Polk Co., Fla.
Proposed by David Taylor, Benjamin L. Miller, George W. Engel.
Born 1871, Summit Hill, Pa. 1892, Lehigh Univ., C. E. 1892–98, Union Pacific R. R. 1898–99, U. S. Engineers at Honolulu. 1899–1902, Manhattan Ry. Co., New York. 1902–04, Cerro de Pasco Co., Peru, So. Amer. 1904–05, Alphons Curtis Chimney Const. Co. 1905–08, Div. Engr., East River Tunnels, Rapid Transit Subway Const. Co., New York. 1908–11, Supt., Pierce Phosphate Co., Fla.
Present position: 1911 to date; Supt., Amalgamated Phosphate Co.

Arthur S. Shoffstall, New Brighton, Staten Island, N. Y.
Proposed by Robert S. Stanley, Walter O. Snelling, E. T. McCleary.
Born 1876, Jefferson Co., Pa. 1895, Public Schools, Brookville, Pa. 1895–96, Preparatory Dept., The Pennsylvania State College. 1896–1900, The Pennsylvania State College, School of Natural Science, B. S. 1905, M. S. 1900–01, Asst. in the Chemical Laboratories, The Pennsylvania State College. 1903–05, Instructor, Dept. of Chemistry, The Pennsylvania State College. 1905–07, Supt., Sulphuric Acid Plants, E. I. duPont Powder Co., Gibbstown, N. J. 1907–08, Supt., Acid Plants, Naval Proving Ground, Indian Head, Md. 1908–10, Technical Staff, Orford Copper Co., Bayonne, N. J. 1910–13, Supt., Sheet Mill Dept., The International Nickel Co., Bayonne, N. J.
Present position: Research Engr., The International Nickel Co., Bayonne, N. J.

Ellsworth H. Shriver, Gillham, Sevier Co., Ark.
Proposed by George A. Morrison, C. W. Whitley, J. H. Frantz.
Born 1890, Washington, Pa. 1909, Martins Ferry High School. 1914, Engr. of

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Mines, The Ohio State Univ. 1911, Genl. Engrg. with W. C. Fawcett, C. E., Martins Ferry, O. 1911–12, Asst. County Engr., Belmont Co., O. 1913, Genl. Min. Engrg., Bellaire, O. 1914, Investigation Coal Lands, East Ky., for Major J. A. Kirkpatrick, Springfield, O. 1914–15, Asst. Dept. Mine Eng., The Ohio State Univ. 1915–16, Investigation Mineral Lands and Water Power Sites, The Ohio State Univ.

Present position: 1916 to date; Met., The American Star Antimony Co.

Daniel Sillars, Yorkshire, England.

Proposed by W. R. Walker, J. E. Stead, William Thomlinson.

Born 1879, Glasgow, Scotland. Three years apprenticeship with Messrs. R. R. Gatlock and Thomson, City Analysts of Glasgow. Two years Metallurgy, Glasgow & West of Scotland Technical College, Senior Prizeman. Eight years Metallurgical Chemist, William Beardmore & Co., Ltd., Parkhead Forge, Glasgow. Five years, Chief Chemist, Messrs. Seaton Carew Iron Co., Ltd., West Hartlepool.

Present position: Chief Chemist, Messrs. Bolckow Vaughan & Co., Ltd., Cleveland Iron & Steel Works, South Bank S. O., Yorkshire.

Laurence Heck Stone, Easton, Pa.

Proposed by Ralph H. Sweetser, Harry P. Sweeny, W. Spencer Hutchinson.

Born 1889, Easton, Pa. 1909, Grad., Easton High School. 1913, Grad., Lafayette College, E. M. 1911, summer, Machinist Helper, Erection shop, Ingersoll-Rand Co., Philipsburg, N. J. 1912, summer, Rod and Chain, Genl. Mgrs. Corps, Lehigh Valley R. R. 1913, Transitman and Surveyor, Wilkes-Barre Anthracite Coal Co., Wilkes-Barre, Pa. 1913–14, Asst. Mine Surveyor, Thomas Iron Co., Richard Mine, Wharton, N. J. 1914–15, Mine Surveyor and Engineer, Thomas Iron Co., Richard Mine, Wharton, N. J.

Present position: 1915 to date; Asst. Supt., and Engr., Thomas Iron Co., Richard Mine.

Charles H. Thole, Rye Valley, Ore.

Proposed by L. A. Walker, Frank W. Parker, A. W. MacNichol.

Born 1878, Dwight, Ill. 1896, High School. 1896–98, So. Dakota School of Mines. 1898–1902, underground, Homestake Min. Co., Lead, S. D. 1905–06, Mine Foreman, Desert King Min. Co., Ballarat, Cal. 1907, underground, City of Los Angeles, Engrg. Dept. 1908, Leasing, Aurora, Nev. 1909–10, Sampler, Shift boss, Assayer, Ruby Gulch Min. Co., Zortman, Mont.

Present position: 1912 to date; Mill Foreman, Rainbow Mine.

James Ezra Thoms, Depue, Ill.

Proposed by Robert H. Richards, W. M. Kelsey, D. C. Wray.

Born 1883, Three Rivers, Mich. Public Schools. 1905–06, Michigan College of Mines. 1908–10, Engr., Live Oak Dev. Co., Globe, Ariz. 1910–13, Asst. Supt., U. S. Gypsum Co., Gypsum, O.

Present position: 1913 to date; Asst. Chief, Milling and Roasting Dept., Mineral Point Zinc Co.

Adolph F. Zang, Denver, Colo.

Proposed by Irving T. Snyder, D. W. Brunton, A. E. Carlton.

Born 1890, Denver, Colo. 1913, Cornell Univ., A. B.

Present position: Director and Asst. Treas., The Vindicator Cons. Gold Mining Co.; Secy., The Cresson Cons. Gold Min. & Mill. Co., both of Cripple Creek, Colo. In Bond Dept., The German American Trust Co., Denver.

Associate Members

Walter S. McKee, Chicago, Ill.

Proposed by A. C. Ludlum, Newton Cleaveland, Charles Janin.

Born 1876, Watervleit, Mich. 1883–87, Grammar School, St. Charles, Ill. 1887–96, High School, Browning, Mo. 1900, International Correspondence School, Course in drafting. 1897–99, Asst. Cashier, Citizens Bank, Aledo, Ill. 1899–1900, Asst. credit man, A. G. Spalding & Co., Chicago. 1900–01, In the accounting department, The Sargent Co., Chicago. 1901–02, Salesman, The Sargent Co., Chicago. 1902–10, Sales Mgr., Steel Foundry Dept., American Brake Shoe and Foundry Co., Chicago. 1910–11, Genl. Sales Mgr., American Manganese Steel Co., Chicago.

Present position: 1911 to date; Vice-Pres., American Manganese Steel Co.

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Carlos Enrique Michels, Huanuni, Bolivia, So. Amer.
Proposed by William Braden, Henry G. Ferguson, Ira B. Joralemon.
Born 1889, Lima, Peru. 1904, Institute de Humanidades, Santiago, Chile. 1906–07, Draughtsman, Braden Copper Co., Chile. 1907–09, Asst. to Chief Engr., Braden Copper Co., Chile. 1910–12, Met. Dept., Braden Copper Co., Chile. 1913–14, Asst. Supt., Llallagua Tin Mines, Bolivia.
Present position: 1915 to date; Supt., Met. Establishment and Mill, Simon Patino Tin Mines.

T. E. Snyder, Hazelton, Pa.
Proposed by A. B. Jessup, Charles Enzian, J. M. Humphrey.
Born 1857, St. Johns, Pa. Midvalley Coal Co., Wilburton, Pa. Maryd Coal Co., Maryd, Pa. J. S. Wentz & Co., Hazle Brook, Pa. Upper Lehigh Coal Co., Upper Lehigh, Pa.
Present position: General Manager.

Roger W. Straus, New York, N. Y.
Proposed by Judd Stewart, P. A. Mosman, Willard S. Morse.
Born 1891, New York. 1913, Princeton, Litt.
Present position: Asst. to Chairman, Board of Directors, American Smelt. & Ref. Co.

Gustave C. Taussig, St. Louis, Mo.
Proposed by George O. Carpenter, Herman Garlichs, Arthur P. Watt.
Born 1884, St. Louis, Mo. 1902, Grad., St. Louis High School.
Present position: Asst. Secy., St. Louis Smelt. & Ref. Co.

Frank J. Gustin, Salt Lake City, Utah.
Proposed by Ernest Gayford, C. W. Whitley, C. W. Stimpson.
Born 1877, Princeton, Ill. 1897, L. L. B., University of Nebraska. Member of American Bar Association; Utah State Bar Association; American Institute of Criminal Law. Interested directly and indirectly with a number of mining companies. Member of the Bar of the States of Nebraska, Utah, Nevada, Idaho and United States Supreme Court.
Present position: Attorney at Law. Member of the firm of Gustin, Gillette & Broyton.

Junior Members

Harlowe Hardinge, Ithaca, N. Y.
Proposed by H. W. Hardinge, R. B. T. Kiliani, Ronald Clark.
Born 1894, Denver, Colo. 1910–12, Attended Tome School, Port Deposit, Md. Have been on a considerable number of extensive trips and have assisted in the operation of Hardinge conical mills at a number of plants.
Present position: 1912 to date; Sibley College, Cornell Univ.

Richard Paul Ernst Hermsdorf, New York, N. Y.
Proposed by William Campbell, E. J. Hall, Charles P. Berkey.
Born 1895, New York, N. Y.
Present position: Student, Met. Engrg., Columbia Univ.

Charles Francis Williams, Chivatera, Son., Mex.
Proposed by John A. McDonald, A. S. Konselman, W. A. Richelsen.
Born 1894, Mansfield, O. 1914, Grad., St. John's Military Academy, Delafield, Wis. 1914 and 1915, New Mexico State School of Mines. 1915, Colorado School of Mines. 1916, Asst. to Met. Engr., Chino Copper Co., Hurley, New Mexico.
Present position: Mine Engr., Cananea Cons. Copper Co.

Change of Status—From Associate Member to Member

Edmund Mozier King, Leura, Launceston, Tasmania.
Born 1848, Launceston, Tasmania. 1889–1902, College Saint Croix, Aux. terms, Paris. 1902–06, Church of England Grammar School, Launceston, Tasmania.

Studied Chemistry under R. N. Hobart. 1887 to date: Director, Mt. Bischoff Tin Min. Co. 1886–1915, Director, Tasmania Gold Min. and its reconstruction until liquidated. 1897 to date: Director, Tasmania Copper Co. (at present Chairman) situated at Rosebery, West Coast of Tasmania. Foundation member of the Australian Institute of Min. Engrs.

Present position: Director of various mining companies.

From Junior Member to Member

William Cristy Pryer, Anaconda, Mont.

Proposed by C. R. Wraith, Enoch A. Barnard, Albert E. Wiggin.

Born 1890, Falmouth, Mass. 1914, So. Dak. School of Mines, B. S. 1914, Assayer, Titanic Min. Co., Deadwood, S. Dak. 1915, Prospect examination with S. R. Thompson, Deadwood, S. Dak. 1916, Testing Dept. and Chemist, Anaconda Copper Min. Co.

Present position: Chemist, Anaconda Copper Min. Co.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Aug. 1, 1916 to Sept. 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 116, February to September, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of Sept. 10, 1916.

AHLERS, R. O., 1–4 Norfolk House, Laurence Pountney Hill, London, E. C., England.
ALLEN, ARTHUR WATTS.....Instructed to hold everything.
AMBLER, HARRY A.....Box 540, Rolla, Mo.
ANDERSON ROBERT JOHN, Asst. to Prof. Henry M. Howe, Broad Brook Road, Bedford Hills, N. Y.
APPLEGATE, J. S.....Material Engr., Wagner Electric Mfg. Co., St. Louis, Mo.
ATHERTON, T. W. T.....Kagesman, Kars., Obi, Russia.
BASSETT, THOMAS E.....236 Park St., Pontiac, Mich.
BELDEN, S. B., Vice-President, The Jeffrey Manufacturing Co., Columbus Ohio.
BENHAM, W. M.....Co. H, 1st Ariz. Infantry, Fort Huachuca, Ariz.
BICKNELL, HAROLD LEWIS.....Hennessy Annex, Butte, Mont.
BILLICK, DON C.....Box 99, Alturas, Cal.
BONSIB, R. S.....U. S. Metals Refining Co., East Chicago, Ind.
BOYDELL, H. C., Prestea Block A., via Seccondee, Gold Coast Colony, W. Africa.
BRADLEY, HENRY F.....11 Bransford Apartments, Salt Lake City, Utah
BRIDGE, SAMUEL D., Care Patricio Milmo e Hijos, Sucs., Banqueros, Monterrey, Mex.
BUELL, L. T.....Ore Trading Co., Ltd., Casilla 851 Antofogasta, Chile.
CALLEN, A. S.....Hold publications
CAREAGA, C. R.....Calle de la Florida 5, Seville, Spain.
CARPENTER, EDWIN LEON.....Box 397, Salisbury, Md.
CARSTENS, CARL EBERHARD.....400 W. 4th St., Anaconda, Mont.
CHATIN, AUGUST. H.....Box 424, Walsenburg, Colo.
CLAGHORN, JAMES L.....Forks of Salmon, Siskiyou Co., Cal.
CLAPP, C. H., Prof. of Geol.....Montana State School of Mines, Butte, Mont.
CLARKE, EDGAR W.....622 Second Ave., Johnstown, Pa.
COE, IRA J.....362 Euclid Ave., Oakland, Cal.
Cole, A. Newton.....H. Koppers Co., Care United Furnace Co., Canton, O.
CUNNINGHAM, NOEL.....200 5th Ave., New York, N. Y.
CUSHING DANIEL.....17 Fairview St., Waterbury, Conn.
DAMMANN, A. C.....5410 Wayne Ave., Chicago, Ill.
DAVIS, JOHN A.....U. S. Bureau of Mines, Washington, D. C.
DICKMAN, ROBERT N.....170 Bay St., St. Augustine, Fla.
DOYLE, JOHN J.....Cosden Oil & Gas Co., Drawer P, Tulsa, Okla.
DU MOULIN, WALTER LOUIS, Mech. Engr., Andes Copper Mining Co., Chanaral, Chile, South America.
DWYER, C. EUSTACE.....Federal Min. & Smelt. Co., Wallace, Idaho.

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DYE, A. V., Asst. Genl. Mgr.....Phelps, Dodge & Co., Douglas, Ariz.
EAMES, LUTHER B.....Hollinger Gold Mines, Ltd., Timmins, Ont., Canada.
ELDREDGE, ROBERT B.....Ray Cons. Copper Co., Ray, Ariz.
EMERY, WILLIAM, JR.....Natalie, Pa.
EMMONS, N. H., Mgr.....Tennessee Copper Co., Copperhill, Tenn.
FILTEAU, CHARLES A.....St. Lawrence Talc Co., Natural Bridge, N. Y.
FISHER, HOWELL T.....6 Spring Lane, Englewood, N. J.
FOERSTER, HALLARD W., Care Tigre Mining Co., Esqueda, Son.,
Mexico, via Douglas, Ariz.
FOX, WALTER V.....Coralbut Mining Co., Joplin, Mo.
FUKETA, F.,.....Mitsubishi Kaisha, Mining Dept., Marunouchi, Tokyo, Japan.
GANTHIER, C. B.....P. O. Box 252, Tucson, Ariz.
GEE, J. EMERSON.....304 H. W. Hellman Bldg., Los Angeles, Cal.
GRANT, BERTRAM.....615 Market St., Wilmington, Del.
GREIT, ADOLPH JULIUS.....Calle 13 Y 6, Vedado, Habana, Cuba.
GROSS, JOHN, Mining and Metallurgical Engr., 423 McPhee Bldg., Denver, Colo.
GRUBNAU, V. CARL.....114 Arch St., Philadelphia, Pa.
GUERNSEY, JOHN B.....132 W. 4th St., New York, N. Y.
GUITERMAN, KENNETH S., Selby Smelting & Lead Co.,
804 Merchants Exchange Bldg., San Francisco, Cal.
HALDEMAN, G. T.....Hold publications.
HAMILTON, C. W., Mexican Gulf Oil Co., Apartado 106, Tampico, Tamps., Mex.
HAMILTON, W. R., Petroleum & Min. Engr., Hobart Bldg., San Francisco, Cal.
HANCE, JAMES H.....815 North Johnson St., Iowa City, Iowa.
HARRINGTON, DANIEL, U. S. Bureau of Mines, 334 Post Office Bldg., Butte, Mont.
HASTINGS, JOHN B.....P. O. Box 954, McGill, Nev.
HEAD, JAMES L.....Care Calumet & Arizona Min. Co., Warren, Ariz.
HILL, WALTER HOVEY.....917 Franklin St., Boise, Idaho.
HODGSON, JOSEPH P., Cons. Engr., Mining Dept., Phelps, Dodge & Co., Bisbee, Ariz.
HOFFMAN, LLOYD.....Pottersville, N. J.
HOLDERER, GEORGE B.....General Chemical Co., 25 Broad St., New York, N. Y.
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Date of Election.	Name.	Date of Decease.
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1891	*Towne, Robert S	Aug. 3, 1916.

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EMMONS' ORE-DEPOSITS

A continuation of the "Posepny" Volume

Comprising Papers Descriptive of Ore-Deposits and Discussions of their Origin, Edited, with an Introduction, by Dr. S. F. Emmons.

Contents.

Genesis of Certain Ore-Deposits. By S. F. Emmons.
Structural Relations of Ore-Deposits. By S. F. Emmons.
Geological Distribution of the Useful Metals in the United States. By S. F. Emmons. Discussion by John A. Church, Arthur Winslow, S. F. Emmons, and William Hamilton Merritt.
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Manganese Ores of the Lafayette District, Minas Geraes, Brazil

BY JOSEPH T. SINGEWALD, JR.,* PH. D., BALTIMORE, MD., AND BENJAMIN LEROY MILLER,† PH. D., SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1917)

Introduction

For a number of years Russia, India and Brazil have outranked all other countries as producers of manganese ores. During the 5 years immediately preceding the European war, the average annual production of Russia was 844,000, of India 694,000 and of Brazil 200,000 long tons. Since the outbreak of the war there has been a considerable falling off in the Russian and Indian production, particularly in the former, whereas the production of Brazilian ores has been greatly increased, amounting in 1914 to over 250,000 tons and in 1915 to nearly 350,000, with conditions favorable for a still larger production in 1916. Manganese is one of the few industrially important metals that are not produced in the United States in quantity commensurate with our needs, so that we have been compelled to import annually about 300,000 tons, having a value of over $2,000,000, and these ores have been obtained from the three countries mentioned above. Their rank as contributors to our imports of manganese ores was India, Russia, Brazil; and in 1913, Brazil contributed only one-fifth. During the past 2 years Brazil has furnished rapidly increasing quantities to this country, and, with the falling off of imports from India and Russia, has become our principal foreign source. In 1914, Brazil furnished two-fifths of the imports and more than India or Russia, and in 1915 over nine-tenths. The manganese ores of Brazil are consequently of more than usual interest to us at this time.

The Manganese-Ore Districts of Brazil

The manganese mining industry of Brazil dates from the year 1894, and since that year the total production has been over 3,000,000 tons. With the exception of a small quantity produced in the State of Bahia from deposits west of the City of Bahia, this output has come from the

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State of Minas Geraes and has been exported through the port of Rio de Janeiro. In Minas Geraes there are two distinct, though not widely separated, manganese districts, known generally as the Miguel-Burnier and the Queluz or Lafayette districts. The Miguel-Burnier district was the first to be developed and for a number of years was the more important producer; but, since the phenomenal development of the Morro da Mina mine, the Lafayette district has assumed far greater importance.

In order to contrast the modes of occurrence of the deposits in the two districts, a brief description of the Miguel-Burnier district will be given, in the nature of a summary of a previously published fuller account.[1] The deposits of this district extend as a narrow belt about 10 miles long, parallel to the Ouro Preto branch of the Estrada da Ferra Central do Bresil, and lie at the southern edge of the great iron-ore region of central Minas Geraes. The orebodies occur as lenses or beds intercalated in a series of sedimentary strata showing a rapid succession of itabirite, schist, calcareous schist, and limestone. Stratigraphically they are found in the upper part of the Itabira iron-formation and in the lower part of the overlying Piraçicaba schist,[2] formations of probable Algonkian age. The ores are very high-grade manganese oxides, chiefly a mixture of psilomelane and pyrolusite, averaging 50 per cent. manganese, 1 per cent. silica, and 0.03 to 0.05 per cent. phosphorus. The great drawback of the district is the fact that the beds are steeply dipping and narrow, rarely over 6 ft. in width, so that expensive underground mining must be resorted to, and the individual deposits are relatively small. Two views have been advanced to explain the origin of the ores. H. K. Scott, who has written the most complete account of them, says:[3] "Whatever may have been the original state of the manganese ore bed, there can be no doubt that in its present condition, and down to the level to which it has been worked, it is a residual deposit from which the other elements have been leached out." O. A. Derby, in a discussion of Scott's paper, endorsed this view and referred to the original state of the manganese ore beds as limestone with varying proportions of metallic carbonates and siliceous impurities.[4] On the other hand, Harder and Chamberlin say:[5] "From their occurrence it must be assumed that they are similar in origin to the associated rocks, that is, that they are original sedimentary

[1] Joseph T. Singewald, Jr., and Benjamin LeRoy Miller: High-Grade Manganese Ores of Brazil. *The Iron Age*, vol. 97, pp. 417–420 (1916).

[2] E. C. Harder and R. T. Chamberlin: Geology of Central Minas Geraes, Brazil. *Journal of Geology*, vol. 23, pp. 358–363 (1915).

E. C. Harder: Manganese Ores of Russia, India, Brazil and Chile. *Bulletin* No. 113, p. 788 (May, 1916).

[3] H. K. Scott: Manganese Ores of Brazil. *Journal of the Iron and Steel Institute*, vol. 57, pp. 188–189 (1900).

[4] *Idem*, p. 212.

[5] *Op. cit.*, p. 406.

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deposits of manganese oxide." The same statement is made by Harder alone.[6] Our own feeling is that while one cannot positively state that the ores were not laid down in the form of manganese oxides as integral parts of a sedimentary series, their relations to the associated limestones are such as to make their interpretation as residual products of decomposition and replacement of manganiferous limestone the more probable explanation of their origin.

The Lafayette District

Manganese deposits were discovered in the Lafayette district immediately after the inauguration of mining operations in the Miguel-Burnier district stimulated a search for manganese ores in Minas Geraes, but it was not until the year 1900 that the district became a regular producer, an output of 31,000 tons coming from the Piquery and São Gonçalo Mines in that year. This was increased to nearly 75,000 tons the following year, a production considerably in excess of that of the entire Miguel-Burnier district. In 1902, the Morro da Mina came in as a producer and firmly established the district as the chief manganese-producing district of Brazil.

Lafayette is a station, located at the edge of the town of Queluz, on the Estrada da Ferra Central do Bresil, about 32 km. south of Burnier and 462 km. from Rio de Janeiro. From the fact that the most important producing mines have been located in its vicinity the district has been generally referred to as the Lafayette district. The most important mine at present is the Morro da Mina, owned and operated by a Brazilian company known as the Companhia Morro da Mina, which has increased its production to 700 tons per day. It is located 7 km. northeast of Lafayette. On the same hill, a German company under the name Mineração de Agua Preta has been working the rubble ores to the east of the Morro da Mina ground. This company is producing at the rate of 2,000 tons per month and in the 6 or 7 years it has been operating has produced a total of 200,000 tons. On a small hill to the southeast of the Morro da Mina, the extension of that zone is being developed by a company known as the Companhia Queluz da Mina. The only other producing mine is the Cocuruto which lies about 40 km. southwest of Lafayette and is connected with the railroad at Christiano, a station 23 km. further south, by a 60-cm. gage line 40 km. in length. This mine has been operated about 7 years and is producing 3,000 tons monthly. It is owned by a Belgian company, the Société Anonyme de Manganese de Ouro Preto, which formerly worked the São Gonçalo Mine. After each had produced about 250,000 tons of ore, the São Gonçalo and Piquery mines were abandoned 10 years ago as worked out. They

[6] *Op. cit.*, p. 791.

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were located about 15 km. northwest of Lafayette. The Piquery Mine is of particular interest in that it is the only one of which there is is a geological description and it was there that Derby obtained the first evidence of the original character of the manganese rock from which the ores were derived.

In addition to those mentioned, a number of other deposits have been discovered and prospected to some extent, but apparently the results were not favorable enough to warrant further development as none of them became important producers. It is certain, however, that the district has not been thoroughly prospected and there is every reason to expect that systematic exploration would discover deposits equal to those that have been found.

Geology of the Lafayette District

The geology of the Lafayette district differs markedly from that of the Miguel-Burnier. It lies to the south of the area underlain by the

FIG. 1.—SHARP CONTACT OF MANGANESE ORE AND COUNTRY ROCK IN MORRO DA MINA MINE.

great iron-bearing series, and its ores are found in the basement complex of supposed Archean age which underlies a large part of the state of Minas Geraes. The rocks making up this basement complex are chiefly granite and gneiss with which are associated amphibolite, and micaceous and quartzose schists. There are also small intrusions of diorite and gabbro, mostly in the form of dikes. The granite seems to be intrusive into the gneiss and schist, but the relations between the schist and gneiss are not clear.

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The manganese deposits occur as elongated masses of more or less lenticular shape within the rocks of the basement complex. As will be explained below, they represent residual products of decomposition of an original manganiferous rock made up of manganese carbonate and silicates. The immediate wall-rock of the deposits has likewise undergone decomposition, in many instances being nothing more than a clay in which the original rock texture is poorly preserved, so that it is usually difficult to determine its original character. In most cases, however, it seems to have been either gneiss or schist. The contact of wall-rock and ore generally appears quite sharp, as is shown in Fig. 1, but closer examination often reveals small nests and stringers of manganese oxide in the decomposed rock. There are also horses of equally decomposed rock and of the same character within the orebodies themselves.

The Piquery Mine

The geological relations of the Piquery orebody have been described by O. A. Derby in two papers published in 1901 and 1908,[7] and the following account is abstracted from them.

"The Piquery orebody presents the appearance of a mass of secondary material, or gossan, resulting from the alteration of a vertical dike or vein, some 10 or 12 m. wide. . . . The ore is a hard spongy black oxide, apparently consisting for the most part of psilomelane but with an admixture of other oxides that frequently occur in beautiful crystallizations in the spongy cavities. . . . In the midst of the merchantable ore occur inconstant bands and patches of hard siliceous material with the appearance of a quartzite, but which on examination proves to be composed almost exclusively of a finely granular mass of ashy white manganese garnet. A complete series of alteration phases between perfectly typical garnet rock and merchantable ore can be readily selected, and there can be no doubt that the latter results from the decay and leaching of the former."

Derby describes three phases of the garnet rock that he observed in 1901:

"1. A very fine-grained, compact and finely jointed rock of bluish-gray color with partings lined with asbestus. Under the microscope the rock is seen to be composed almost exclusively of closely appressed idiomorphic grains of white garnet showing a clear border but with the center highly charged with a fine black opaque powder that appears to be graphite. . . .

"2. A dark brown rock heavily charged with manganese oxide and too friable to permit the preparation of microscopic sections is evidently of the same type but more completely decomposed. . . .

[7] O. A. Derby: On the Manganese-Ore Deposits of the Queluz (Lafayette) District Minas Geraes, Brazil. *American Journal of Science*, Ser. 4, vol. 12, pp. 18–32 (July, 1901).

O. A. Derby: On the Original Type of the Manganese-Ore Deposits of the Queluz District, Minas Geraes, Brazil. *American Journal of Science*, Ser. 4, vol. 25, pp. 213–216 (March, 1908).

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"3. A milky white rock which under the microscope is seen to be composed of about equal parts of garnet and quartz. . . . The quartz in a fine mosaic about the garnet grains and in minute refilled joints is almost certainly secondary."

Residues of manganese garnet were likewise found by Derby at the São Gonçalo, Morro da Mina, Agua Limpa and Barroso Mines. The last two mines were located about 10 km. southeast and 9 km. south of Lafayette respectively. As the result of his observations at these mines, Derby drew the following conclusions concerning the origin of the deposits:

"The orebodies of the Queluz district are residual deposits derived through decomposition and leaching from an original type or types of rock in which manganese garnet was the most constant and characteristic silicate element. . . . This type, which may appropriately be denominated *queluzite*, is more or less intimately associated at São Gonçalo, Morro da Mina and Barroso with decomposed schistose rocks that evidently contained an original manganese-bearing silicate and which from the absence of recognizable clastic elements and from other characteristics, so far as they can be made out, is presumed to have been an amphibolic schist representing a sheared basic eruptive. . . . In the Agua Limpa schist moreover, the manganese-bearing element is spessartine, as in the orebodies, thus giving greater plausibility to the hypothesis that the relation between these last and the above-mentioned rocks may be a genetic one. If thus related, the orebodies present strong analogies with those of magnetic, titaniferous and chromic iron ores that are now generally considered as magmatic segregations in various types of eruptives, and, all things considered, this hypothesis seems the most plausible one for the manganese ores here discussed."

After the Piquery orebody was worked out and the original manganiferous rock was exposed in the bottom, Derby made a further study of it, the results of which are given in the 1908 paper. He found that the rock consists mainly of "a black, fine-grained, highly jointed and somewhat flaggy rock with the aspect of a limestone, with broad bands and patches of a more massive, yellowish-gray rock with the aspect of a quartzite." The latter is the garnet rock described in the previous paper, but which now turns out to be of secondary importance. The limestone-like rock on treatment with cold weak acid effervesces freely with an abundant separation of gelatinous silica and an insoluble residue containing spessartite and graphite. In places there is also a considerable admixture of rose-colored rhodonite in streaks and patches as the predominant component. Microscopic examination by Dr. Hussak showed the rocks to consist of manganese carbonate, tephroite, and spessartite with a small amount of rhodonite. The paper gives three analyses of these rocks, of which No. I was selected with reference to a supposed high carbonate content, No. II as having a lower carbonate content, and No. III was taken from the earlier paper and represents the garnet rock first found.

For the purpose of comparing the rocks represented by these analyses with the Morro da Mina rocks described below, their mineralogic composition has been calculated approximately on the basis of their described

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Analyses of Original Manganese Rock at Piquery

	I	II	III
CO_2	22.62	4.59	
SiO_2	11.80	27.69	38.47
MnO	47.52	57.48	27.90
Al_2O_3	7.50	1.41	21.07
Fe_2O_3	.....	2.48	7.38
CaO	3.76	1.82	4.70
MgO	6.27	4.60	
	99.47	100.05	99.52

mineralogic characteristics. The commercial ores average twice as high in alumina as in ferric iron and on this basis there is just enough ferric iron and alumina in analysis I to take care of the silica in forming spessartite. If all the CO_2 is calculated as rhodochrosite, there is a deficiency of 4 per cent. MnO. But there is more than enough CaO and MgO to take care of this deficiency, so that roughly this rock consisted of 32.6 parts of spessartite and 59 parts of manganese carbonate by weight, or 32 and 68 parts respectively by volume. If in analysis II, the ferric oxide and alumina are calculated as manganese garnet, the CO_2 as rhodochrosite and the remaining silica as tephroite, there is a deficiency of nearly 9 per cent. MnO which is slightly more than covered by the CaO and MgO. On this basis the rock consisted of 12 parts of rhodochrosite, 14.5 parts of spessartite and 75.2 parts of tephroite by weight, or 17, 17.5 and 65.5 parts respectively by volume. Analysis III represents a garnet rock with a little more alumina and about 4 per cent. more silica than is needed to take care of the MnO, CaO, MgO and iron considered as ferrous iron. On this basis it consisted of 65 parts of spessartite, 12.6 of grossularite, and 15.3 of almandite by weight. Actually, of course, the rock consisted of a garnet intermediate in composition between the three but more nearly approaching the composition of spessartite. These calculations show the same great variation in the mineralogic composition of the Piquery rock that is found at the Morro da Mina Mine, except that rhodonite does not appear to have the prominence it attains at the latter.

These new observations necessitated different conclusions and Derby in this second paper says of the deposits that they "seem to be due to the alteration of an original rock with predominant carbonate of manganese and tephroite rather than of spessartite and rhodonite as hitherto supposed." In regard to the genesis of the rock he is silent in this paper, nor does he apply the term *queluzite* to it.

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The Morro da Mina Mine

The Morro da Mina Mine is located on a hill 2 or 3 miles north of the town of Queluz, which has an elevation of 1,110 m. above sea level and rises to a height of 200 m. above the surrounding country. A branch

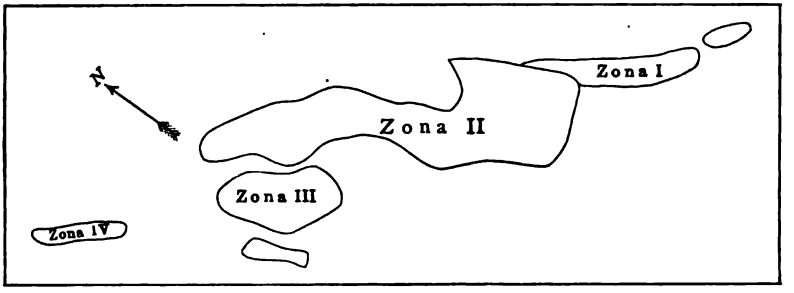

FIG. 2.—HORIZONTAL SECTION OF MORRO DA MINA OREBODIES AT LEVEL 55 M. BELOW SUMMIT OF HILL.

line of the railroad connects the mine with the Estrada da Ferra Central do Bresil at Lafayette, so that the ore can be loaded into the cars at the

FIG. 3.—SOMEWHAT DRUSY MASSIVE PSILOMELANE, THE MOST COMMON TYPE OF ORE AT THE MORRO DA MINA MINE.

mine and requires no further handling until transferred to ships for exportation at Rio de Janeiro. The mine presents one of the most remarkable manganese deposits in the world both in respect to size and quality

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of the ore; in fact, the manager of the mine, J. de A. Lustosa, says it is the largest known deposit of high-grade manganese ore. Since it became an important producer in 1902, it has yielded a total of over 1,000,000 tons, and in 1915 its production was about 200,000 tons. Development work has proved an ore reserve of 10,000,000 tons.

The orebodies occur at the top and on the flanks of the hill as a series of more or less overlapping lenses extending in a direction N 35° W., with a vertical dip and a pitch of 45° to the southeast. The relative positions, shapes and sizes of the orebodies that have been developed as they occur on the level 55 m. below the top of the hill are shown on the mine map in Fig. 2. The four largest are known as Zona I, II, III and IV respectively and have maximum dimensions of 200 by 30 m., 420

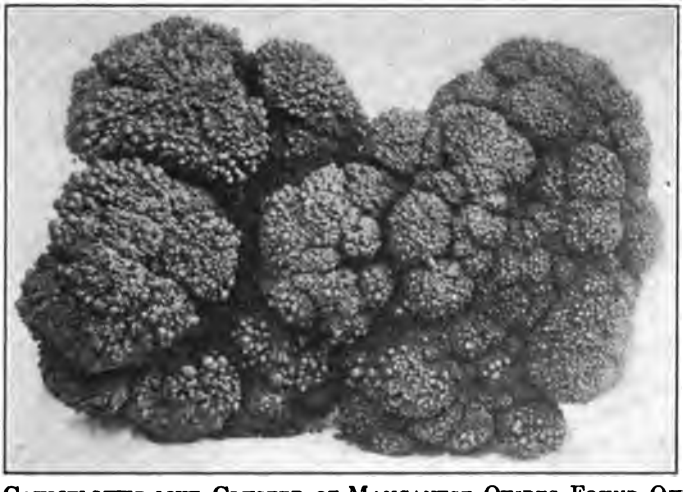

FIG. 4.—CAULIFLOWER-LIKE CLUSTER OF MANGANESE OXIDES FOUND QUITE FREQUENTLY IN THE MORRO DA MINA ORES.

by 120 m., 140 by 70 m., and 100 by 20 m. In depth Zona I and Zona II have been cut by a development tunnel 130 m. below the summit of the hill, indicating that the ores extend at least that far down. In addition to the ores *in situ*, a large part of the hillside below the ore outcrops is covered with rubble ore derived from them.

The ore consists for the most part of psilomelane, which occurs in a variety of forms. Most commonly it is simply more or less drusy massive psilomelane as in Fig. 3, but mammillary, botryoidal and concretionary forms are abundant and frequently quite elaborate. The surface of some of these, except for the black color, reminds one of cauliflower. A typical specimen is illustrated by Fig. 4. Associated with the psilomelane is considerable manganite and pyrolusite which occur for the most part as cavity linings and fillings in the former. The manganite occurs lining the cavities both in radiating groups of acicular crys-

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tals and in distinct prismatic crystals. The pyrolusite is often pseudomorphic after manganite showing that it has in part been derived from it. The average composition of the ore as shipped is:

Average Analysis of Morro da Mina Ore

Water at 100° C.	2.50 per cent.
Volatile	12.40 chiefly oxygen.
Insol. Residue	3.46
Fe_2O_3–Al_2O_3	8.75 alumina about twice ferric content.
Silica	1.76
P	0.069
S	Absent
Manganese	50.47

It is of interest to compare this average with the following mean of analyses of cargoes of manganese ores landed at Middlesborough, England, during the years 1897 to 1906 as compiled by L. L. Fermor[8] for samples dried at 100° C.:

	India	Russia
Mn	50.86	49.58
Iron	6.31	0.83
Silica	5.71	10.17
P	0.127	0.161
Alumina, etc.	6.80	12.77

These figures indicate that the three countries produce very high-grade ore with the advantage in favor of the Brazilian ores, particularly over the Russian, as regards silica and phosphorus.

The ore is mined by hand for the most part in open cuts. A number of tunnels and adits have been driven, but these were intended either for development purposes or to connect the various open cuts to give access to the loading platforms and bins at the railroad. The method in vogue is to strip the orebody of such overburden as it may carry, and then as the ore is mined screen it over iron screens with 0.8-in. square openings. The oversize is the merchantable ore. The screenings constituting 15 per cent. of the crude ore, carry 34 to 35 per cent. Mn and are being stored apart from the waste to be beneficiated at some future time. The ore *in situ* furnishes two-thirds of the present output, and one-third is derived from workings in the rubble ores.

The Original Manganese Rock

Though decomposition of the original manganese rock has extended to considerable depth in places in the Morro da Mina Mine, to at least 130 m. as demonstrated in the exploratory tunnel previously mentioned,

[8] L. L. Fermor: Manganese-Ore Deposits of India. *Memoirs of the Geological Survey of India*, vol. 37, Pt. III, p. 518 (1909).

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portions of the rock have escaped alteration and are well exposed in the mine workings. A large mass of the rock is actually exposed at the surface at the southwest corner of the outcrop of Zona II. It is characterized by a predominance of the silicates, and especially garnet, and for that reason has not succumbed to the processes of weathering. The abrupt transition from this rock to ore is shown in Fig. 5, and it is only the presence of numerous stringers, tongues and patches of manganese oxide penetrating the rock close to the ore that makes clear their genetic relations. More interesting and instructive exposures are found in the 130-m. tunnel. This tunnel crosscuts in a westerly direction the decomposed country rock for 120 m. to Zona I, then cuts diagonally across it in good ore for 150 m., and on emerging penetrates 50 m. of the manganese

FIG. 5.—SHARP CONTACT BETWEEN THE, MANGANESE ORE AND THE ORIGINAL MANGANESE ROCK CHARACTERIZED BY AN ABUNDANCE OF MANGANESE GARNET.

rock to the ore of Zona II. After again cutting 50 m. of ore, a small horse of the manganese rock is encountered with ore on the opposite side. The four contacts of ore and rock exposed in this tunnel showed the same rapid change from one to the other with only a narrow transition zone marked by stringers and tongues of the black manganese oxide penetrating the manganese rock.

Most of this rock has the appearance of a fine-grained dark gray crystalline limestone and is easily scratched with the point of the pick. Its specific gravity, however, is considerably above that of limestone. Here and there are brown patches and streaks with a violet tinge that consist of massive garnet, and there are frequently spots and stringers of pink rhodonite that at once attract attention. A closer examination

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reveals the presence of sufficient light pink silicate in much of the material to give a pinkish tone to its dominant gray color.

Examination of thin sections shows that the essential constituents are manganese carbonate, spessartite, rhodonite and tephroite. Taking the average of all the rock, the manganese carbonate is the most abundant mineral and the tephroite the least abundant. The spessartite is more widespread in its distribution but probably not much in excess of the rhodonite in actual quantity. The relative quantity of the different minerals varies most widely, so that some of the rock consists almost entirely of one of the minerals while some has them present in almost equal quantity, with the exception of the tephroite, which, in the sections examined, was never present in more than subordinate amount. In this respect the rock differs from the Piquery rock described by Derby in which tephroite was prominent and rhodonite a subordinate constituent. Which of the two minerals was formed in a given case depended undoubtedly on the relative amount of silica available, a low silica content giving rise to the formation of the orthosilicate tephroite and a higher silica content to the metasilicate rhodonite.

A thin section of a specimen most closely resembling crystalline limestone consists almost entirely of anhedral grains of manganese carbonate averaging between 0.16 and 0.25 mm. in diameter. That the carbonate is essentially manganese carbonate rather than calcium or magnesium carbonate is shown by the specific gravity of the rock, which is 3.55. The manganese carbonate is swarming with flakes of graphite ranging in size from 0.03 to 0.08 mm. and somewhat less abundantly with small garnets ranging from 0.01 to 0.03 mm. in diameter. In the entire slide there are only a few small areas containing rhodonite or tephroite. The minute garnets in this slide differ from the larger garnets in the more siliceous phases of the rock in being perfectly clear, whereas the latter are characterized by the presence of numerous minute black inclusions such as described by Derby as graphite in the rocks studied by him. Another specimen of more siliceous looking rock with a pinkish tone and streaks of rhodonite shows in thin section manganese carbonate, garnet, rhodonite and tephroite in abundance. The constituents average in size from 0.08 to 0.16 mm. with a few larger crystals of rhodonite. As is generally the case, the garnets are pronouncedly euhedral while the other minerals are anhedral. A specimen of particularly siliceous looking rock having a specific gravity of 3.8 consists for the most part of rhodonite with which is associated considerable euhedral garnet. The rock itself is light brown in color with a distinct pinkish tone. In addition to the silicates there is also an appreciable amount of the carbonate present. The garnets range in size from 0.10 to 0.15 mm. and the rhodonite crystals average about two or three times as large. Other thin sections show essentially similar features.

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The manner in which this rock undergoes alteration to manganese oxides is very interesting. A specimen undergoing oxidation is illustrated

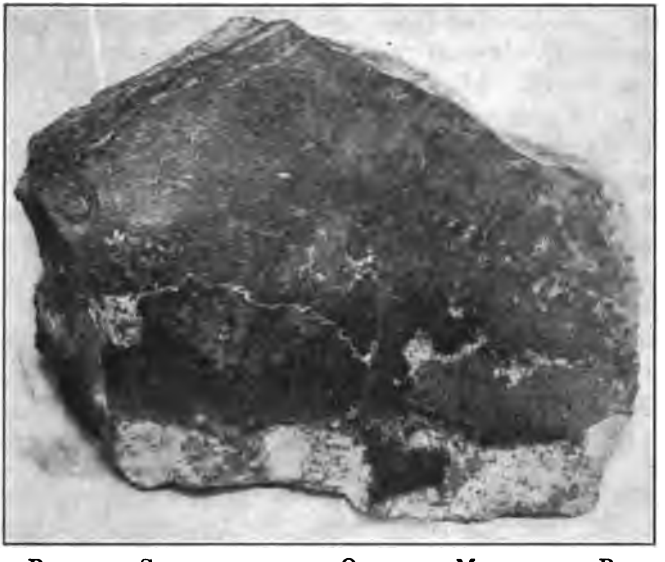

FIG. 6.—POLISHED SURFACE OF THE ORIGINAL MANGANESE ROCK IN PART ALTERED TO OXIDES OF MANGANESE. (Black areas and patches represent the manganese oxides.)

in Fig. 6. As the oxidizing solutions penetrate the rock they first break down the manganese carbonate, tephroite and rhodonite and leave

FIG. 7.—RESIDUAL GARNET GRAINS IN A GROUNDMASS OF MANGANESE ORE RESULTING FROM THE OXIDATION OF THE MANGANESE ROCK.

behind embedded in the manganese oxides the grains of garnet. That is, oxidation does not run ahead decomposing the manganese carbonate first

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and then as a more advanced stage tephroite and rhodonite successively, as one might expect; but the three minerals are replaced simultaneously at the very front of the advance of oxidation. A magnified polished surface of rock which has undergone this first stage of alteration is shown in Fig. 7, in which the garnet crystals are seen disseminated in a matrix of manganese oxides. Such material from this mine was described by Derby in his first paper and both minerals considered primary elements of the rock. In his second paper he doubted this first interpretation and was more inclined to regard the manganese oxide a residue of the alteration of original carbonate and silicate minerals with retention of the spessartite. The evidence of the thin sections makes clear that the latter is the true explanation. As this material is further worked on by the oxidizing solutions, the garnet begins to succumb around the periphery of the grains and along fractures until finally it too has been replaced by the manganese oxides.

Genesis of the Lafayette Type of Deposit

The manganese deposits of the Lafayette district inevitably call to one's mind the Indian deposits which likewise consist of manganese oxides associated with manganese silicate rocks, and as we shall have occasion to refer to them in the discussion of their genesis, it will be helpful to summarize at the outset certain of the salient features of the Indian deposits as described by L. L. Fermor.[9] There are two main types of manganese deposits in India, associated respectively with the *kodurite* and *gondite* rocks. The former are found in the Vizagapatam district on the east coast, the latter in the Central Provinces and other parts of India. Typical kodurite is composed of potash feldspar, manganese garnet and apatite, with or without pyroxene. From the mineralogical and chemical composition of the rock and its geologic relations, Fermor concludes that kodurite is an igneous rock. It was subjected to alteration under oxidizing conditions and gave rise to the manganese ores. He finds that the manganese garnet is the most stable mineral and is often left behind in a matrix of psilomelane. To the gondite rocks he gives an entirely different interpretation. He thinks they were originally deposited as sediments and the manganese which they contain as chemical sediments, most probably in the form of oxide, though perhaps to a limited extent as carbonate. These sediments were later subjected to intense dynamo-metamorphism, and where they consisted of relatively pure chemical sediments of manganese were converted into compact psilomelane and braunite. Where alumina and silica had also been deposited, spessartite and rhodonite were formed, and if there was an excess of silica it was crystallized as quartz. The resultant rock is called

[9] *Op. cit.*

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gondite. Typical gondite is a very fine-grained rock consisting of tiny round grains of spessartite set in a mosaic of quartz. Where alumina is deficient rhodonite is found. The ores associated with the gondite rocks, Fermor believes are the result of combined decomposition and replacement of them by waters containing CO_2 and O, but that this action took place at considerable depth and that the carbon dioxide was a portion of that liberated in the metamorphism of the rocks of the region and the oxygen a portion of that liberated in the conversion of the original manganiferous sediments into the manganese silicate rocks. A small proportion of softish and more or less porous ore he attributes to the later surficial alteration of the gondite.

An explanation of the genesis of the Lafayette manganese deposits involves two problems: First, the genesis of the original manganese rock; second, the alteration of it to manganese oxides, that is, to the manganese ores.

As the evidence is beyond question in regard to the second stage in the formation of these ores, that problem will be discussed first. The manner of occurrence of the orebodies, as for instance the change in depth of the Piquery ores into the manganese rock, the abundant evidence of the alteration near the surface of the manganese rock into ore and the microscopic evidence of the same phenomenon, point unmistakably to the derivation of the ores from the manganese rock under conditions of weathering in the zone of oxidation. That is, the ores were formed under conditions similar to those that gave rise to the ores of the kodurite rocks of India and not as Fermor thinks most of the ores in the gondite series were formed. The chemistry of the alteration of the Brazilian manganese rock is simple compared with that of the kodurite. The principal constituents to be removed are the CO_2 of the manganese carbonate and the SiO_2 of the silicates. Examples of the efficacy of meteoric waters for that purpose are so abundant and generally recognized that this part of the process hardly requires elaboration. The universal alteration of iron carbonates to oxides and hydroxides in the zone of oxidation and the enormous amount of silica removed from the iron silicates in the formation of the Lake Superior iron ores are illustrations of the same chemical actions on similar compounds of a chemically closely related element. In the kodurite rocks the removal of considerable alumina was also necessary and the chemistry of this was the only step that Fermor found at all difficult to explain. But even the very slight solubility of alumina in ordinary dilute meteoric waters he thought would suffice for its ultimate removal. This difficulty, however, hardly enters in our problem. Except in the case of the local phases of garnet rock, which has anyhow to a large extent resisted alteration, the alumina content of the original rock is no higher, and the few available analyses would indicate actually a little lower, than that of the ores; and, further-

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more, the total amount involved in either is not over 5 per cent. Consequently the problem of the removal of silica does not confront us.

The ores have been described as somewhat porous and drusy, but by no means to such an extent as would be called for if they merely represented the residual product of leaching of manganese carbonate and silicates. Nor is there evidence to indicate that their present more compact form is due to shrinkage in volume or compression of the residual manganese oxides; but the evidence both megascopic and microscopic shows that the decomposing solutions deposited manganese oxide, volume for volume for the silica and carbon dioxide removed. Meteoric waters encountering such large quantities of manganese carbonate cer-

FIG. 8.—VIEW OF SOUTH END OF OPEN CUT OF THE MORRO DA MINA MINE, SAID TO BE THE LARGEST KNOWN DEPOSIT OF HIGH-GRADE MANGANESE ORE.

tainly took a great deal of it into solution as manganese bicarbonate which would be deposited at other points as the oxide; and in this way the deposition of manganese oxide accompanied the leaching of the other constituents of the rock. The alteration of the manganese rock to ore was, therefore, one of simultaneous leaching and addition, resulting in a relatively compact mass of the oxides. The present drusy character of some of the ore is clearly the result of subsequent action of solution and redeposition by meteoric waters upon the manganese oxides formed in this way.

The problem of the origin of the manganese rock is not so easy to solve. One is at once confronted with the fact that our knowledge is rather meager concerning the exact nature of the rocks with which it is associated

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and its geologic relations to them, a full knowledge of which is essential to a final solution. The chemical, mineralogic and petrographic characters of the rock, however, do rule out some explanations, and point strongly to another. Further, the geologic relations that are reasonably well established are not at variance with this explanation but in a measure support it. That is, the sum of all evidence points to an analogy of this rock with the gondite of India. We believe that the rock is the product of dynamo-metamorphism of manganese sediments deposited in the form of manganese carbonate with varying but considerable quantities of silica and varying but smaller quantities of alumina. These sediments differed, therefore, initially from the gondite sediments in averaging much lower in silica, and by the deposition of the manganese in the form of carbonate instead of oxide; and this initial difference in composition accounts for the present difference in mineralogy of the two rocks.

The greater resistance to decomposition of the garnet in the manganese rock at the Piquery mine led Derby in 1901 to the erroneous conclusion that it consisted essentially of manganese garnet and to establish the rock type *queluzite* to which he attributed an igneous origin. Later, in 1908, he found that the garnet rock was but a subordinate phase of an entirely different type of rock which is essentially the same as that which we found abundantly at the Morro da Mina Mine. Attention has been called to the fact that in this later paper he does not apply the term queluzite to this rock and ventures no opinion as to its genesis. Though he did not specifically retract his earlier statement, the presumption is that, in the light of the discovery that the original rock was entirely different from what he had supposed, he abandoned his interpretation of its origin. In the meantime, L. L. Fermor,[10] on the basis of Derby's first paper, drew an analogy between this rock and his kodurite as regards their genesis, though calling attention to their dissimilarity in composition. A persistence of this same view is manifested in the statement by Harder and Chamberlin[11] in their discussion of these ores that "manganese ores associated with igneous rocks, such as those described above, occur abundantly in India—" and the statement by Harder[12] that "The relation of the manganese rock to the enclosing crystalline rocks has not been definitely determined; it may be interlayered with the gneiss or crystalline schist, or perhaps intrusive into them." The chemical and mineralogic composition of this rock as now established at both Piquery and Morro da Mina effectually precludes an igneous origin for it and demonstrates the presence of an original carbonate rock.

Even before Derby's second paper appeared, a view at variance with it

[10] *Op. cit.*, pp. 273–274.
[11] *Loc. cit.*, p. 405.
[12] *Loc. cit.*, p. 790.

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was expressed by his associate E. Hussak.[13] He regarded the spessartite rock described by Derby together with a banded spessartite-rhodonite rock, which he mentions as occurring on the periphery of the Piquery orebody, the products of contact metamorphism by the eruptive gneiss of impure manganese carbonate sediments. The postulation of the derivation of the silicates from manganese carbonate in advance of the discovery of the presence of large amounts of carbonate is interesting. Beyschlag, Krusch and Vogt[14] are inclined to carry Hussak's ideas a step further and suggest that the manganese may have been contributed as part of the process of contact metamorphism. That this was not the case is proved by the fact that the carbonate still present, constituting a large percentage of the rock, is the manganese carbonate; and that consequently the original carbonate was a manganese carbonate. There remains to decide between the view of Hussak that the rock is the result of contact metamorphism of sedimentary beds of impure manganese carbonate and our own that it is a product of dynamo-metamorphism of such beds.

In the first place, the country rock of the ore deposits is not everywhere gneiss, but is in some cases schist, and these schists are admitted to be metamorphosed sediments. Nor are all of the gneisses necessarily igneous, though it has been generally held that they are such for the most part. Consequently the presumption is against assuming an igneous contact for all of the deposits. In the second place, the texture and appearance of the silicates is not that of typical skarn minerals, or products of contact metamorphism; but it is that of silicates crystallized in a carbonate rock that has been subjected to regional metamorphism. The reply might be made that the whole region has been subjected to dynamo-metamorphism subsequent to the contact metamorphism, but it is equally true that the texture of the rock is not what one would expect to result from a skarn rock subjected to regional metamorphism. On the other hand, we cannot escape the conclusion, demanded by both hypotheses, that there first existed impure manganese carbonate, and it is obvious that the rocks have undergone regional metamorphism. The inevitable result would be a rock such as we have. Our explanation is, therefore, in harmony with the available evidence and makes the manganese rock of the Lafayette district genetically identical with the gondite of India, the mineralogic difference of the two rocks being due to their initial difference in chemical composition.

[13] Eugen Hussak: Ueber Atopit aus den Manganerzgruben von Miguel Burnier, Minas Geraes, Brasilien. *Centralblatt für Mineralogie, Geologie und Paläontologie*, 1905, pp. 240–245.

Eugen Hussak: "Über die Manganerzlager Brasiliens. *Zeitschrift für praktische Geologie*, vol. 14, pp. 237–239 (1906).

[14] Beyschlag, Krusch, Vogt: *Die Lagerstätten der nutzbaren Mineralien und Gesteine*, vol. 2, p. 596 (1913).

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 39 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Recent Geologic Developments on the Mesabi Iron Range, Minnesota

BY J. F. WOLFF, E. M.*, DULUTH, MINN.

(New York Meeting, February, 1917)

DURING the past 4 or 5 years, much has been added to the detailed geologic knowledge of the Mesabi Range. This has not been in the direction of discovery of any new fundamental facts, but of detailed study, subdivision and correlation of different parts of the whole formation and of individual orebodies. Prior to this time, mining engineers in the district were so engaged with the commercial interests of exploring and developing orebodies that close geologic study and subdivision was done in only a few instances. The demand for refinements of work in this direction has caused extensive structural subdivision and correlation to be done in all exploration work during the past 4 years by the engineers of the Oliver Iron Mining Co. Such work has become a commercial necessity rather than a scientific refinement, and at the present time is being extended to all parts of the Range.

In the summer of 1914, the author of this paper wrote a series of articles on the orebodies and special features of the Mesabi Range, which was published in the *Engineering and Mining Journal*, issues of July 17 to Aug. 7, 1915. Since that time studies of subdivision of the iron-formation have been extended considerably. The principle feature of this paper is the presentation of the subdivisions of the iron-formation. To this is added a discussion of the relations of the orebodies to the gentle folding and fracturing of the formation, and special features of the Range.

The outline will be as follows:

I. General geology.

II. Subdivision of Biwabik iron-formation.

III. Relation of orebodies to folding and fracturing of the iron-formation.

IV. Special features.

Acknowledgment is due to W. J. Olcott, President, and John Uno Sebenius, General Mining Engineer, of the Oliver Iron Mining Co., for permission to use the information presented in this paper.

* Mining Engineer, Oliver Iron Mining Co., Duluth, Minn.

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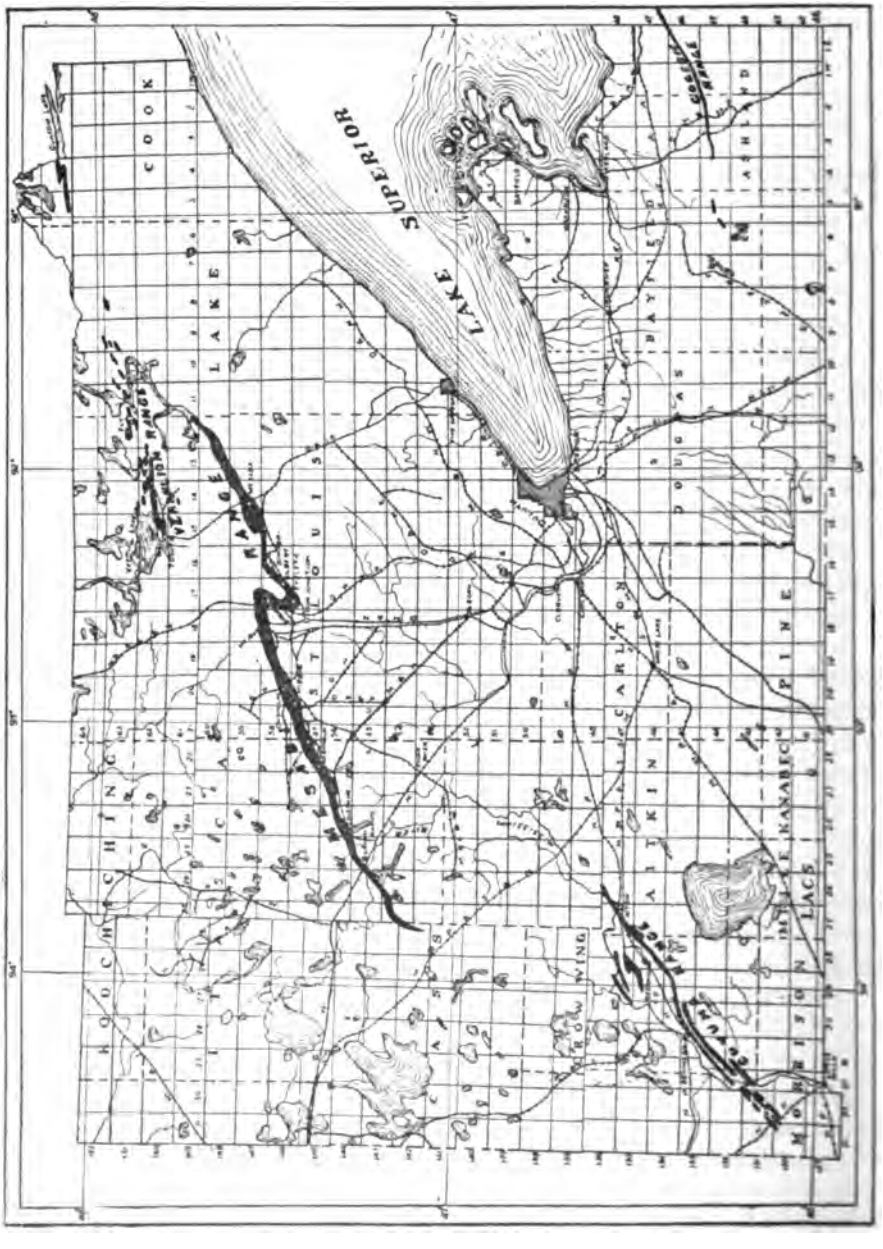

Fig. 1.

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General Geology

Correlation

Although the general geology of the Mesabi Range has been described most exhaustively in *U. S. Geological Survey Monographs* Nos. 43 and 52, and elsewhere, a short synopsis may be a necessary preliminary to the detailed discussion of the Range presented in this paper, because many of the readers of this article may be unfamiliar with the above publications or the general geology of the Mesabi district.

The topographic feature of the Mesabi Range is a line of fairly prominent and continuous hills ranging in elevation from 1,400 to 1,900 ft. above sea level, composed of a complex of granites, greenstones, green schists, slates, graywacke and conglomerate. Resting unconformably upon this basement complex, and sloping away at a gentle angle to the southeast, is a series of sedimentary rocks, the middle member of which is iron-bearing. The outcrop of this iron-bearing member is the geologic feature known as the Mesabi (or Missabe) Iron Range. Within this formation the iron ore-bodies are found. Its outcrop has been traced by explorations from Sec. 12, T. 142 N., R. 25 W., northeastward to Birch Lake, in Sec. 26, T. 61 N., R. 12 W., a distance along the strike of about 112 miles. Its width varies from ⅜ to 3 miles, due to variation in the dip and thickness. The sedimentary series above mentioned consists of a basal quartzite, named Pokegama, an intermediate iron-formation, named Biwabik, and a top black slate, named Virginia slate.

The location and general extent of the Range is shown on the Map, Fig. 1. The general relations of the iron-bearing member to associated rocks are shown by the cross-section, Fig. 2, better than further description could explain.

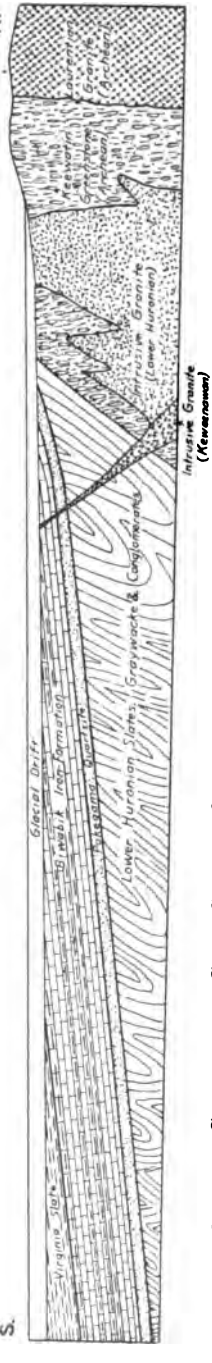

Fig. 2.—Generalized Cross-Section Showing Relation of Iron Formation to Associated Rocks.

Instead of the complete correlation table as given by the U. S. Geological Survey, a simplified geologic column is herewith given, which is adequate for the engineer in the district.

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Quaternary System		
Pleistocene Series	Glacial drift, 0 to 300 ft.	
Unconformity.		
Cretaceous System	Conglomerates and shales, 0 to 50 ft.	
Unconformity		
Algonkian System, Keweenawan Series	Duluth Gabbro	East end of range.
	Embarrass Granite	
Unconformity		
Upper Huronian Series (Sedimentary)	Virginia slate 0 ft. to great thickness. Biwabik Iron-formation, 475 to 775 ft. Pokegama Quartzite, 50 to 150 ft.	
Unconformity		
Algonkian and Archean Series	Basement complex of Slate-graywacke-conglomerate series, granites, greenstones, greenschists and porphyries.	

From a commercial standpoint the Upper Huronian series, Virginia Slate, Biwabik Iron-formation, Pokegama Quartzite are the important rocks of the district.

The characteristics of the different formations need not be discussed. This will be done for the iron-formation later on.

Orebodies

The Upper Huronian Quartzite-Iron-formation-Slate series is an ordinary (except for the iron) series of clastic sediments, deposited in fairly shallow water, contemporaneous with the middle member of which was deposited or precipitated out of solution an enormous quantity of iron, chiefly in the form of a ferrous silicate, $FeSiO_3$, occurring as coarse and minute green granules of the mineral greenalite, embedded in a matrix of chert. Iron carbonate, grunerite, amphibole, actinolite, and hematite occur in small quantities. Original magnetite in bands and finely disseminated grains occurs in much larger quantity than has been recognized heretofore, in all parts of the district. The bedding-planes of all three members are approximately parallel. Interbedded with the iron-formation in certain horizons are numerous slate layers, varying in thickness from a few inches to many feet. The iron-formation from Virginia slate to quartzite varies in thickness from 475 to 775 ft., the average being about 600 ft.

After this series of sediments had been laid down, earth movements raised them above water level, allowing erosive agencies to cut through the overlying slate into the underlying formations. These earth movements warped the formations and cracked the brittle iron-formation quite extensively, allowing surface waters to enter its upturned edges. Especially where such cracking was pronounced, the ground-waters entering the iron-formation, carrying carbon dioxide in solution, attacked the ferrous iron compounds and oxidized them. In such localities much of the ferrous silicate has been changed to a hematite-chert rock, the hematite occurring as bands and as disseminated particles in the chert, the rock still retaining its solidity. The whole iron-formation, whether

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thus altered or not, is a ferruginous chert with interbedded slate layers and locally is called "taconite." This term should apply strictly to the ferruginous chert and not to the slate layers, though in some horizons the slate bands and chert layers are so intimately interbedded as to make a distinction quite impossible.

Where the cracking has been most intense the circulation of ground-water has been most vigorous, the solvents in the ground-waters have leached out the silica and other minor constituents of the rocks and have oxidized and left in place the iron. Such residual material now constitutes the orebodies. They are surrounded on all sides by the rock walls of the iron-formation from which they are derived. Pore-space was developed by this removal of silica and the settling or slumping in place of the layers of the orebodies is a characteristic feature. The typical orebody thus developed has a trough structure and an irregular trough shape. Orebodies vary in size from a few acres to several hundred acres, and from a few feet to several hundred feet in thickness. In many

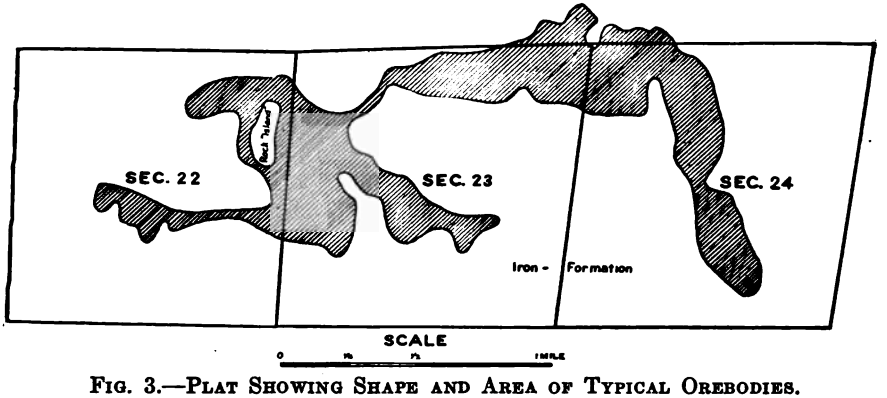

FIG. 3.—PLAT SHOWING SHAPE AND AREA OF TYPICAL OREBODIES.

places small troughs unite to form a large one. Fig. 3 shows the shape and area of a typical orebody. While the trough orebody is the typical one, there are two other types, namely, the flat-layered body and the fissure-type orebody. The former is either the remnant left by the erosion of a former trough-body or it is an ore layer continuing down the dip from a trough-body. Usually such a layer has a rock (slate) capping. The fissure-type orebody is an incompletely developed trough-body and is usually associated with a larger trough orebody. Fig. 4 shows the development of a trough orebody on the axis of an anticline, where cracking has been pronounced. All four stages here shown can be observed in the field. Notice the slumping of ore layers near the rock walls. The features of the three different types of orebodies were discussed at length in the *Engineering and Mining Journal*, July 17 to Aug. 7, 1915, and will not be repeated here.

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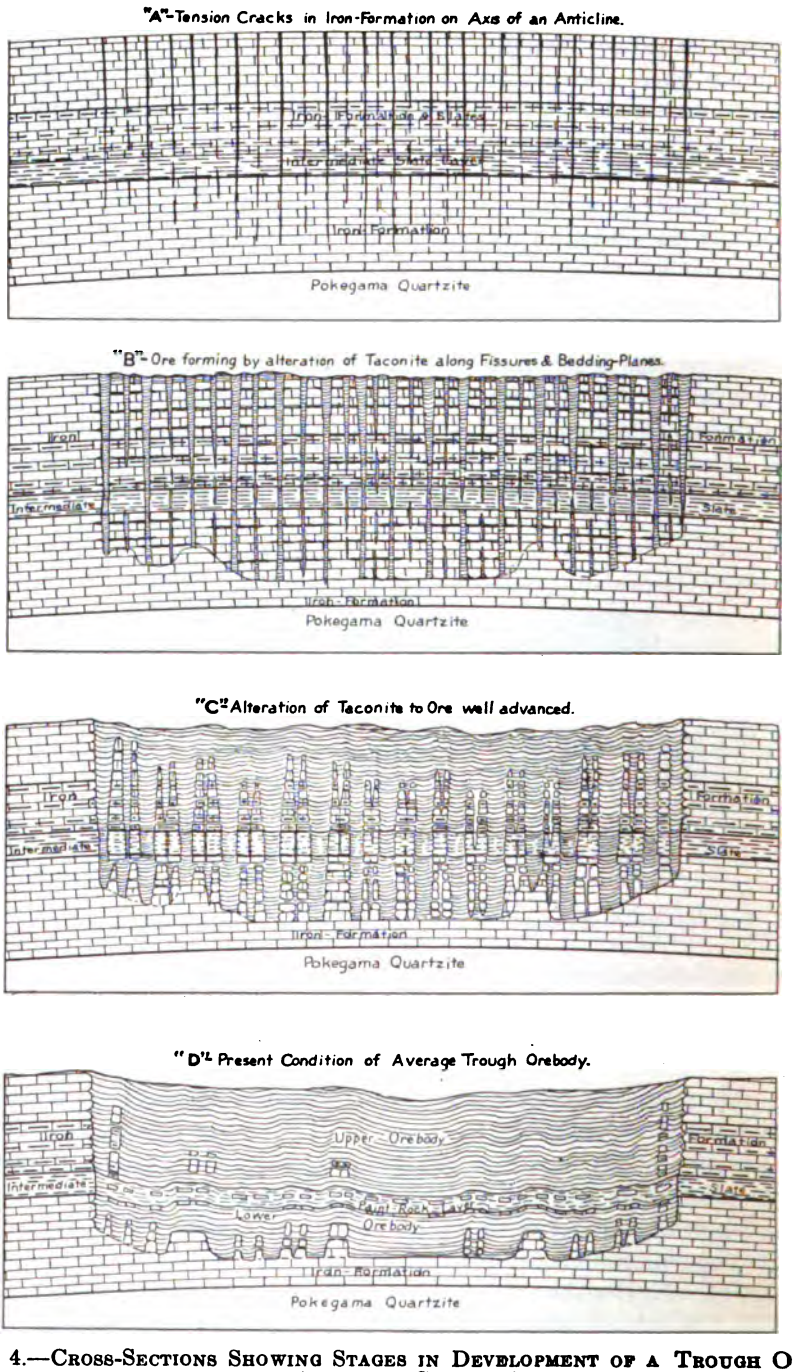

FIG. 4.—CROSS-SECTIONS SHOWING STAGES IN DEVELOPMENT OF A TROUGH OREBODY ON THE AXIS OF A GENTLE ANTICLINE.

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Covering the entire iron-formation, except at a few isolated spots, is a mantle of glacial drift, varying in thickness from a few feet to 300 ft.

Subdivisions of the Iron-Formation

Detailed Subdivision

The item of principal interest and most practical value to mining men in the Mesabi district, presented in this paper, is the subdivision of the iron-formation into its several horizons. As indicated above, the engineers of the Oliver Iron Mining Co. have found it absolutely essential to do this kind of work in the exploration and development of their orebodies. Undoubtedly others will meet with the same experience. The determination of the structure of an orebody depends upon the proper subdivision and correlation of the different ore and rock layers in the drill records.

Such correlation, not only in one area but over the length of the Range, has shown that the formation can be subdivided into four principal horizons, each of which has remarkably uniform characteristics from one end of the Range to the other. Each horizon can be subdivided further. Of course, the thickness of each horizon is not uniform because the thickness of the whole formation varies. There are local peculiarities in some places, but these do not destroy the major subdivisions. This subdivision and correlation has been done chiefly by F. B. Cronk, Mining Engineer for the Oliver Iron Mining Co., and the writer.

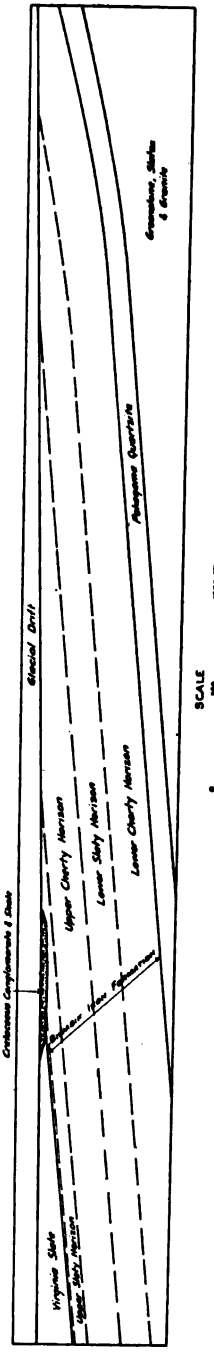

Fig. 5.—Generalized Cross-Section Showing Subdivision of the Biwabik Iron-Formation.

Fig. 5 shows the four subdivisions which can be made in any part of the Mesabi district. From the top down they are, Upper Slaty Horizon, Upper Cherty Horizon, Lower Slaty Horizon and Lower Cherty Horizon. They are named from the predominant physical characteristic of the rock in them. Commercially the two cherty horizons and Lower Slaty Horizon are most important, as the principal orebodies occur in them. The two cherty horizons contain very few slate seams, but the slaty horizons contain a considerable number of interbedded chert layers and a great amount of greenalite. The lower

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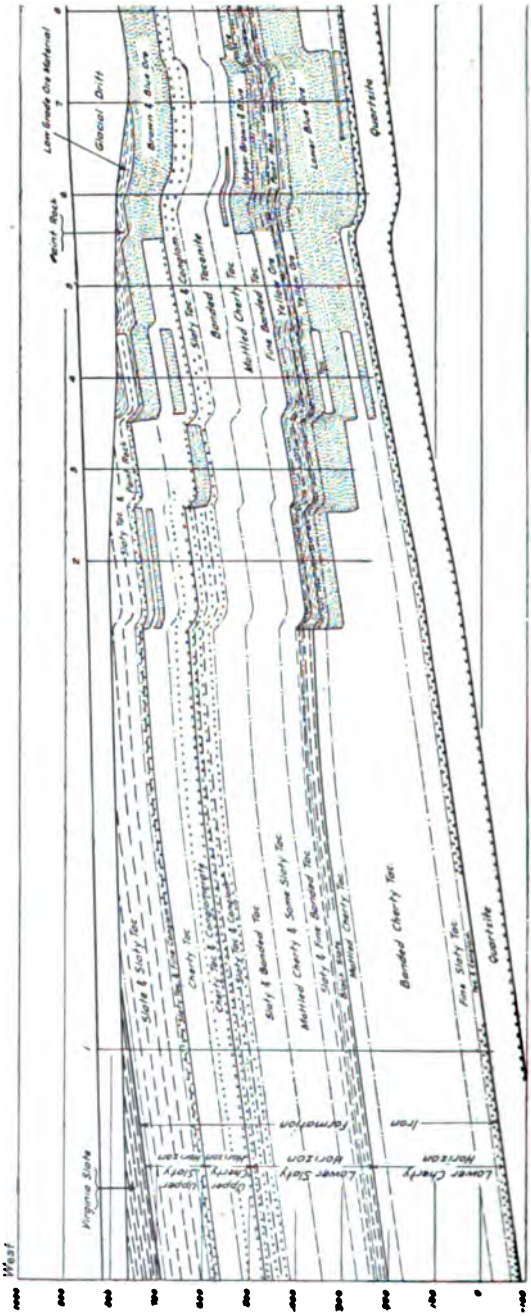

FIG. 6.—LONGITUDINAL CROSS-SECTION OF A LARGE TROUGH OREBODY SHOWING ORE DERIVED FROM ALL HORIZONS OF THE IRON FORMATION. (*Continued on two succeeding pages.*)

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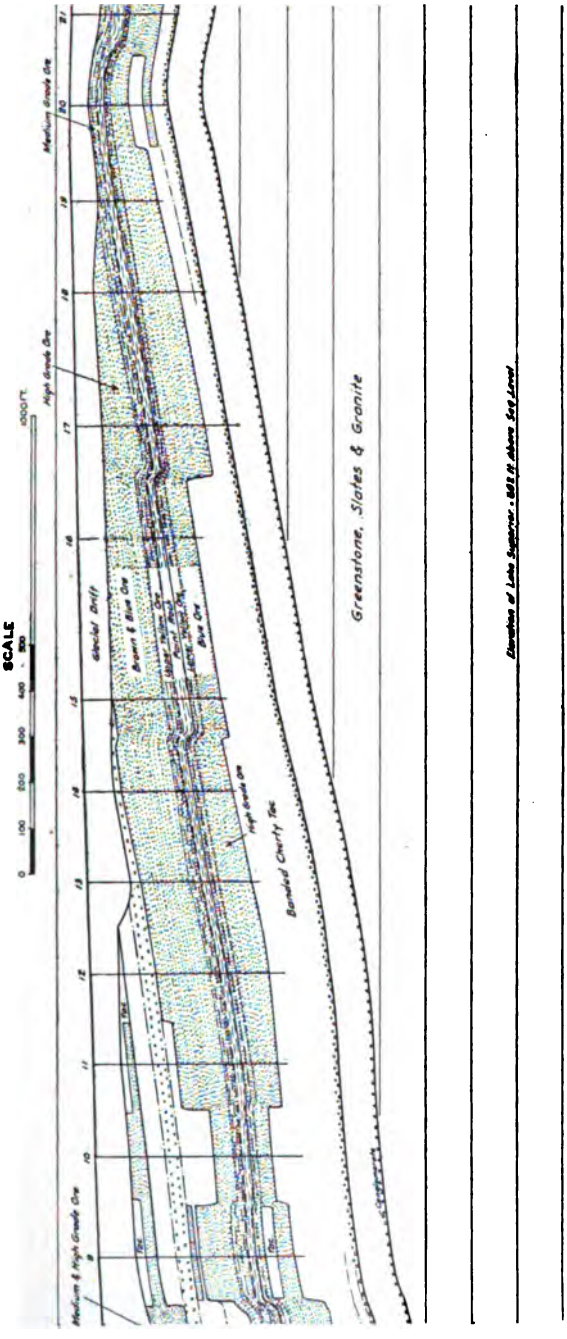

FIG. 6.—(*Continued.*)

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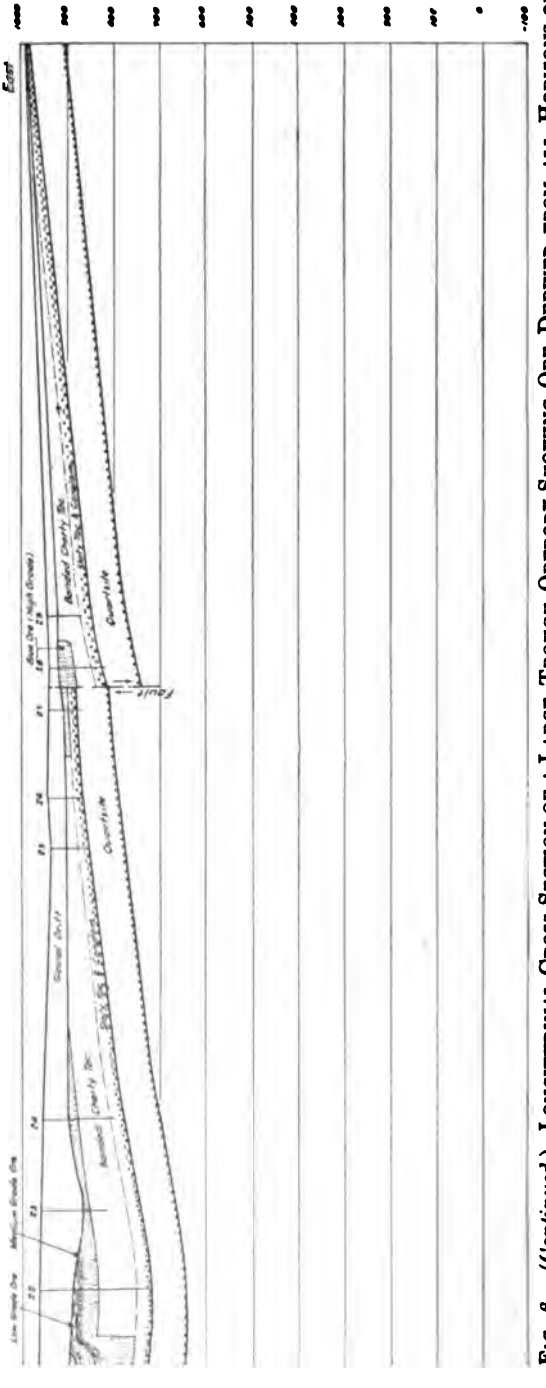

FIG. 6.—(*Continued.*) LONGITUDINAL CROSS-SECTION OF A LARGE TROUGH OREBODY SHOWING ORE DERIVED FROM ALL HORIZONS OF THE IRON FORMATION.

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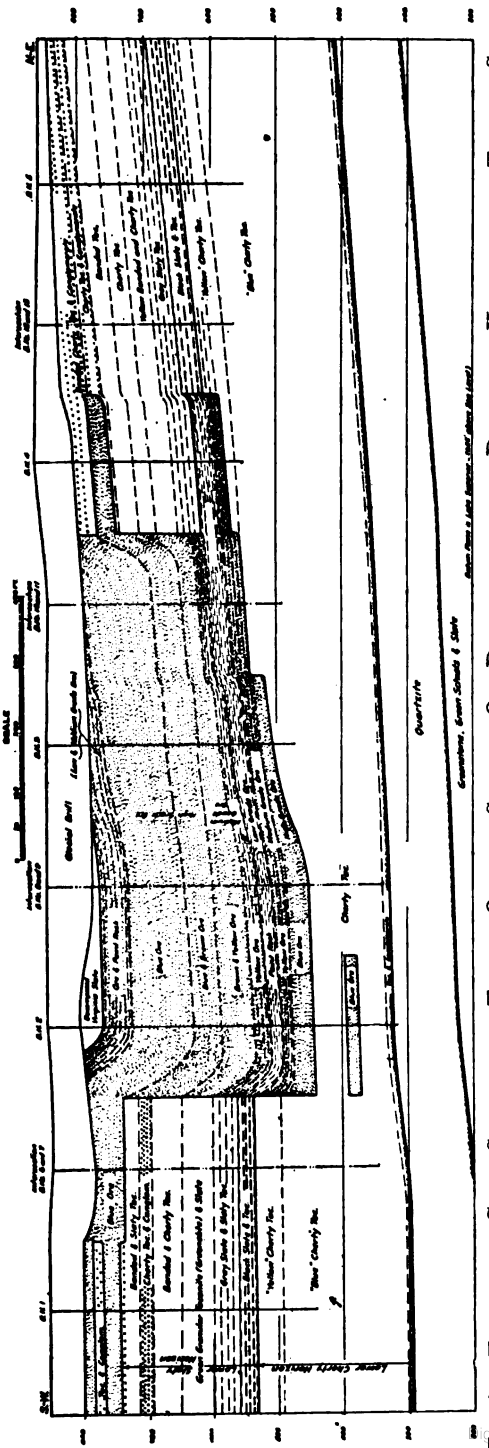

FIG. 7.—TRANSVERSE CROSS-SECTION OF A TROUGH OREBODY SHOWING ORE DERIVED FROM THE DIFFERENT HORIZONS AND TROUGH STRUCTURE OF OREBODY.

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half of the Lower Slaty Horizon contains the most slate, the bottom 30 ft. or so being almost pure slate. But the upper half is slaty or banded in structure rather than in composition of the rock layers. The subdivisions are made by eye rather than on any chemical basis or microscopic examination.

The Upper Slaty Horizon is composed about half and half of slate layers and greenalite lenses. The upper part approaching the Virginia slate is the more slaty. It is separated from the Virginia slate by a layer 10 or more feet thick of a carbonate rock, probably an impure limestone.

Two cross-sections, Figs. 6 and 7, are presented to show in detail the four main subdivisions, the minor subdivisions of each, the relations of the orebodies to these horizons and the kind of ore derived from each.

Fig. 6 shows an east-west cross-section, looking north, through the Adams, Hull, Nelson, Leonidas orebodies just west of Eveleth. It is the best cross-section that has been or can be made from present explorations, so far as the author knows. The location of the section is shown on the accompanying map, Fig. 11, and is approximately at right angles to the strike. It is developed from drill-hole classifications, as can be seen, but practically the entire section of ore is being developed now by open-pit and underground workings. Because this cross-section is taken along the longitudinal axis of the ore trough, the trough structure does not show. In the area of drill holes 6 to 8 inclusive, a north-south trough is tributary to the east-west trough through which this cross-section is taken, and the trough structure is apparent. The gentle warping of the entire series can be noted. The principal warping is at right angles to the strike, however.

The relations between the different horizons and their derivative ores are so evident from the cross-section that a detailed description need not be given. In the area of drill holes 2 to 15 inclusive, the taconite in the Lower Slaty and Upper Cherty Horizons is so badly altered that classification is very difficult.

A few features may call for comment. The black slate indicated at the base of the Lower Slaty Horizon is the so-called "Intermediate" Slate, which makes the characteristic paint rock layer of the typical Mesabi orebody. The orebodies first developed on the Range were located in that part of the formation shown between drill holes 15 and 24; therefore, it is easily seen that the typical orebody had five layers—upper and lower blue ore layers, upper and lower yellow ore layers, and an intermediate paint rock layer.

Interbedded in the Upper Cherty Horizon and the top of the Lower Slaty Horizon are distinct conglomerates. At the base of the Upper Slaty Horizon there is also a fine interbedded conglomerate. The Upper Cherty conglomerate is continuous from the far eastern part of the Range

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to the western part. It was recognized in drill cores only a few years ago by W. H. Crago, head of the exploration division of the Mining Engineering Department of the Oliver Iron Mining Co. All of the earlier drill cores have not been reëxamined for it as yet, but it has been correlated extensively in different parts of the Range by F. B. Cronk and the writer. The cross-section in Fig. 6 shows a maximum development of interbedded conglomerate. It is not as thick either on the western or eastern end of the range. On the eastern part it is distinctly developed in both Upper Cherty and upper part of Lower Slaty Horizons, but in the west central part of the Range only the Upper Cherty conglomerate has been recognized, and it is quite thin. This section (Fig. 6) shows practically a maximum thickness of iron-formation. The Lower Slaty Horizon is abnormally thick (260 ft.) whereas the average thickness is only about 150 ft. In the central and western part of the Range, the Upper Cherty Horizon is exceptionally thick, the slaty layers of the upper part of the Lower Slaty Horizon being replaced by cherty and banded taconite. Evidently, more muds were deposited with the iron-formation in the east central part of the district than in the central and western parts of the district. Perhaps the underlying rocks outcropping to the north are accountable for this. From Mt. Iron east to Aurora, large areas of greenstones and slates lie to the north of the iron-formation. If the original shore line of the sea in which the iron-formation was laid down occupied approximately its present position, the weathering and erosion of these rocks contributed muds and argillaceous sediments to the sea water contemporaneous with the deposition of iron.

Fig. 7 shows a cross-section across a typical trough orebody in the Virginia district. The Upper Cherty and Slaty Horizons have been eroded from the sides of the rockwalls of the orebody, but all horizons from the Virginia slate down are represented in the orebody. The typical trough structure is well shown.

Records and Subdivisions of Drill Holes in Different Districts

Fig. 8 shows records and subdivisions of drill holes in the Nashwauk, Hibbing, Virginia, McKinley and Aurora districts, comprising the territory from the west central to the eastern part of the range. These records all show the same major divisions, though varying in the minor subdivisions and dimensions. The interbedded conglomerate in the Upper Cherty Horizon is persistent in all the districts.

Ores Derived from the Different Horizons

The characteristic ore derived from the Lower Cherty Horizon is a coarse "blue" high-grade ore. It contains practically no paint rock

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seams. About 30 or 40 ft. of the top of this horizon is a brown or yellow mottled cherty taconite (originally a greenalite rock) containing some

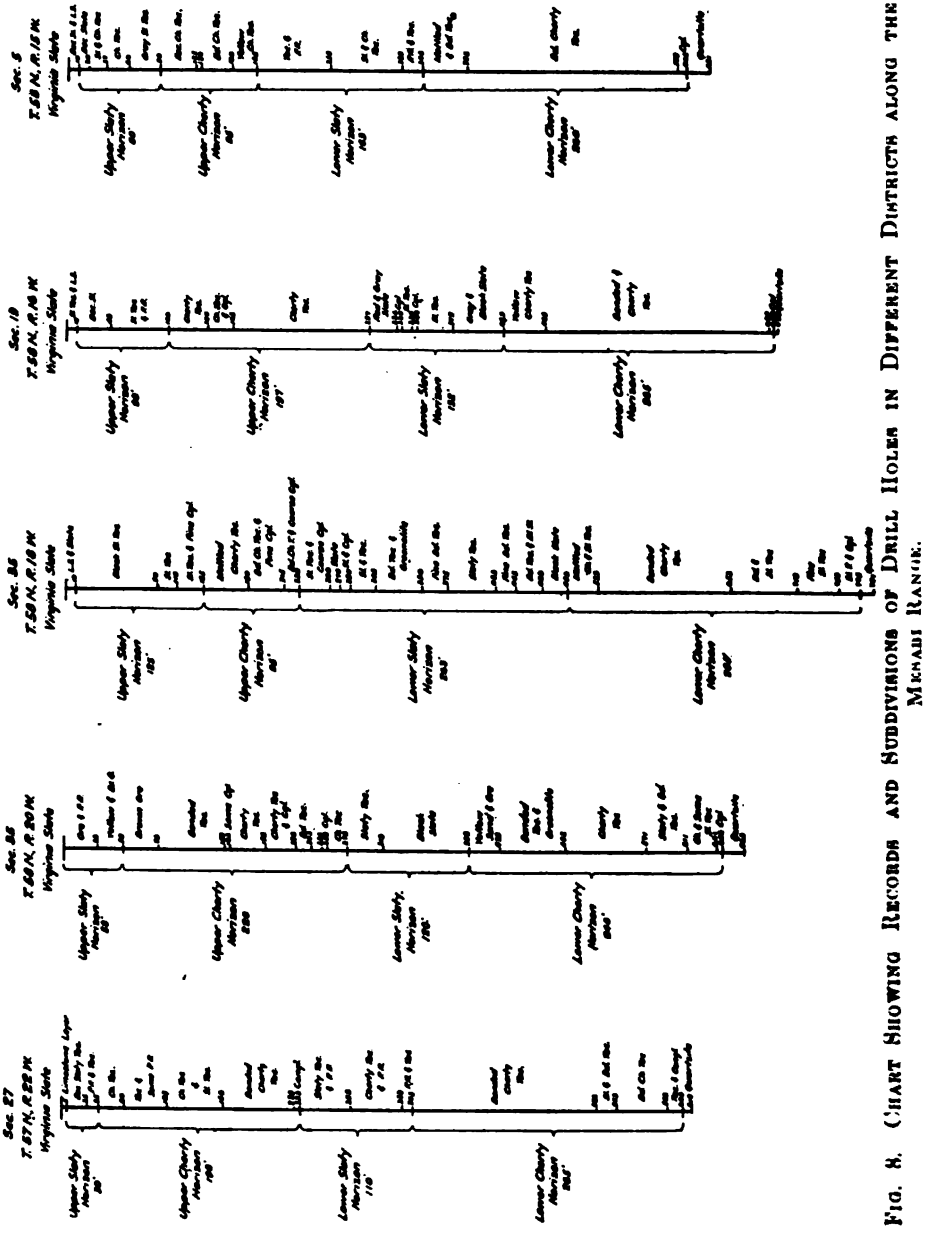

FIG. 8. CHART SHOWING RECORDS AND SUBDIVISIONS OF DRILL HOLES IN DIFFERENT DISTRICTS ALONG THE MESABI RANGE.

slate seams in its upper measures, and this layer makes a yellow ocherous ore of medium grade. At the base of the Lower Cherty Horizon just

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above the basal conglomerate is a layer of fine slaty taconite (see Fig. 5) which makes a yellow ocherous ore. With the exception of these top and bottom layers, the Lower Cherty Horizon makes a "blue" high-grade ore. As used here, high-grade ore means ore averaging above 59 per cent. dry iron, medium-grade averaging 55 to 56 per cent. dry iron, and low-grade averaging about 50 per cent. dry iron.

The characteristic physical feature of ores from the Lower Slaty Horizon is their finely banded and slaty texture. As previously indicated, the black slate at the base makes the paint rock layer so prominent in every typical orebody. This material is not a commercial ore. It is highly aluminous and contains 20 per cent. or more of moisture. The gray slate and greenalite and slate above this black slate (Fig. 7) make a medium-grade yellow and brown ore, the yellow ore being quite aluminous. The banded-cherty and banded-slaty taconite of the top of the Lower Slaty Horizon make a very fine-grained blue and brown ore of high grade. It was this kind of ore which was so objectionable to furnace men because of excessive fines, in the early days of the Mesabi Range exploitation.

The Upper Cherty Horizon makes a high-grade coarse blue ore, in all of the large well-concentrated bodies. It is indistinguishable in texture from the blue ore of the Lower Cherty Horizon.

In some orebodies toward the south side of the formation, such as the Morton Mine in the Hibbing district, the Duncan in the Chisholm district, the Leonidas in the Eveleth district and the Schley and Hobart in the Gilbert district, the ore in this horizon is somewhat sandy and cherty, due in part to incomplete concentration and in part to secondary silica deposited by ground waters.

The Upper Slaty Horizon generally makes a low-grade non-merchantable ore. The one known exception to this is shown in Fig. 7, in which orebody most of this ore probably will be merchantable. This orebody, however, is one of the most highly concentrated on the Range. In most orebodies which have this Upper Slaty member, the material from it is a soft plastic paint rock with decomposed chert and greenalite layers. It resembles very much the so-called "Intermediate" paint rock layer.

From the cross-section, Fig. 6, it is evident that all orebodies will not contain all of these horizons or layers. The unit of land subdivision is a 40-acre tract and many mines occupy one or a part of one such tract. If a mine is located near the quartzite outcrop, most of the upper horizons will be eroded away, and as the mine location approaches the Virginia slate outcrop more of the upper layers will be found in the orebody.

Relation of Orebodies to Folding and Fracturing of the Iron-Formation

In the *Engineering and Mining Journal* articles above referred to, the author stated that the data then at hand indicated that the orebodies

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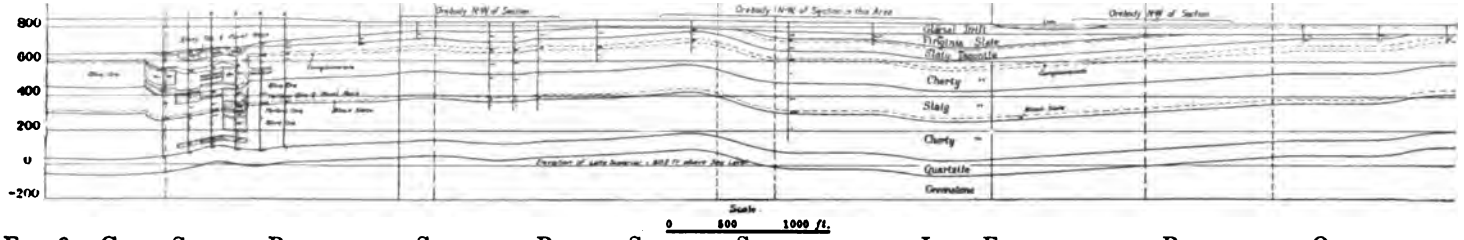

FIG. 9.—CROSS-SECTION PARALLEL TO STRIKE OF RANGE, SHOWING SUBDIVISIONS OF IRON FORMATION AND RELATION OF OREBODIES TO GENERAL STRUCTURE.

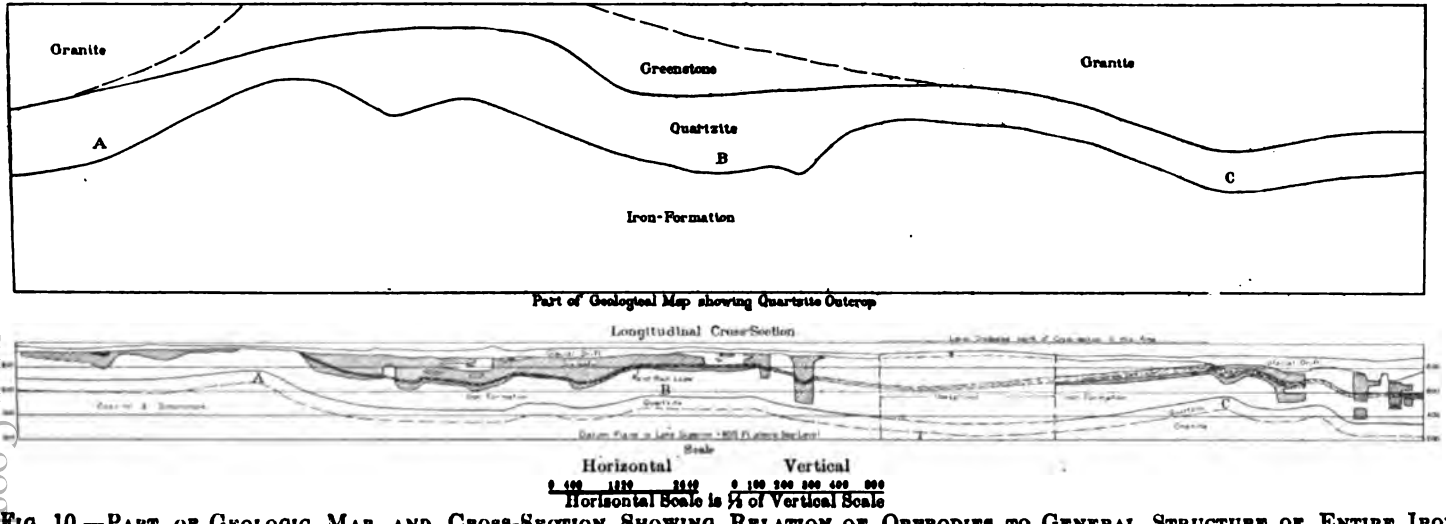

FIG. 10.—PART OF GEOLOGIC MAP AND CROSS-SECTION SHOWING RELATION OF OREBODIES TO GENERAL STRUCTURE OF ENTIRE IRON FORMATION IN HIBBING-CHISHOLM DISTRICT.

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have formed in places where gentle folding and warping of the iron-formation had fractured it considerably, allowing easy access and circulation of ground-waters. Evidence along this line has been assembled since the publication of those articles and in every case where the exploration data is complete enough it has been found that the orebodies occur where the whole formation has been warped. In the eastern part of the district (Virginia and eastward) the orebodies are on the crests of gentle anticlines. Fig. 9 shows a cross-section parallel to the strike of the formation in T. 58 N., R. 16 W., location of which is shown on the map in Fig. 11. This section is taken well to the south side of the formation and only one orebody reaches it. But the locations of other orebodies northwest of it are shown on the cross-section, and in each case where the exploration data is complete enough to show it, the orebody is located on the axis of an anticlinal flexure or combined anticline and syncline.

In the central part of the Range, great broad flexures rather than merely localized ones seem to have determined the locations of the orebodies. The formation was very generally cracked up and the broad structural basins directed the flow of underground waters. Fig. 10 shows a structural cross-section taken midway between Virginia slate and quartzite outcrops approximately parallel to the strike, through the Hibbing-Chisholm districts. A part of the quartzite outcrop to the northwest is also shown. Three prominent anticlines, *A*, *B* and *C*, with two intervening synclines, are shown. The cross-section is taken about a mile south of the quartzite outcrop. It has been published previously in the *Engineering and Mining Journal*, Aug. 7, 1915. It shows that the orebodies occur quite continuously over both crests of anticlines and troughs of synclines, indicating a very general fracturing of the formation, vigorous circulation of ground-water and consequent complete alteration and concentration of iron-formation. These major flexures can be determined only by such correlation and drawings as are shown in the two cross-sections, Figs. 9 and 10, but the minor flexures within the larger ones often can be observed in the field.

Special Features

Special features of the Mesabi Range may be of interest and deserving of inclusion in this paper.

Faults

The two principal faults known to date on the Mesabi Range are those known as the Biwabik and Alpena faults, both of which were described in the *Engineering and Mining Journal*, July 24, 1915. Mention is made of them here only because they have been followed further since that time. Fig. 11 shows the location of both. The Biwabik

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fault has been traced to the NW of SE, Sec. 5, 58–15, where it practically disappears. It is a hinge-type gravity fault, the south side of which has been depressed. The greatest throw is at its west end at the Biwabik Mine, Lot 4, Sec. 2 and Lot 1, Sec. 3, Tp. 58 N., R. 16 W., where the vertical displacement exceeds 200 ft. The underlying greenstone is faulted up against the ore, though the faulting probably occurred prior to the formation of the ore. Fig. 12 is a cross-section of the Alpena ore-body north of Virginia, showing the largest fault known on the Range. The location of the cross-section is shown on Fig. 11. As indicated, the strike of the fault is approximately north and south. It is a faulted thrust-fold, the probable development of which is shown by Fig. 13.

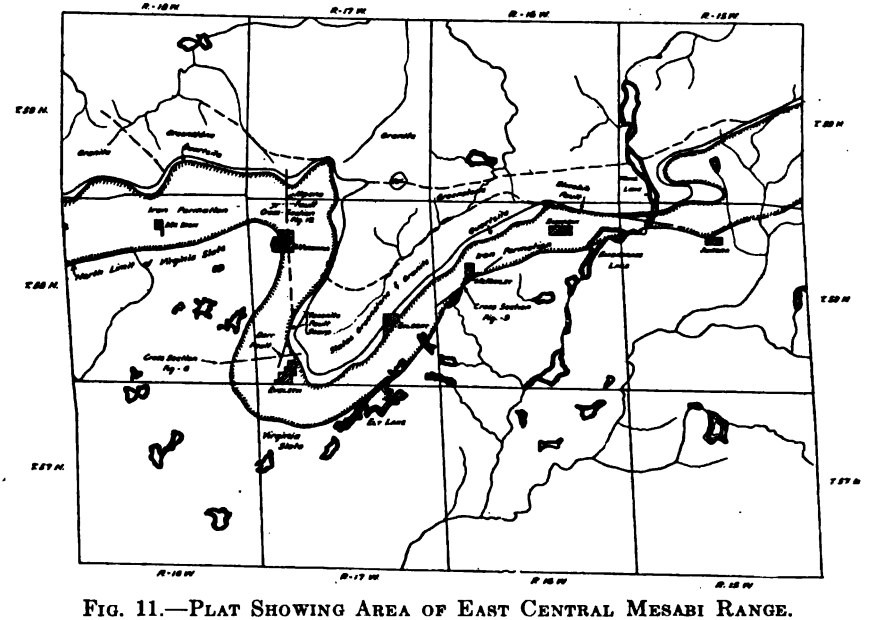

FIG. 11.—PLAT SHOWING AREA OF EAST CENTRAL MESABI RANGE.

The genesis of the fault is directly connected with the gentle uplift or crustal warping which produced the large Z-shaped bend in the Range known locally as the Virginia "horn." This was discussed also in the *Engineering and Mining Journal*, July 24, 1915. It is repeated here because since that series, a fault (shown on the east end of cross-section in Fig. 6) has been discovered by correlation of drill holes. The location of this fault, which may be called the Dorr fault from the property on which it is located, is shown on the map, Fig. 11, and is of the same type (reverse or thrust) as the Alpena fault. It is almost directly south of the Alpena. Between the two and about ¾ mile north of the Dorr fault is a taconite bluff, the east side of which is a very steep wall, undoubtedly a fault scarp. It is of the same type as the Dorr and Alpena faults.

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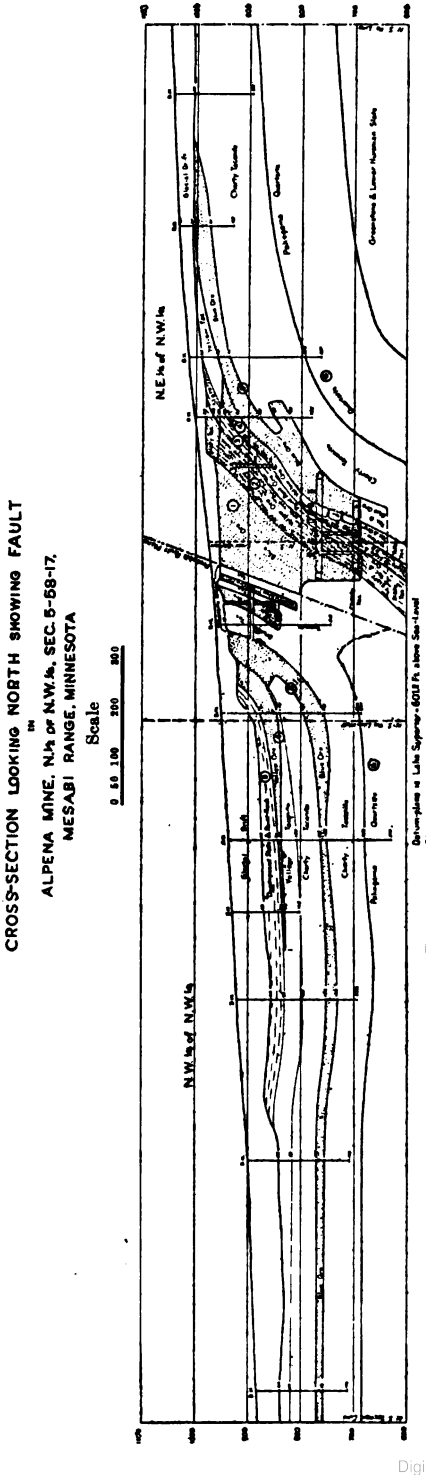

FIG. 12.—SHOWING ALPENA THRUST-FAULT.

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Although we have as yet no complete exploration data, it appears that the Alpena and Dorr faults and the intervening escarpment are one continuous fault produced by the crustal movements which caused the Virginia "horn." This probable connection is indicated on Fig. 11. However, it is possible that the Alpena fault is entirely separate from the Dorr fault.

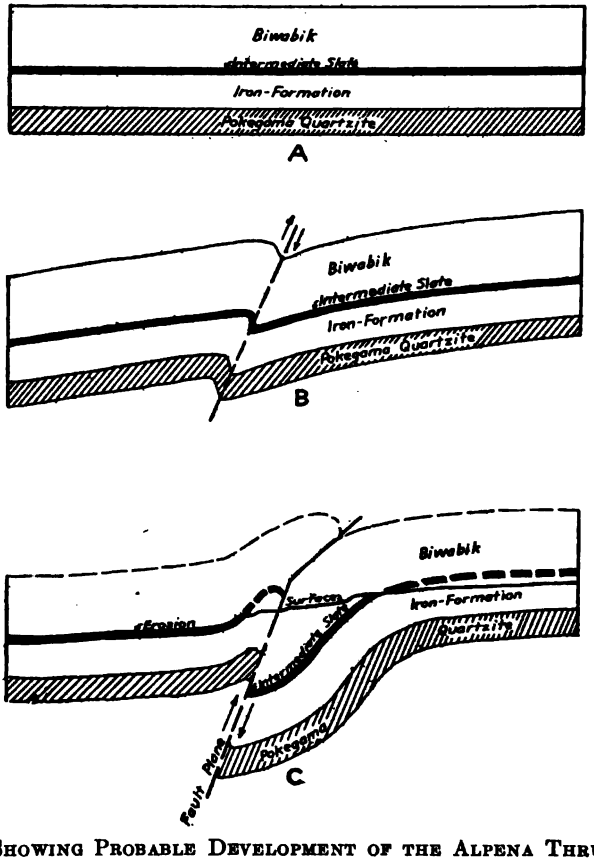

FIG. 13.—SHOWING PROBABLE DEVELOPMENT OF THE ALPENA THRUST-FAULT.

Thickness and Dip of the Iron Formation

The average thickness and dip of the whole iron-formation in different parts of the Range may be of interest. So far as they have been determined from the present very extensive explorations, they are shown in the accompanying table, tabulated by Ranges.

It will be noted that west of the Alpena fault the dips are quite low and uniform, while east of it they are higher and rather irregular. From this evidence and that of the Alpena-Dorr fault, it appears that the disturbance caused probably by the intrusion of the great mass of Duluth

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	Average Dip, Degrees	Thickness, Feet	
T. 56 N.—R. 24 W.	6	520	
T. 56 N.—R. 23 W.	3 to 8	475	(approx.)
T. 57 N.—R. 22 W.	5	615	
T. 58 N.—R. 21 W. T. 57 N.—R. 21 W.	5	590	
T. 58 N.—R. 20 W. T. 57 N.—R. 20 W.	4½	650	
T. 58 N.—R. 19 W.	3 to 7	660	
T. 58 N.—R. 18 W.	3	630	west of Alpena fault
T. 58 N.—R. 18 W. T. 58 N.—R. 17 W.	6 to 9	755	east of Alpena fault
T. 57 N.—R. 17 W.	10 to 15		
T. 58 N.—R. 16 W.	12	650	
T. 58 N.—R. 15 W.	25	530	(Secs. 5 and 6)

gabbro between Lake Superior and the eastern Mesabi range tilted the eastern part of the Range considerably; that the Alpena fault took up most of the crustal shortening in the Upper Huronian series due to the compression from the east; and that, because of the faulting, the sedimentary series west of it was relatively undisturbed.

The thickness is seen to be greatest in the vicinity of Eveleth and thinnest on the extreme eastern and western ends. The average of the figures given is about 620 ft.

Volumetric Shrinkage in Solid Taconite

Recent structural study has shown that the alteration which the original iron formation has suffered has produced a volumetric shrinkage even where the iron formation still retains its solid condition. However, in such solid taconite, hard layers of high-grade iron ore are interbanded with chert layers, the whole mass being firmly cemented together. Drill holes in fresh unaltered iron-formation alongside of holes in altered but still solid taconite show this fact to exist. Such a case is shown in comparing the depths of formation below the Lower Slaty Horizon in holes 1 and 2 with those in holes 4 and 5 in the orebody in Sec. 19, Fig. 9; also holes 1 and 2, Fig. 6.

This fact is a very important one to the mining engineer in the district in working out the structure of orebodies. The knowledge of it will aid him to establish more accurately the correct structure of certain ore layers. It explains the apparent lack of parallelism between different members of the iron-formation, which cross-sections often indicate unless this fact is known and applied.

Post-Algonkian Conglomerates on the Upper Huronian Series

In the *Engineering and Mining Journal*, July 17, 1915, the writer called attention to at least one conglomerate, and possibly two, on the top of the

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Biwabik iron-formation, older than the so-called Cretaceous conglomerate, beds and remnants of beds of which are found on most of the orebodies west of Eveleth. Since that time he has observed in two mines a conglomerate capping the orebody, containing large boulders which themselves were composed of the typical Cretaceous conglomerate. These conglomerates were overlain by layers of plastic shale and muds. Undoubtedly these are local lakebed or stream bed conglomerates and muds, intermediate in age between Cretaceous and Pleistocene.

Newly Discovered Fossil Remains in the Cretaceous (?) Shale

The conglomerate and shale found on top of many orebodies were correlated by the U. S. Geological Survey as Cretaceous from the fossil remains of bivalves and some teeth and vertebræ of the Mosasaur found in the shale. During the past year, in the Canisteo pit at Coleraine, one almost perfect fossil and part of others of Ammonites were found. A reproduction is shown in Fig. 14. As nearly as the writer and associates can correlate these specimens, they belong in the Jurassic and not in the Cretaceous. They show no such complicated sutures nor such ornamentation as is characteristic of the Cretaceous Ammonites. The Specimen *A*, Fig. 14, is about 15 in. in diameter and 3 in. thick. On Specimen *B*, which fits into the cast *C*, and *C* also, can be seen minute veins, but no evidence of complex sutures. It is possible, of course, that this Jurassic form lived over into the Cretaceous, but the numerous Saur remains the writer has seen could hardly have belonged to a reptile as large as the Mosasaur. They seem more fitted to a smaller creature, perhaps not over 10 ft. long, rather than to one 30 or 40 ft. long. May they not belong to a Jurassic saur, one less fully developed than the Cretaceous Mosasaur? Is it not possible, or even probable, that the conglomerates and shales on the Mesabi Range which have been called Cretaceous really are of Jurassic age and formerly were connected with the extensively developed Jurassic shales of Northwestern Minnesota? Northern Minnesota is such a well-developed peneplain that it is difficult to imagine a Jurassic ocean covering any part of it without reaching at least the foothills of the Giants or Mesabi Range.

Virginia Slate, Iron-Formation Contact

Because of possible value in connection with a revision of the correlation of the Huronian series in the Lake Superior district, the writer wishes to append to this paper the following record of observations of the relation of the Virginia slate to the underlying iron-formation, particularly because it is not in accord with statements as to this relation made in *Monograph* 52 of the *U. S Geological Survey*. The latter publication states that at the top of the iron-formation and the base of the Virginia

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FIG. 14.—FOSSILS FOUND AT COLERAINE.

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slate, there is a horizon perhaps a few hundred feet thick which is one of gradation, in which layers of iron-formation and slate alternate, and that "the layers of slate are found well down in the iron-bearing formation, and layers of the iron-bearing formation are found well up in the slate" (page 174, *Monograph* 52, *U. S. Geological Survey*).

An examination of cores from a great number of holes which penetrated through the Virginia slate into the underlying iron-formation has failed to substantiate this statement. Cores from two drill holes which penetrated between 500 and 600 ft. of Virginia slate and the entire underlying iron-formation to quartzite, and from scores of other holes, failed to show the presence of a single layer of greenalite in the true Virginia slate. Several thin layers of a carbonate rock (probably calcium carbonate) and a few crystals of iron carbonate were observed. In the iron-formation proper a very few bands of a carbonate rock were discovered. As shown on Fig. 5 of this paper, the upper horizon of the iron-formation is a slaty horizon in which layers of dark slate and greenalite alternate. At the top of this horizon and separating it from the true Virginia slate (which is a dense dark gray or black slate), is a layer several feet thick of calcium carbonate, amorphous or very finely crystallized. This layer is mentioned on page 171, *Monograph* 52, *U. S. Geological Survey*. Wherever drill holes have penetrated through the Virginia slate all along the Range, this carbonate has been found immediately beneath it. In a few holes, cores of which were examined very carefully, a small amount of conglomeratic material was found in the upper part of this carbonate layer.

Although the average thickness of the iron-formation in adjacent areas is quite uniform, as shown in the table of average dips and thicknesses, there are marked irregularities within short distances. Differences in total thickness of iron-formation of 20 to 50 ft. in drill holes ¼ mile apart are common. Two holes, 1¼ miles apart show a difference in total thickness of 121 ft.; two holes 2½ miles apart show a difference of 184 ft. Not enough close subdivision and correlation work has yet been done to determine whether such differences are due to initial deposition or erosion from the top of the iron-formation. There is so much interbedded fine conglomerate in the upper horizons of the iron-formation that we know definitely that this part of the formation at least was deposited in very shallow water. It is not at all improbable, therefore, that prior to the deposition of the Virginia slate, the iron-formation may have been raised above water, and its upper surface somewhat eroded. The variation in thickness of the Upper Slaty Horizon gives support to this idea. No marked unconformity between the two formations can be established, however. They are conformable stratigraphically, as far as now known, but the significant fact of the absolute lack of any known greenalite (so far as revealed by many years of observation of thousands of feet

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of drill cores by the director of explorations of the Oliver Iron Mining Co.) in the Virginia slate and its marked prevalence immediately beneath the Virginia slate seems to call for such a pronounced change in conditions of deposition as to demand some time interval between the two. The universal prevalence of the calcium carbonate layer with some conglomerate between the two gives further support to this idea of a time interval.

These facts, minor though they may be, are presented here in the hope that they may be of some value to the geologists who are engaged in the revision of the correlation of the Huronian Series of the Lake Superior District.

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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba

BY MAX ROESLER, PH. D.,* SANTIAGO DE CUBA, CUBA

(New York Meeting, February, 1917)

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* Geologist, Spanish American Iron Co. and Juragua Iron Co.

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I. INTRODUCTION

THE following article concerns the geological occurrence of the iron-ore deposits on the south coast of Cuba. The article is based on a detailed field study, made in the hope that some information would be gained which might be of value in the search for further orebodies, or in the economic development of the ore already found.

LOCATION

The iron-ore deposits of the Firmeza district lie near the coast, in the southeastern part of Cuba, in the Province of Oriente. The orebodies of this district form part of a belt of deposits that extends from Sigua, 25 miles east of Santiago, to Sevilla, 5 miles east of Santiago, and lies on the seaward slope of the Sierra Maestra range of mountains.

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This range roughly parallels the coast in the southern part of Oriente Province. Its crest is about 6 miles north of the mines near Firmeza. The town of Firmeza lies 9 miles east of Santiago, and about 2½ miles from the Caribbean Sea. The mines included in the Firmeza district are the Ocania Mine and a group of mines that extends from West Five Mine to the Concordia Mine. The elevation of the mines is from 400 to 1,000 ft. above sea level. Their exact location is shown on the map (Fig. 1) which is a copy of a map furnished by the Juragua Iron Co.

Immediately east of the Firmeza district lies the Daiquiri district of the Spanish-American Iron Co.

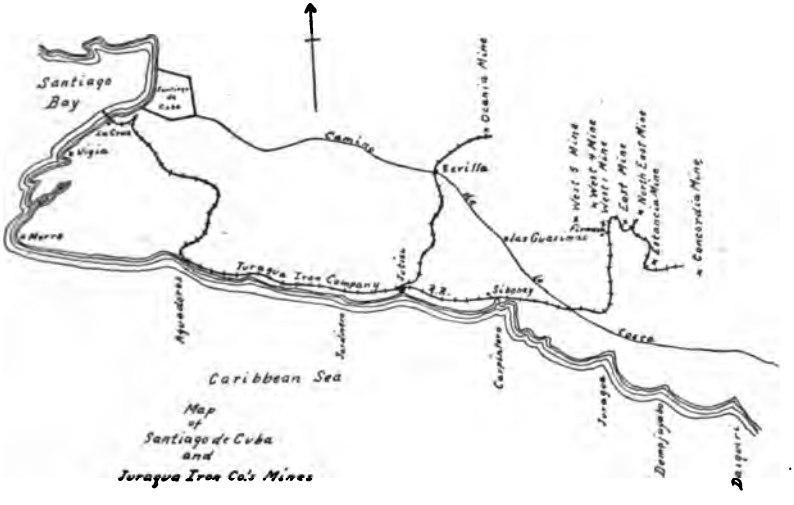

FIG. 1.

SCOPE OF WORK AND ACKNOWLEDGMENTS

Detailed examinations have been made by J. P. Kimball,[1] A. C. Spencer,[2] and J. F. Kemp,[3] and many others have contributed to our knowledge of the deposits. The writer has made use of all the available information and will endeavor to give full credit for it.

The present article has been written under the first award of the S. F. Emmons Memorial Fellowship, and the writer takes this occasion to thank the committee in charge of the fellowship for their assistance and criticism. Thanks are also due to the Juragua Iron Co. for permission to make the examination, for the use of their maps, and for their kindness in giving access to all the available data and facilities. The Spanish-American Iron Co. also made possible the visiting of neighboring

[1,2,3] See Bibliography at end of paper.

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properties and most grateful acknowledgment is hereby made to the officials and staff of both companies.

The Geological Faculty of Yale University, and Professor C. P. Berkey of Columbia University, have been freely consulted on different phases of the problem, and their advice has been of much service.

History and Mining

The Firmeza orebodies have been worked continuously since 1884, except for a short period during the Spanish-American war, and the total production of the mines, from 1884 to 1913 inclusive, has been 6,776,171 tons.[4] The mines of the Firmeza district belong to the Juragua Iron Co., which is now controlled by the Bethlehem Steel Co., though it was at one time also allied with the Pennsylvania Steel Co.

The mining of the ore and the stripping of the overburden is done by steam shovel. Where the orebody is small or intimately associated with waste, it is found advantageous to lease the workings to contractors and pay for the ore on the basis of tonnage and grade, delivered in mine cars.

The ore from the mines is put through the crusher at Firmeza, from which the major part is shipped direct to La Cruz, the company's shipping point on Santiago harbor. A small part of the output, high in sulphur, is roasted in the valley south of Firmeza, near Siboney, before it is shipped to La Cruz.

II. TOPOGRAPHY AND ITS INTERPRETATION

The topography of the Firmeza district falls naturally into three main divisions which have a lateral extent parallel to the sea coast and the main range of the Sierra Maestra,—that is, in an east-west direction. Beginning at the coast there is a range of terraced cliffs rising to uniform height of about 300 ft. The terraces all face the south, to the sea, and appear to be of wave-cut origin, excavated in flat-lying coral limestone. There is no beach, the sea beating directly against the limestone cliffs, except at the mouth of the Rio Carpintero at Siboney, where there is a small beach made up of coral fragments and an arkosic sand in which feldspar predominates.

Back of the terraces there is a fairly gentle landward slope to a silt-covered, flat, east-west valley. In this valley the streams from the mountains are generally lost in lagoons and swamps. Some of the larger ones, such as the Rio Carpintero, find their way to the Caribbean through comparatively narrow gorges cut in the cliffs that border the sea. These streams appear to be of an intermittently torrential type. In the dry season they show very small streams of water flowing through stream beds covered with boulders up to several tons in weight.

[4] D. B. Whitaker: *Engineering and Mining Journal*, vol. 97, p. 677 (1914).

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North of this valley rise the foothills of the Sierra Maestra—sharp, steep hills, covered as a rule with dense forest growth. They are connected with the main range of the mountains by ridges that have steep slopes and narrow crests. The connecting ridges are less heavily forested than the foothills but are covered with high grasses and an occasional grove of trees. The mountains themselves are thickly covered with pines and rise to an elevation of about 3,500 ft.

The limestone cliffs, with their sea-cut terraces, have been interpreted as evidence of periodic movements of the island, but it seems to the writer that the widespread occurrence of a terrace about both Jamaica and Cuba at the same elevation above sea level indicates a movement of the sea surface, rather than of the land.

In speaking of the Seboruco (the local name for the coastal limestone) R. T. Hill says:[5]

"Nowhere have I seen the elevated reef rock folded or otherwise disturbed except by the gently sloping coastward inclined elevation it has undergone.——The Seboruco as a whole represents a recent and uniform elevation of the whole periphery of the island——."

The following terrace elevations have been taken from the report of C. W. Hayes, T. W. Vaughan, and A. C. Spencer,[6] and arranged in tabular form by the writer:

Havana, Feet	Matanzas, Feet	Gibara, Feet	Baracoa, Feet	Manzanillo, Feet	Santiago, Feet
4–5	5–6	5–20	5–6	5–20	20
10–15	30	40	90	100	100
100	140	100	250	200	280
200	200	150–180	.....	.....	.....
.....	300	.....	.....	.....	.....

R. T. Hill[7] in the report on Jamaica has given less definite figures for the terrace elevations, and the following conclusions:

"In general the old reef rock of Jamaica consists of three distinct formations, occurring at three levels, 70, 25, and 10 ft. (or less) respectively. From the persistency of these three levels on the north, east and southwest end of the island, it is evident that their present position above the water is due to continuous epeirogenic elevation after the present outlines of the island had been chiefly defined."

For Hayti and Porto Rico no exact data are available to the writer.

[5] R. T. Hill: Notes on the Tertiary and Later History of the Island of Cuba. *American Journal of Science*, Ser. 3, vol. 48, p. 203 (1894).

[6] *Op. cit.* (in Bibliography), pp. 18, 19.

[7] R. T. Hill: Geology of Jamaica. *Bulletin* of the Museum of Comparative Zoology at Harvard, vol. 34, pp. 92–100 (1899).

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The data quoted show that on the north, northeast, and southeast coast of Cuba, and on the north, east, and southwest coast of Jamaica, there is a marine terrace from 5 to 20 ft. in elevation. It seems to the writer that this evidence might suggest that, at least in part, the differential movements of land and sea in Cuba have been due to movement of sea level rather than of land surface. The slight variations in elevation would then be explained by unequal erosion of the gently sloping surface of marine planation. The slight seaward slope of the coast limestone, in the neighborhood of Santiago, is no greater than is to be expected of a near shore surface of marine planation.

There is further evidence of emergence of the island, in respect to sea level, in the tuff-limestone found at an elevation of about 1,400 ft., and the steep-sided, sharp, mountain topography points to fairly recent rejuvenation.

The silt-covered, east-west valley, in which the streams are aggrading, suggests that the last movement of land with respect to sea level has been a downward one, and this is in accord with the evidence of the harbors of Santiago and Guantanamo, which have been interpreted as bays formed by drowning.[8]

The topography seems to be almost entirely independent of the lithology. An exception to this is the coarse crystalline marble that forms a capping to many of the foothills. Apparently the massive marble offers greater resistance to tropical weathering than do the fine-grained, diabasic or dioritic rocks which marmorized it, for the arroyos are cut into the latter. The occurrence of marble as a capping is too frequent to be entirely fortuitous.

III. PETROLOGY

Sedimentary Rocks

Description

The sedimentary rocks are represented in the Firmeza district entirely by limestones. At the coast, and rising in three sea-cut terraces to an altitude of about 350 ft., are recent limestones. According to the report of Hayes, Vaughan, and Spencer,[9] this rock is

> "replete with the remains of numerous species of corals which are all, so far as examined, at present living in the surrounding Antillean seas."

There seems to be one rather striking point of difference between the higher landward terraces and the lower terrace bordering the Caribbean. This is, that while all three terraces contain coral remains, the landward

[8] Hayes, Vaughan, and Spencer, *Op. cit.*, p. 17.
[9] *Op. cit.* pp. 23–24.

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terraces are almost entirely massive, while the seaward terrace is made up of more loosely cemented coral remains. On the land, where the limestone comes into contact with the underlying igneous rock, there are boulders of rock and of iron ore cemented by the lime, showing that erosion of the orebodies had begun before these coral rocks were formed.

In the foothills of the Sierra Maestra, which vary in altitude from 600 to 1,300 ft., there occur, usually as a capping on the hilltops, bodies of massive limestone now largely marmorized. Bedding is almost entirely lost, but where thinner masses have been involved in the volcanic rocks it is possible to discern a pitch toward the southeast at an angle of about 30°. The freshest pieces of this limestone, taken from boulders on the north side of the hill north of West Five Mine, show it to be a blue, dense, fine-grained limestone. The microscope shows no evidence

FIG. 2.—TUFF-LIMESTONE FROM RIDGE CONNECTING FOOT HILLS WITH MAIN RANGE OF THE SIERRA MAESTRA. Limestone—groundmass. Plagioclase fragments—white. Volcanic rock fragments—dark. × 18.

of organic remains in the thin section, but only a granular aggregate of calcite.

On one of the ridges connecting the foothills with the main range, at an elevation of about 1,400 ft., an unaltered limestone of unusual type was found. This rock has the deeply pitted surface typical of the weathered outcrops of all the limestone in this region. The color of the rock on a fresh fracture surface is blue-gray. Closer examination shows minute areas with a vitreous luster embedded in a granular calcite matrix, giving to the rock the appearance of a porphyry. A thin section of the rock shows that the crystals are fragments of plagioclase, of about andesine-labradorite composition. There are also angular fragments of a diabasic rock (Fig. 2). The fragments are all about 0.5 mm. in diameter, and their presence in the limestone shows that the sediment was

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formed at a time of volcanic activity on the adjacent land. The perfect freshness of the feldspars is probably due to the fact that the sea water was already saturated with calcium carbonate, and so was not chemically active in so far as the calcic feldspars were concerned. The ferromagnesian minerals of the diabasic fragments are entirely altered. This tuff-limestone seems to be very similar to that described by C. P. Berkey[10] among the "probably" pre-Tertiary rocks of Porto Rico.

Age

The nature of this rock shows that at least part of the volcanic activity was contemporaneous with the sedimentation. As the andesitic rocks accompanying this tuff-limestone are believed to represent the initial stages of the igneous cycle, the determination of the age of the sediment must also fix the age of the igneous rocks.

The age of these limestones and the associated volcanics has been a rather vexed question. J. F. Kemp speaks of an article by H. Wedding,[11] in which he places them in the Jurassic, in the horizon of Quenstedt's Beta, of the upper White Jura. This determination was made upon material furnished by G. W. Goetz. The only other definite information on this subject of age is that given by J. T. Singewald, Jr., and B. L. Miller.[12] They found a fossiliferous limestone in the Daiquiri district, and submitted some of the fossils to T. W. Vaughan, who correlated one of the species with the Cretaceous of Jamaica and determined the age of the limestone as "Mesozoic, probably Cretaceous."

W. Lindgren[13] says:

> "The idea of the geologists who have done the most work in this section seems to be that the lavas and tuffs and associated limestones are of Eocene age——."

R. T. Hill,[14] speaking of the volcanic clastics of the Jamaican Blue Mountain Series of Cretaceous age, says:

> "In Cuba these clastic rocks constitute the high divides of the Oriente——."

Experience during historic time shows that it would not be justifiable to infer volcanic activity in one of the islands of the Antilles on the basis of such activity in another. In the present case there is, however, the evidence of Cretaceous sedimentation in Jamaica determined by R. T. Hill, the direct correlation of the Daiquiri specimen with the Jamaican

[10] C. P. Berkey: Geological Reconnaissance of Porto Rico. *Annals of the New York Academy of Sciences*, vol. 26, p. 20 (1915).

[11] H. Wedding: Die Eisenerze der Insel Cuba. *Stahl und Eisen*, vol. 12, No. 12, pp. 545–550 (June 15, 1892).

[12] J. T. Singewald, Jr., and B. L. Miller: The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, pp. 67–74 (1916).

[13] W. Lindgren and Clyde P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vol. 53, p. 41 (1916).

[14] R. T. Hill: The Geology and Physical Geography of Jamaica. *Bulletin* of the Museum of Comparative Zoology at Harvard, vol. 34, p. 170 (1899).

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fauna by T. W. Vaughan, and R. T. Hill's reference to the clastic rocks of Oriente. E. T. Hodge has informed the writer that he found Comanche fossils among the pre-Tertiary.

Igneous Rocks

The igneous rocks of the Firmeza district form a natural series of differentiation products from a basic magma. They range from a fine-

Fig. 3.—Diabase Porphyry with Included Fragment of Earlier Volcanic Rock. West Four Mine. × 18.

Fig. 4.—Diabase Porphyry with Epidote-Filled Amygdule. From Railroad Cut Opposite West One Mine. × 18.

grained diabase to a highly quartzose aplite. J. F. Kemp[15] has given a discussion of the nomenclature used in former articles on the district.

[15] The Geology of the Iron-Ore Deposits in and Near Daiquiri, Cuba. *Trans.*, vol. 53, pp. 3–38 (1916).

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For purposes of mapping, the writer divides the igneous series into four groups:

1. The diabasic rocks.
2. The dioritic group.
3. The granitic group.
4. The later dike rocks.

FIG. 5.—DIORITE FROM LEVEL 1 OF WEST FIVE MINE. CROSSED NICOLS. × 18.

FIG. 6.—QUARTZ DIORITE FROM EAST MINE. THIS IS THE BASIC MEMBER OF THE GRANITIC SERIES. CROSSED NICOLS. × 18.

Diabasic Rocks

The diabasic rocks are fine-grained, as a rule, and porphyritic. They are the representatives of the original magma. Megascopically they are dense, dark rocks. Under the microscope they show as felty aggregates of plagioclase laths with interstitial ferromagnesian minerals. They are

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usually fragmental (Fig. 3) and frequently show epidote-filled amygdules (Fig. 4).

These diabasic rocks show a bedded structure in places, with intercalated limestones and are undoubtedly in part extrusives.

FIG. 7.—GRANITE SHOWING MICROGRAPHIC QUARTZ FELDSPAR INTERGROWTH ABOUT PLAGIOCLASE NUCLEUS. FROM THE JURAGUA VALLEY. CROSSED NICOLS. ×18.

Dioritic Rocks

The wall rock of many of the orebodies is a gray, fine-grained, and

FIG. 8.—SPECIMEN B FROM DIKE IN WEST SIDE OF WEST FIVE MINE. ELEVATION 520 FT. NOTE CORROSION OF PLAGIOCLASE AND DOMINANCE OF QUARTZ. CROSSED NICOLS. × 18.

even-textured rock. Plagioclase, varying from labradorite to bytownite, is the most abundant mineral, with much hornblende between the feldspars. On the basis of the feldspars it is a gabbro, but in the older

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classification the hornblende makes of the rock a diorite. Since that terminology has been adopted by earlier writers on the district it is accepted here. Fig. 5 shows a typical diorite.

The diorites and diabases are frequently found merging into each other without any visible contact. Both are the hosts of the orebodies.

Granitic Rocks

J. F. Kemp[16] has used the term granite for the quartz-bearing diorites and the true granites "to avoid all confusion of this rock with the diorites which are associated with the ore." That usage is adopted in this paper, and the scope of the term granitic rocks is enlarged to include the aplites and quartz porphyries. This inclusive usage seems justified by the fact that the quartz-bearing rocks form a group later than the ore-bearing rocks, though pre-mineralization.

All the rocks of this group are hornblende-bearing except the most acid aplites, and even the most acid show very little potash feldspar. The basic members of the series are gray, coarse-grained, feldspathic rocks (Fig. 6) while the acid members are made up of striking feldspar quartz intergrowths (Fig. 7), or else are fine-grained quartz aplites (Fig. 8). The extremely rapid variation in chemical composition is shown by three microscopic analyses of rocks from the same dike at different elevations and ascending order.

	Feldspar	Quartz	Ferromag.	Magt.
A.....	61.8	28.0	5.5	4.7
B.....	54.0	41.6	3.4	1.0
C.....	52.7	45.2	1.1	1.1

Later Dike Rocks

The entire mineralized area is cut by basic dikes that vary from basalt to andesite. They are post ore, but usually carry pyrite. A detailed description would serve no definite purpose in this connection.

IV. AREAL GEOLOGY

Rock Types Found in the District

The study of the petrographic features shows that there are several types of igneous rocks and two limestone formations in the Firmeza district. The igneous rocks form a continuous series from diabasic extrusive and intrusive rocks, through diorite and granite, to highly acid

[16] *Op. cit.*, p. 12.

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aplites. This series is cut by dike rocks of various types, which range in composition from basalt to andesite. The sedimentary rocks fall naturally into two divisions;—earlier marmorized limestone, probably Cretaceous in age, and now exposed only in scattered outcrops, and younger recent coral limestones deposited as a fringe on eroded igneous rocks.

Distribution of the Various Rock Types

Surface Distribution

The two appended maps show the surface distribution of the rocks and orebodies. Owing to the lack of topographic maps, and the difficulties attending work in an area covered by tropical verdure, they cannot lay claim to great accuracy of detail. Future work will undoubtedly shift the contacts in many places and add a number of dikes. But enough work was done, and enough outcrops plotted to justify the making of the maps and the belief that they show the general relationship of the formations. Acknowledgment is made to Dean Corsa, whose unpublished map in the possession of the Juragua Iron Co. was in part used.

The different formations lie, as shown by the map, in belts of irregular width, roughly parallel to the coast line.

Nearest the sea is the belt of coral limestone. This belt is about 3,000 ft. wide in the area mapped. To the west, where the railroad runs to the Ocania Mine, the fringing limestone extends inland nearly 2 miles. Apparently the width of the belt is largely determined by the slope of the pre-deposition erosion surface, the wider belt lying ont he more gently sloping surface.

North of the coral limestone belt, the main mass of granitic rocks is exposed. This forms a belt from 6,000 to 8,000 ft. wide. Most of the granite area is covered by an alluvial deposit, through which rise knobs of the igneous rock. These knobs are all granite, usually with inclusions of dioritic material.

North of the granitic area lies the diorite. This forms a very irregular belt on the lower part of the foothills of the Sierra Maestra. From the Demajayabo River south of the Concordia Mine, to south of Loma Alta, the width of diorite is from 1,200 to 3,000 ft. At West Five Mine the diorite area runs back into the hills north of the region mapped. In the Juragua valley the diorite is exposed about 1 mile north of Firmeza.

The diabasic rocks lie higher in the hills than the diorite and in general their exposures on the surface are north of the diorite. No attempt has been made to show on the map the contact between the extrusive and intrusive facies of the diabases. This contact is exceedingly indefinite and rarely distinguishable.

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Bodies of granitic rock, that differ from the main granite mass, cut through the diorite and the diabases in the region mapped. These bodies are apophyses of the main granite massif and form dikes, sills, or irregular bosses. They lie in a roughly east-west belt in the foothills, and are of great importance because the orebodies are scattered irregularly as a fringe to these apophyses. This can be seen clearly both on the map of the district and on the larger scale map of the mines near Firmeza.

The older limestones lie in the diorite and diabase area, rarely in contact with the granite. They have been metamorphosed to a dense marble in the area mapped, and are preserved as irregular masses capping the foothills, or as beds interbedded with the extrusive rocks. The outcrops vary in size from small blocks to areas that are as much as 1,200 ft. in diameter. An even larger area at the Ocania Mine does not show on the map.

All the formations are cut by the later basic dikes. Except in the mines these dikes have not been mapped. They fall into two systems: one almost vertical, striking northerly, or slightly east of north; the other an almost horizontal system, that might equally well be classed as a system of sills. Petrographically there is no difference between the two systems.

Vertical Distribution

The vertical arrangement of the different rock types is even more marked than their linear distribution, as it shows on the surface maps.

The coastal limestones rise in terraces to an elevation of about 300 ft., where they terminate in a flat top.

The granite massif has an upper surface which rises gradually from an elevation of about 350 ft. at the eastern end of the district, to about 700 ft. at the Ocania Mine. The granitic apophyses, as now exposed, are rarely more than 150 ft. above the top of the granite massif, and their downward extension where exposed by erosion can be seen to merge into the granite.

The diorite and the diabasic rocks must be considered together, because, while the lower limit of the diorite is determined by the top of the granite intrusion, the upper limit is, as a rule, indeterminate, and the diorite merges into the diabases. The two rocks were mapped separately in the field and a contact drawn, but it is generally arbitrary. More rarely the contact mapped represents an observed intrusive contact. As established, the contact between diorite and diabase rises from 500 ft. elevation in the eastern to 800 ft. at the Ocania end of the area. Fragmental igneous rocks are found at 700 ft. in the East Mine, and at 1,000 ft. elevation at Ocania. No continuous contact between intrusive and extrusive diabase was mapped.

The older limestone masses have not been found in this district, at

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a lower altitude than 400 ft. The highest body of limestone examined was the tuff-limestone found at 1,400 ft.

The lack of contour maps makes it difficult to draw accurate sections. But the elevations can be tabulated as follows:

West End of District (Ocania Mine), Feet		East End of District, Feet
1,000	Fragmental igneous rocks	700
800	Diorite-diabase contact	500
700	Top of granite massif	350
300	Top of coastal limestone	300

This table shows the distinct vertical distribution of the igneous rock types, and a well-defined pitch of their surfaces of contact.

Causes of Present Areal and Vertical Distribution of the Rock Types

Faulting.—The linear distribution of the rock types, as shown on surface maps, suggests at once a direction of major faulting. But except for two minor faults in the East Mine, the one northeast-southwest, the other northwest-southeast in strike, and a somewhat larger east-west fault in West Five Mine, no displacements worthy the name of fault were found in the area. The nearest approach to a fault system is shown by the later basic dikes. These, with the one vertical north-south trend and a horizontal system, appear to have filled a definite fissure system, but there is no evidence of any appreciable movement along most of them. In West Five Mine an aplitic dike is offset 6 in. on the opposite sides of a 2-ft. wide basic dike. If the dikes do represent an older fissure system, it is a system due to contraction during cooling of the igneous rock, rather than to faulting. The Firmeza district must be added to the long list of those in which ore deposits are connected with fissures of minor or no displacement.

It is the writer's opinion that the present areal and vertical distribution is due to three different causes—magmatic differentiation, tilting, and erosion. Each of three must be considered in an attempt to solve the problem.

Magmatic Differentiation.—To show that the vertical distribution of the rocks is due to magmatic differentiation it will be necessary to establish the comagmatic origin of the igneous rocks and their age relationship.

The detailed petrographic study has shown that there are present in the district igneous rocks showing all the stages of a gradual transition from diabase porphyry to highly acid aplites. The merging of the types can also be found in the field.

Northeast of Estancia hill the transition of a granite-porphyry sill into the main granite and quartz-diorite massif can be followed. There is no contact between the two types. The granite massif itself contains numerous inclusions of more basic material. These inclusions are an-

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gular, subangular or rounded. J. F. Kemp[17] fou
inclusions in the Daiquiri district and his interpret

"Apparently the inclusions represent some older
With this interpretation the writer does not feel in
inclusions certainly represent early differentiates,

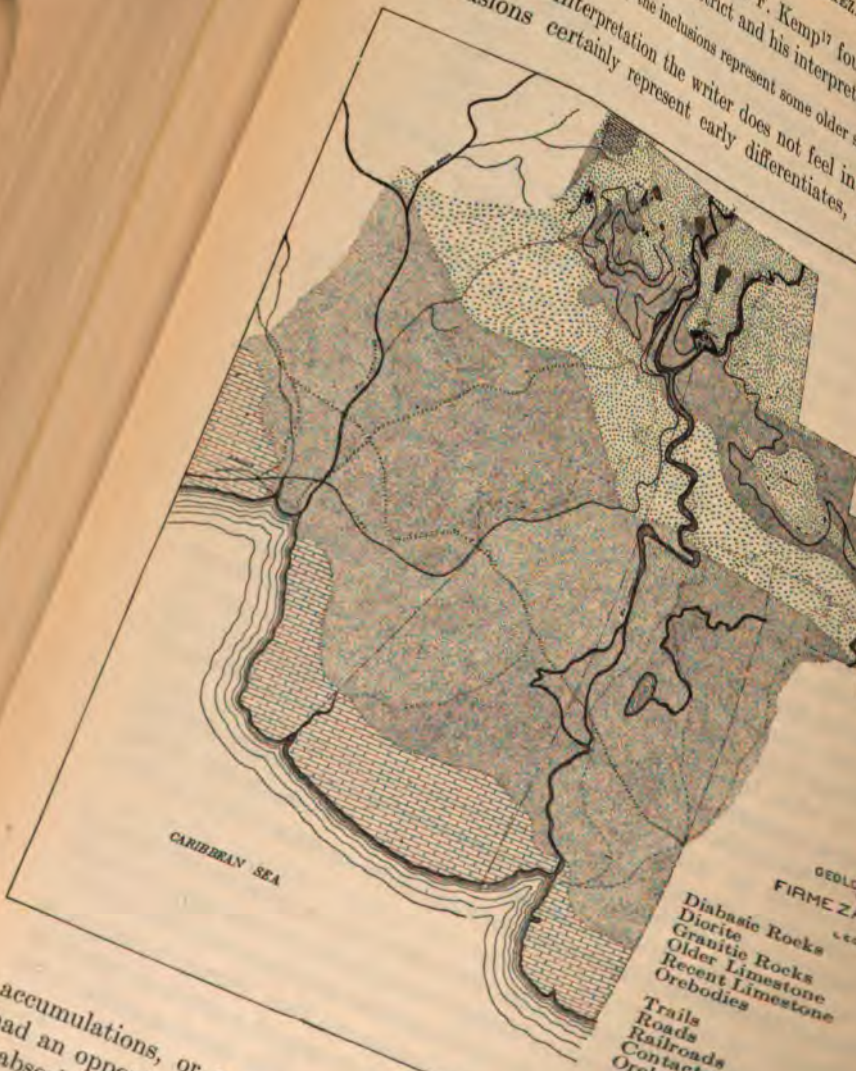

FIG. 9.

accumulations, or possibly segregations in the still liquid
had an opportunity to crystallize; but either way th
reabsorbed portions of the magma itself, and

[17] Op

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After initiation of crystallization the character of the magma changed until it had power to reabsorb part of the already crystallized material. The result of the same action was observed on a small scale in a microphotograph of a specimen from the lower part of the aplite dike in West Five (not reproduced in this paper). It shows a zonal plagioclase, calcic in the central part, sodic at the edges, and much corroded. Obviously the plagioclase must have formed before the liquid was of such a composition that it could corrode it. Similarly it seems possible that, in a slow-cooling magma, crystals form and gather in clusters, and the interstitial liquid, changing in composition, could reabsorb them more or less completely.

Beside the evidence of the endogenous inclusions, the transition from quartz-bearing to quartz-free diorite can be observed in the upper levels of West Five Mine, and that from diorite to diabase in West Four Mine. That the extrusive diabases and intrusive diabases are of the same magmatic origin can be seen from the complete similarity of the two rocks both megascopically and microscopically. The only difference is the greater fineness of the ground mass in the volcanics.

Other features that seem to point toward comagmatic origin of the igneous rocks are the almost total absence of orthoclase in the entire series, and the predominance of hornblende as a ferromagnesian constituent. The latter is considered an indication of the presence of abundant crystallizers which would aid in holding the magma fluid at comparatively low temperature.

The age relationship of the igneous rocks in a country lacking in sediments that are chronologic guides must be determined by intrusive contacts. The intrusive nature of the granitic apophyses in the diabases and diorites can be established in almost any of the mines. That the main granite massif is later than the diorite is well shown in La Posa brook just north of its junction with the Rio Carpintero, where dikes of massive, even-grained granite cut the diorite. Diorite cutting diabase is seen at the contact of the two rocks in the Juragua valley north of the mines. Diabase containing fragments of the volcanic diabases is found all through the district.

The order of formation is, then, the normal one from the basic to the acid end of the series; *i.e.*:

1. Diabasic extrusive.
2. Diabasic intrusive.
3. Diorite.
4. Granite.
5. Aplite.

The origin from a common parent magma and the age relationship established, it can be shown how this would account for the observed

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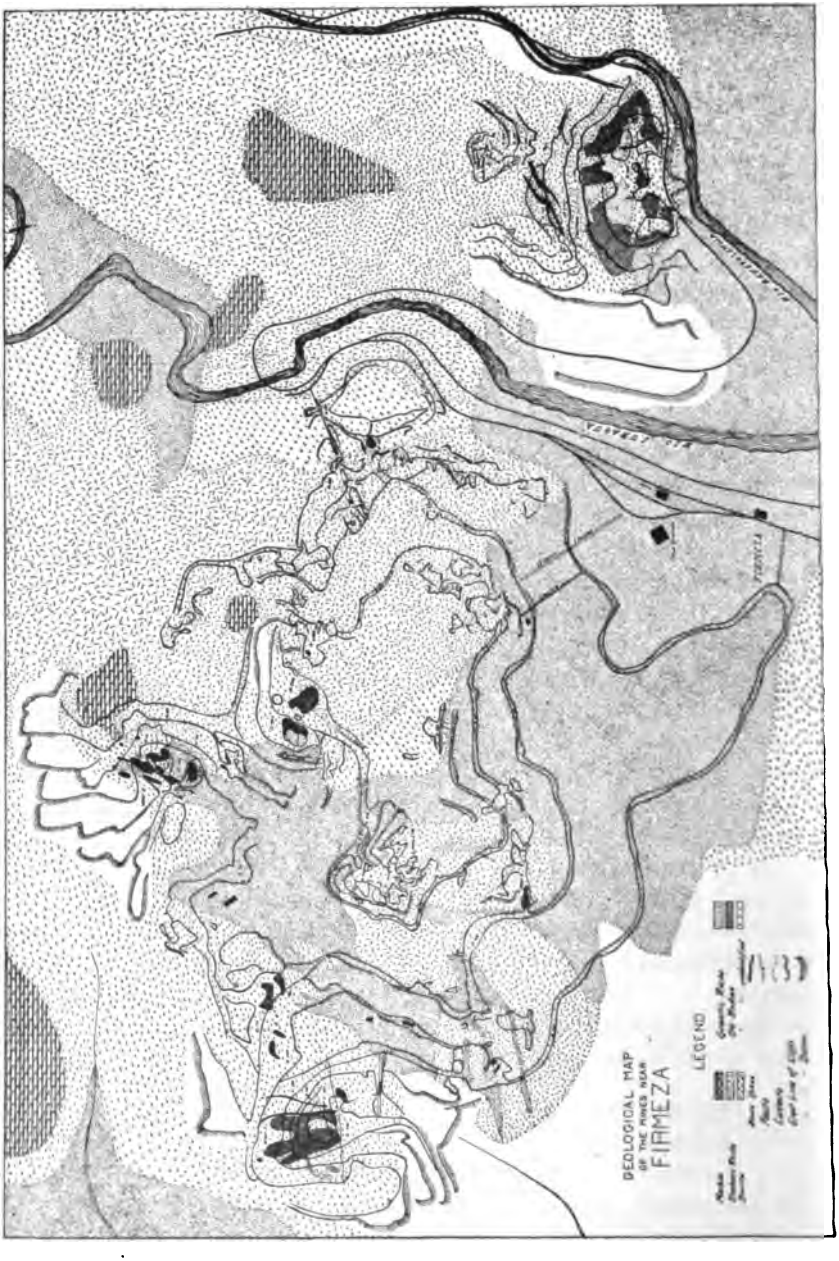

FIG. 10.

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ical distribution of the igneous rock types. The series from diabase ranite agrees so completely with the series described by N. L. Bowen[18] ne normal result of fractional crystallization in a basaltic magma, this mode of differentiation seems to be the probable one in this

The diabase extrusives mark the initiation of igneous activity, and the ase intrusive and the diorite mark early chilled phases of a magma ming more and more acid. The granitic massif is believed to be batholythic invasion of the more completely differentiated acid liquid, the granitic apophyses part of the final differentiate from the granite. will be discussed more fully with the genesis of the ore deposits.

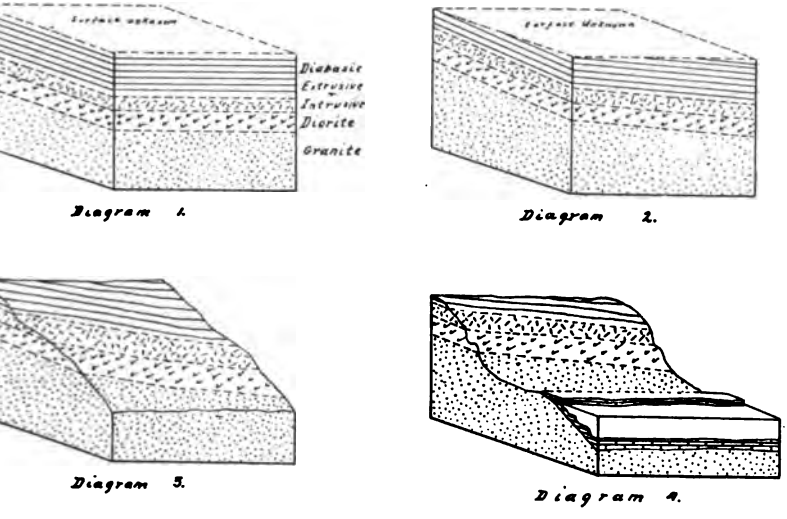

FIG. 11.—GENESIS OF PRESENT ROCK TYPES.

position of the granite at a definite horizon is thought to be due to act that the cooled shell of the magma had developed cooling cracks at depth, and as a result the crystallizers were able to escape from the te, which could then no longer remain liquid.

Thus differentiation may account for the horizontal arrangement of ock types observed in the field.

ilting.—However, the arrangement is not entirely horizontal. e is a distinct slant toward the southeast of all the surfaces of con- The extrusives and the limestones, where bedding can be de- ined, all show a pitch to the southeast. Just when the tilting took is difficult to establish, but since it does not affect the coastal lime- it may be placed as previous to its deposition.

N. L. Bowen: The Later Stages of the Evolution of the Igneous Rocks. *Supple- o Journal of Geology*, Vol. 23 (November–December, 1915).

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Whether tilting accompanied or followed the igneous cycle and the ore deposition cannot be determined. As the orebodies in the eastern end of the district are lower than those at the western end, it seems more probable that the tilting took place later than the period of ore deposition. The evidence is inconclusive and indefinite, but it is unimportant as the time of the tilting cannot affect any of the major hypotheses.

Erosion.—Since the time of ore deposition erosion has produced a surface that slopes down in general from the north to the south. In this way it has cut across the basic rocks and into the granitic massif.

Fig. 11 shows a diagrammatic representation of the way the present arrangement of the rock types is believed to have been brought about. The diagrams are not to scale, and are intended merely as an aid to visualizing the conditions.

Diagram 1. Shows conditions after differentiation and before tilting or erosion.
Diagram 2. Shows conditions after tilting and prior to erosion.
Diagram 3. Shows condition after partial erosion but prior to deposition of coastal limestones.
Diagram 4. Shows conditions as they are at present.

V. GENERAL DESCRIPTION OF ORE DEPOSITS

Nature of Ore

The ore mined in the Firmeza district is a mixture, in varying proportions, of magnetite and hematite. The ore is remarkably pure and contains little foreign matter. J. P. Kimball[19] gives the following figures:

	Per Cent.
Moisture (in part hygroscopic)	0.24 – 0.81
Silica and insoluble	5.00 –10.50
Phosphorus	0.009– 0.065
Sulphur	0.045– 0.248
Iron	61.00 –68.50

Although present mining methods make it possible to ship ores somewhat lower in iron content, and greater depth in mining has exposed ores somewhat higher in sulphur, the figures Kimball gave in 1884 are substantially correct for the ores now being mined. The low phosphorus and the absence of appreciable amounts of titanium make the ore a very valuable one for the manufacture of high-grade steel.

Shape and Size of Orebodies

In shape the deposits are extremely irregular. Whatever their size they show one common feature throughout the area—a far greater extension in two dimensions than in the third dimension. They resemble a

[19] J. P. Kimball: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Science*, Ser. 3, vol. 28, p. 426 (1884).

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ries of scattered lenses of irregular outline, that lie in every conceivable sition. Fig. 12 shows a series of horizontal projections of the ore-dies and a few vertical sections. The projections are drawn to scale, d the sections are sketched from exposures in the walls of the mines. ne outlines represent the boundary of the ore of commercial grade. ney do not represent the edge of the mineralization. There is gener-y a transition from ore to rock, not a definite contact between them. nis subject will be more fully discussed under the mineralogy of the ore posits.

The size of the ore deposits varies from pockets containing a few ns to lenses whose larger diameters are measured in hundreds of feet d whose thickness is from 10 to 50 ft.

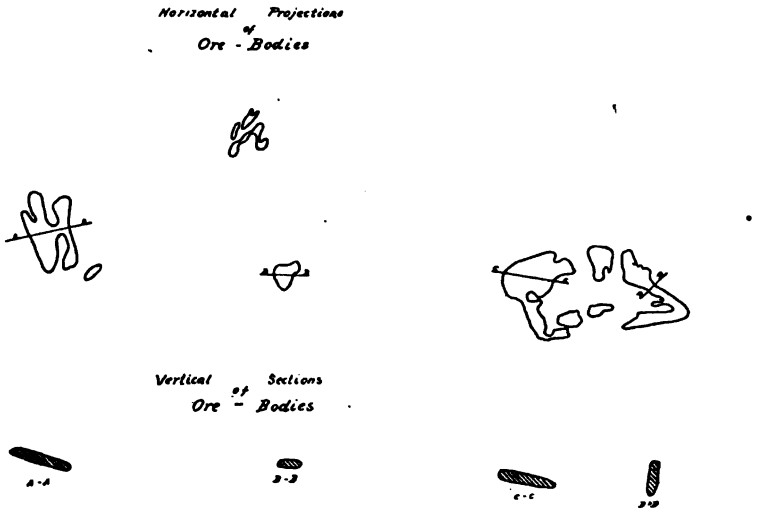

FIG. 12.

GEOLOGIC POSITION

With the possible exception of the Ocania and Chicharron Mines, the e deposits lie within 200 ft. of the granitic apophyses. These excep-ons are probably more apparent than real and may indicate that the lite has not yet been exposed, not that it is absent. In most cases the es are directly in contact with aplitic or pegmatitic rock.

The relation of aplite to ore is exceedingly intimate. In most of the ines the ore lies against a floor or wall formed either of aplite or of peg-atitic granite with much micrographic quartz-plagioclase intergrowth. requent aplitic dikes traverse the ore, and form prominent features cause of their light colors, in strong contrast to the black ore and the een igneous rocks. In spite of their apparent position as dikes, they e *not* to be considered as later than the mineralization. This will be own in the discussion of the genesis of the ores.

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The host rock of the ore is either one of the basic igneous rocks, or limestone. By far the greater part of the ore is in diorite or intrusive diabase; some is found in the diabasic extrusive rocks, and only a minor amount in the limestones. In at least three of the mines the ore deposits are very close to the overlying limestone masses, but there has been no localization in the limestone, and the larger masses are separated from the limestone by basic igneous rock.

Mineralogy

The ore minerals are magnetite and specularite, with some amorphous, more or less hydrated, hematite in the upper parts of the ore deposits. The common gangue minerals are quartz, wollastonite, epidote, and lime-iron garnet. The epidote comes in two clearly distinct forms. The one is a well-developed, crystalline form, transparent and pleochroic from green to yellow. The other is a fibrous and scaly form, lacking in well-defined terminations. The well-crystallized form is the one that

Fig. 13.—Pyrite Crystal from West One Mine.

is associated with the ores, and other lime silicates. In some of the mines calcite is intimately intergrown with the ore. Apatite, titanite, and a mineral determined with some doubt as scapolite, were also found in rare cases. Secondary calcite, introduced by recent weathering, is not uncommon. Pyrite and chalcopyrite are found in most of the mines. They are almost always introduced and rarely if ever original constituents of the ore deposits. The pyrite occurs in well-developed crystals. One of these from the West One Mine is shown in Fig. 13. Associated with them is chlorite and some sericite.

Distribution

The distribution of the minerals appears to be haphazard and extremely irregular, but natural phenomena are not haphazard, they are logical and sequential. Detailed study has convinced the writer that the minerals in these deposits are distributed according to a definite order. The most clearly localized material is the massive, fine-grained

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intergrowth of magnetite and specularite. This massive ore lies in, and close to, fissures, or else directly next to aplitic or pegmatitic granite. In the minute intergranular spaces of this ore occur wollastonite and some quartz. As the ore becomes less massive toward the margin of the deposit the proportion of specularite to magnetite increases, wollastonite becomes more abundant, as does quartz, and epidote and garnet appear. Still further out from the massive ore, garnet predominates, with some epidote, little or no wollastonite, very little quartz, and some more coarsely crystallized specularite and magnetite. In some cases this zone is composed entirely of massive garnet with very few impurities. Beyond the garnet is an area in which epidote and quartz are prevalent. Mineralization of this type is the most widely diffused, and merges gradually into partially chloritized and epidotized country rock. The above description holds only for the typical deposits in igneous rocks. The complete sequence is rarely well-exposed. It can best be observed in the East Mine and in West Five Mine.

Those deposits which are obviously in limestone show two kinds of mineral distribution. One is shown in the Chicharron Mine. It is a dike-like mass of magnetite and specularite, with interstitial wollastonite. The contact with the marble is sharp, and no contact minerals are found. The other form of orebody in limestone is a central mass of almost pure specularite, in rosette-like clusters, or granular masses, surrounded by an intimate intergrowth of quartz well-crystallized calcite, garnet and epidote. This form of orebody is best exposed in the North Mine. The transition to unaltered marble is not exposed there. In the Ocania Mine this transition is exposed and shows a sharp change from garnet rock to unaltered marble.

Apatite is so rare that it is a curiosity in the district. Fig. 14 shows a microphotograph of the one section in which it was found in appreciable amount. It appears in the usual hexagonal basal, and elongated prismatic sections. Later than the apatite in time of crystallization are magnetite and the minute garnet-epidote intergrowth. The scarcity of apatite and so of phosphorus in the ore is of the greatest economic importance. It is also an indication that phosphorus had little or no share in the mineralization.

The secondary calcite, the kaolin, and the amorphous and hydrated hematite, are found either very close to the surface or in channels to which meteoric waters have obviously had access. Pyrite is common in all the mines and lies close to the surface in an unaltered, well-crystallized condition.

Interpretation of the Mineralogy

The first inference to be drawn from the mineralogy of the ore deposits is that they were formed at a high temperature. The temperature of

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mineral formations is not yet so well known that accurate conclusions can be drawn from mineralogy as to the exact temperature at which ore

FIG. 14.—SPECIMEN FROM WEST FIVE MINE. × 18. Magnetite—black. Epidote and garnet—gray. Apatite—clear white.

deposits were formed, but the occurrence of lime-iron garnet and of wollastonite indicates high temperatures. The association with quartzose pegmatitic dikes, which, as will appear later, are believed to be con-

FIG. 15.—DIABASE PORPHYRY, SHOWING INTRODUCED QUARTZ AND EPIDOTIZATION. FROM EAST MINE. × 18. Quartz—white. Epidote—dark.

temporaneous with the ore formation, places the temperature at from 575° to 800°C.[20]

[20] F. E. Wright and E. S. Larsen: Quartz as a Geologic Thermometer. *American Journal of Science*, Ser. 4, vol. 27, p. 421 (1909).

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The second inference drawn from the mineralogy is that the iron, lime, quartz are either entirely, or in part, introduced material. The e quantity and great concentration of the iron oxides makes their ation from the rock in which they are found highly improbable, it can be assumed that they represent, almost entirely, introduced erial.

The lime silicates, epidote, wollastonite, and garnet are so abundant an addition of lime seems extremely probable. Because the rock, entirely replaced by ore, was a calcic igneous rock, it is impossible to rmine whether the lime, which was added to form garnet zones, came n the magma or from the replaced rock. Since, in places like ncia Hill, the garnet zones are out of all proportion to the size of the odies, an addition of lime from the magma seems probable. The or part of the lime is deposited in a zone beyond the iron. In this ion the lime silicates always show a later crystallization than the iron es. This makes it appear as if the mineralizing agent deposited most he iron oxides while still able to retain much lime in solution. Further ence of the addition of lime from the magma is afforded by the apa- intergrown with the magnetite, in the specimen from West Five e described above (Fig. 14). The epidotization and silicification in outermost zone seems to be largely recrystallization, and represents a rothermal alteration, more or less *in situ*, rather than an addition of erial. The chloritization is also an effect of this hydrothermal amorphism. Six samples of diabase showing more or less epidote were yzed for lime. They were selected at various distances from a lime- e inclusion, in order to show a supposed absorption of lime. The ation in the lime content is so slight that no addition of lime in this e can be postulated. The quartz is probably largely a byproduct of alteration of hornblende to epidote. It is also in part introduced, s shown in the microphotograph of a specimen from the diabasic phyry in the East Mine (Fig. 15).

In regard to the chemical form of the mineralization, inferences are ely negative. Evidence of the participation in the mineralization of of the halogens is practically lacking. The apatite and dubious olite indicate some chlorine, but the occurrence of chlorine-bearing erals is so rare that the introduction of the iron as chloride cannot be wn. CO_2 would be expected to show its presence in the formation arbonates in the zone of hydrothermal alteration. No appreciable unt of carbonates was found, except in the occurrences in limestone. presence of much water is shown, however, by the extensive hydro- rmal alteration in the outer zone of mineralization. If the inferences in rd to the temperature of the formation of the ore deposits is correct, more than 200° above the critical temperature of water (358°C.). fact that this water carried material in solution makes it impossible

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to prove that it was in the form of vapor; but the high temperature an the fact that the mineralizers were able to permeate dense rocks so readil makes it probable that they were in the gaseous rather than the liqui phase. The mineralizers are, therefore, believed to have been domi nantly water vapor, carrying with it iron, lime and silica. The variatio in mineralization with decreasing intensity is believed to be as shown i the diagram, Fig. 16.

A later mineralization is represented by the pyrite and chalcopyrite Field observation shows that the pyrite in the ore lies in channels alon which solutions have passed. In the ore it is usually accompanied b chlorite. Where the pyritization has taken place in the wall rocks, a at the west side of the Ocania Mine, both chlorite and sericite occur. Th pyrite seems to represent a very much later stage of mineralization tha the magnetite and hematite. It is impossible to prove any definit

MINERALIZATION DIAGRAM

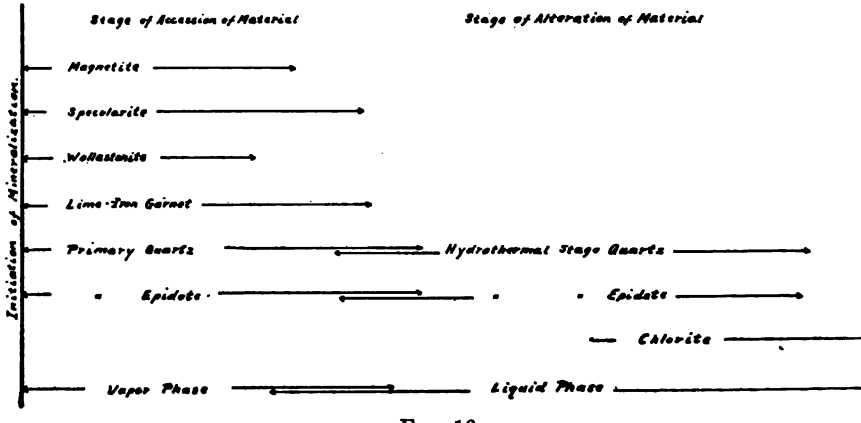

FIG. 16.

connection between the pyritization and the mineralization that produce the orebodies. It seems probable that the solutions that brought th pyrite came in at the time of the intrusion of the later dikes.

Superficial Alteration

W. Lindgren believes that the hematite is the result of the alteration o magnetite by surface agencies. He says:

"——the hematitization is probably a low-temperature process developin gradually under the influence of oxidizing atmospheric waters."[21]

The occurrences in the Firmeza district are in entire accord with thos

[21] W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.* vlo. 53, p. 52 (1916).

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corded by Lindgren at Daiquiri. There is an increase of hematite in
e higher levels of the mines. A polished specimen of the massive ore
om the Chicharron Mine shows very clearly the intimate intergrowth
specularite and magnetite, and some of the specularite forms a later
einlet through the magnetite; but to accept without question specu-
rite as an oxidation product of magnetite is difficult. The writer has
een unable to find any record of the synthesis of specularite except at
gh temperatures, or of the occurrence of specularite as the result of
rface oxidation except under superimposed regional metamorphism.
is not the crystalline facies of hematite that forms under surface
gencies, as the result of dehydration of limonite, in the ores of Mayari.
ndgren does not state that the process of hematitization produces the
ystalline facies of hematite (specularite). But since it is the crystalline
ase that is abundant, and the earthy phase rare, it must be the specu-
rite to which he refers. He does say:[22]

"The pits in the hematite are probably caused by the local development of a fter or earthy facies of the mineral."

spite of the fact that all the observations accord with the theory
lvanced by Lindgren, the writer desires to suggest another hypothesis—
at is, that both the magnetitization and the hematitization are due to a
imary mineralization by gaseous solutions deficient in ferrous oxide.
F. W. Clarke says:[23]

"Ferric oxide can crystallize out as hematite only when ferrous compounds are ther absent or present in quite subordinate amounts, for ferrous oxide unites with it form magnetite."

he crystallization of specularite as a later phase, from a gaseous solution
ntaining insufficient ferrous oxide to produce all magnetite, is in accord
ith Clarke's observation. Deficiency of FeO could account for a change
om magnetite to specularite about centers of crystallization. It could
so account for a diffusion of the hematitization to a higher level, further
om the centers of mineralization than the magnetitization. Local varia-
ons in the mineralizing solutions account for the variations in the pro-
rtion of hematite to magnetite in bodies equally close to the parent
agma.

The only important effect ascribed to surface waters by the writer is
e oxidation of the pyrite near the surface, and the resulting decrease in
e undesirable sulphur content.

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[22] *Op. cit.*, p. 52.

[23] F. W. Clarke: Data of Geochemistry. *Bulletin* No. 616, *U. S. Geological rvey*, p. 347.

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VI. GENESIS OF THE ORE DEPOSITS

Previous Theories

The preceding description of the ore deposits, and discussion of their mineralogy, makes it possible to consider the larger features of their mode of formation. As this is purely a matter of interpretation, it is necessary to consider first the interpretation given by others. J. P. Kimball, writing in 1884–85, and F. F. Chisholm, in 1890, formed their conclusions from observations made in the Firmeza district. A. C. Spencer based his theories on work done at both Firmeza and Daiquiri in 1901. The papers by J. F. Kemp, and by W. Lindgren and C. P. Ross, were written more largely from evidence gathered at Daiquiri, with some specimens and a brief visit by Kemp to the Juragua mines in 1914. J. T. Singewald and B. LeRoy Miller made a brief visit to the Firmeza and Daiquiri districts in the fall of 1915.

J. P. Kimball[24] recognizes two types of ore deposit in the Juragua district. These are "replacements" of the coral limestone by iron-bearing solutions, and "concentrations" of ferric oxide in the diorite, "almost *in situ*." Both replacement and concentration are ascribed to the action of circulating meteoric waters.

F. F. Chisholm[25] discusses the theory advanced by J. P. Kimball, and disagrees with it. He states his own opinion as follows:[26]

"My conclusions, after going in detail over most of the exposure made by the Union cut, were that, whatever the exact character of the ore deposit, the present position of the ore cannot properly be considered the result of local metamorphism of limestones by the action of surface waters containing iron leached from the overlying mass of iron-bearing diorite. I am much more strongly inclined to consider the ore here either the result of concentration within a diorite dyke which was originally characterized by the presence of a large percentage of iron, or else a distinct band forming a portion of a larger dyke. In other words, I am strongly of the opinion that the source of the ore is from below, and consequently that the loyalty of these deposits may be relied on below the limits of atmospheric action. I regret that I was unable to go into the question in detail, and get positive facts in support of my belief, but that I am obliged to admit that my examination was too superficial to enable me to prove my views."

A. C. Spencer,[27] after considering several possible explanations of the genesis of the ore deposits, comes to the conclusion that they are parts of

[24] J. P. Kimball: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Sciences*, Ser. 3, vol. 28, p. 426 (1884).

The Iron-Ore Range of the Santiago District of Cuba. *Trans.*, vol. 13., pp. 613–634 (1884–85).

[25] F. F. Chisholm: Iron-Ore Beds at the Province of Santiago, Cuba. *Proceedings of the Colorado Scientific Society*, vol. 3, part 3, pp. 259–263 (1888).

[26] *Op. cit.*, p. 262.

[27] C. W. Hayes, T. W. Vaughan, and A. C. Spencer: *Report on a Geological Reconnaissance of Cuba*, 1901.

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an older limestone-schist series which has been completely involved in the later igneous intrusions. The following quotation shows how completely Spencer supposed this older series to have been immersed in the igneous rock.[28]

"A mass of many million tons weight floated upward by the buoyant effect of molten rock in motion from the interior toward the surface of the earth, is the only conception which adequately accounts for the mode of occurrence of the orebodies of the Magdalena and the Lola Mines at Daiquiri; while, though less strikingly shown, at Firmeza it is likely that the masses of schist, marble and ore have been likewise actually suspended in the molten lava."

J. F. Kemp[29] divides the orebodies into two types—the "Distinctive Contact Zones" in limestone, and the orebodies in the diorite. The former he regards as due to contact-metamorphic effects produced in the limestones by the granite. Of the orebodies in the diorite he says:

"One is led to the conclusion that while the diorite mass was still hot in the depths or after it had consolidated and had been penetrated by some other and still hot intrusive in depth, a pronounced northwest and southeast fissured zone was formed, up through which came the emissions, fluid or gaseous, which brought the iron for the ore, the pyrite, the garnet and the epidote; the sulphur for the pyrite; and the silica for the quartz, the garnet and the epidote. The lime required by the garnet and the epidote may have been derived from the plagioclase and hornblende of the diorite, or srom included blocks of limestone, or deep-lying limestone, or from several of these fources."

Another quotation from the same article might indicate that Kemp suspected what has become the conviction of the writer:[30]

"One cannot help associating the granitic or pegmatitic dikes with some large parent body. The natural one is the intrusive granite mentioned at the outset. Yet this granite has produced contact zones on the older limestone with orebodies, whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies. We can only suspect the possible connection without being able to prove it."

Kemp apparently bases his conclusion that the aplite dikes represent a later intrusion into the ore on the fact that the dikes appear to cut the ore. That another hypothesis is possible will be shown later.

W. Lindgren and C. P. Ross,[31] on evidence gathered by the senior author, deduced the following conclusions:[32]

"From the above it is clear that the primary iron oxide of the deposits at Daiquiri is a magnetite, which subsequently has been altered more or less completely to a

[28] *Op. cit.*, p. 82.

[29] J. F. Kemp: The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. *Trans.*, vol. 53, p. 30 (1916).

[30] *Op. cit.*, p. 21.

[31] W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vol. 53, p. 52 (1916).

[32] *Op. cit.*, p. 53.

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hematite. The discussion of genesis may therefore be divided into two parts: the first relating to the origin of the magnetite; the second to its subsequent hematitization.

The mineral association of the magnetite, particularly the presence of much garnet, shows that it originated under high-temperature conditions.

On the other hand, the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters."

The problem of hematitization has already been discussed.

Further on three theories discussed by A. C. Spencer are rediscussed by the authors. After dismissing the other two theories, the second theory is accepted, as follows:

"There remains the second theory, accounting for the large masses of magnetite included in the diorite by contact-metamorphic limestone, by which the latter has become almost entirely replaced by magnetite derived from magmatic emanations rich in iron. While it is freely admitted that the genesis of the Daiquiri deposits is difficult to explain, it will be shown that the view outlined in the previous sentence has much in its favor."

J. T. Singewald and B. LeRoy Miller discuss the theories advanced by J. F. Kemp, and by W. Lindgren and C. P. Ross in the papers cited above and come to the conclusions summed up in the following paragraph:[33]

"To sum up our opinions, the Cuban iron ores are contact-metamorphic deposits localized about engulfed blocks of limestone in diorite. In such cases, where there was a limited supply of magmatic emissions, there resulted the contact metamorphism of only a part of the limestone block. Where the supply was ample and the action most intense, not only was the block of limestone completely replaced, but complete endomorphism of the igneous rock on a large scale occurred in the vicinity."

The above are the interpretations offered by others who have studied the field more or less closely. But a study of different exposures and a different viewpoint, have led the writer to conclusions which are somewhat at variance with those quoted.

Hypothesis of the Genesis of the Ore Deposits

Any comprehensive theory concerning the formation of an ore deposit in a rock of which it is not an original part—*i.e.*, an epigenetic deposit—must account for four things:

I. The source of the material forming the deposit.

II. The vehicle that introduced the material.

III. The channel through which the material was introduced.

IV. The cause for the deposition of the material.

Source of the Iron Ore

The ultimate source of the iron ore was the diabasic magma. The extruded part of this magma, with the involved limestones, together with

[33] J. T. Singewald and B. LeRoy Miller: The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, p. 73 (1916).

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e chilled upper part of the magma (the diabases and diorite) form the ost rock of the ore deposits. It has already been shown, in the discus- on of the areal geology of the district, that this magma was differen- ated, that the earlier igneous rocks represent the parent magma and its rly differentiation products, and that the latest rock differentiate was e granite massif with its aplitic apophyses. These aplites, which are, nder anhydrous conditions, extremely viscous, must have contained large uantities of crystallizers, to keep them fluid and enable them to pene- ate the basic rocks as sills and dikes. It is believed that these crys- llizers were mostly composed of water vapor well above the critical mperature, and that they held in an ionized state the ore minerals.

Vehicle to Carry the Material

These concentrated crystallizers were also the vehicle to carry the ineral burden into the rocks which act as host for the ore deposits, *i.e.*, e crystallizers plus the minerals are the mineralizers. Their nature has ready been discussed under the interpretation of the mineralogy.

Channels through which the Materials Were Introduced

One of the striking features of the larger Firmeza deposits is that e ores have almost obviously avoided the masses of marmorized lime- one. This can be explained by the fact that the channels, through hich the mineralizers entered, were not lines of weakness or shear zones oduced by structural faulting, but were cooling cracks in the igneous cks.

Causes of Deposition

The causes of deposition may be of either a chemical or physical ature, or a combination of the two. It has been the tendency of ologists to ascribe to chemical action the dominant rôle in contact- etamorphic actions. This makes it difficult to account for ore deposits two rocks as chemically different as diabasic igneous rock and marmor- ed limestone, except by postulating different theories. If it is believed at the dominating cause of deposition is physical, and is inherent in e mineralizing solutions, whether liquid or gaseous, this difficulty is viated. It is the writer's opinion that such was the case in the forma- on of these deposits. The mineralizers were under such great pressure d at such high temperature that they had the power to diffuse through, d either partially or wholly absorb, any rock with which they came to contact. This diffusion and partial absorption would rob the mineral- ers of a large part of their activity, and would cause deposition of the inerals in a definite order. The order would be dependent on the

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saturation of the gases or solutions with the material to be deposited. The field evidence for such an order is clear, and as shown in discussing the mineralogy of the deposits, magnetite and specularite are the first minerals deposited. As a result, the magnetite and specularite are localized in and near the fissures through which the solutions had access, while the epidote, which is probably only in part due to accession of material from the magma, and is more largely a recrystallization of material present in the host rock, is much more widely diffused. Joseph Barrell, in conversation with the writer, suggested that such a hypothesis may also account for the association of aplitic and pegmatitic dikes with basic mineral segregations. The dikes are regarded as the residual, acidic rock material left in fissures, from which the mineralizers had gone into the rockbody, and in the process had effected a separation of acidic and basic material.

Under the hypothesis advanced, the orebodies owe their localization in igneous rock or limestone almost entirely to the accessibility of the rocks to the mineralizing solutions. The same causes govern all the cases.

Briefly stated, it is the writer's belief that the orebodies in the Firmeza district are due to mineralizers concentrated by further differentiation from the granitic massif, which is itself a differentiate of the diabasic magma. These mineralizers diffused from the igneous-rock material into those rocks to which fissures gave access, and, in diffusing, deposited their mineral burden, and left the rock material with which they had been mixed in the fissures, as aplites and pegmatites.

Under this hypothesis the mineralization becomes a definite event in the igneous cycle, which cycle may be restated as follows:

1. Extrusion of diabasic rock material.
2. Intrusion of diabasic magma.
3. Formation by differentiation of more acidic residual liquid in magma reservoir, with concentration of mineralizers.
4. Batholythic invasion of granitic material, together with mineralizers.
5. Further differentiation of the granitic batholith and escape of mineralizers with some acidic rock material, into cooling cracks in country rock.
6. Separation of basic mineralizers, and acidic rock material, with contemporaneous formation of ore deposits and aplitic dikes and pegmatitic apophyses.

The hypothesis submitted above is given prior to a detailed description of the ore deposits. It must be tested by its application to the phenomena observed in the field, and by its ability to meet the objections raised by other writers on the same subject.

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Discussion of the Different Hypotheses

The hypotheses of the different geologists who have visited the region have been given, either as abstracts or by quotations. The earliest of these, that by J. P. Kimball, does not account for the formation of what are now known to be high-temperature minerals: garnet, wollastonite, etc. Kimball recognized, however, that some of the ores lay in diorite and could not be accounted for by a hypothesis which depended entirely on limestone as a precipitant. F. F. Chisholm does not attempt to formulate a complete theory of the genesis of the ore deposits. With his conclusion that the mineralizing solutions came from below, and were not of meteoric origin, the writer entirely agrees.

The theory accepted by A. Spencer requires an older limestone-schist series. Limestone was found in the Firmeza district in considerable abundance, but schist could not be found by the writer. The occurrence of tuff-limestones interbedded with the volcanic extrusive material makes it seem probable that the period of sedimentation, and the initiation of the igneous cycle, were not separated by a time interval sufficient for the forming of surface orebodies and their entombing in igneous rock. This theory also fails to account for the forming of ore in diorite. In a tunnel under West Five Mine the transition from fresh diorite to granular magnetite can be traced. The transition is gradual and there is no contact which could possibly be interpreted as the margin of a partly assimilated block.

Later writers all agree in ascribing an igneous origin to the mineralizing solutions. The principal divergence is in the amount of influence ascribed to the limestone. Lindgren and Ross believe the ore to be definitely localized in limestone. Singewald and Miller regard the ore as in, and about limestone. Kemp believes that some of the deposits are independent of limestone. In regard to this feature the writer agrees entirely with Kemp. The conception of the orebodies as metamorphosed, engulfed limestone masses is difficult. The shape and position of the ore-masses demand that the entombed blocks should have been comparatively thin, and that they should have come to rest in every conceivable position. This theory completely fails to account for the gradual transition from ore to diorite shown at its best in West Five Mine. Such a deposit could be accounted for only on the basis of complete assimilation of the limestone, and a deposition of magnetite about the locus of assimilation. Any theory that places the burden of producing the orebodies upon the diorite, is open to one great objection. It demands that a magma of almost gabbroic basicity shall have produced contact-metamorphic phenomena of the most intense variety. Kemp recognized this, and ascribes the orebodies in limestone to the granite, and the orebodies in the diorite to solutions from "some other" intrusive. But Lindgren and

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Ross, and Singewald and Miller regard the diorite as the source and as the cause of the mineralization. Another objection is the fact that, under that theory, the diorite must have exercised a strong selective tendency. Unaltered marble within 20 ft. of massive ore, and separated from the ore by recognizable diorite, can be seen at West Four. Such a case is not rare, it is a common feature. In one of the old cuts above West One Mine there is exposed a small block of marble, completely surrounded by diabase containing small ore masses. But the marble is entirely unaffected. Contact-metamorphic phenomena are undoubtedly capricious, but if another explanation will solve the problem in such a way as to show that the capriciousness is more apparent than real, it must at least be considered.

The great objection to ascribing the source of the ores to the granitic batholith lies in the interpretation of the aplitic and pegmatitic dikes. Field observation clearly establishes the direct connection between these rock types and the granitic massif. If these most acidic facies represent a later intrusion into the ore, the ore cannot come from the granite. Kemp believes they are later. He says:[34]

"——whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies."

Lindgren and Ross[35] agree with this.

"The dikes of igneous rock[35] intruded into the iron ore appear to be somewhat different from the diorite and are either granite porphyry or aplite; that is, complementary dikes of a later generation."

F. Klockmann, writing on contact-metamorphic magnetite deposits in general, puts the matter even more strongly:[36]

"Wiederholt wird angegeben, dass aplitische Gänge und Granitapophysen die Magnetitlager stätte durchsetzen. Gibt es denn für den Unbefangenen dafür noch andere Deutung als die, dass der Granit in eine von ihm vorgefundene Lagerstätte eine Apophysen entsendet hat?"

As a general proposition, the very fact that aplitic dikes and granite apophyses are repeatedly reported as traversing magnetite deposits is the strongest possible argument against the purely fortuitous intrusive nature of such apophyses. It was this constant relationship observed in the field study, that first convinced the writer that there must be definite connection between the forming of the orebodies and of the granitic apophyses.

J. H. L. Vogt recognizes a connection between the granite and its apophyses, and the magnetite deposits of Kristiania. He says:[37]

[34] *Op. cit.*, p. 21.

[35] *Op. cit.*, p. 54.

[36] F. Klockmann: Über Kontaktmetamorphe Magnetitlagerstätten. *Zeitschrift für Praktische Geologie*, vol. 12, p. 81 (1904).

[37] J. H. L. Vogt: Problems in the Geology of Ore Deposits. *Trans.*, vol. 31, p. 138 (1901).

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"A study of the Kristiania contact-deposits indicates that the formation of the ores preceded the solidification of the granitic magma. Even when the ores occur in slates immediately adjacent to the granite, or in the small Silurian fragments completely surrounded by granite, they are never found also in the granite itself. This is to be simply explained by the supposition that from the still liquid magma the ores were 'blown into' the adjoining rigid rocks. —The presence in these deposits of granitic apophyses, already mentioned, is another proof that they were formed before the solidification of the granite."

He regards the occurrence of the apophyses with the ore in the Kristiania district as proof that the granitic magma had not yet solidified.

The absence of ilmenite in the Firmeza occurrence is also an indication of their intimate connection with the granitic magma. Vogt has given some very complete studies of the relationship between the different

FIG. 17.—APLITIC GRANITE BORDERED BY MAGNETITE AND EPIDOTE. TUNNEL AT ELEVATION OF 299 FT. UNDER WEST FIVE WORKINGS. MAGNETITE CAUGHT IN APLITE = M. ACTUAL SIZE. Aplite—white. Magnetite—dark.

magnetite deposits and their sources. A quotation that shows some of his conclusions follows:[38]

"Die Erfahrung ergiebt, dass bei der 'oxydischen' Erzaussonderung es namentlich das 'Eisenoxyd-mineral'—bei den Gabbros Titaneisen oder Titanomagnetit, bei den Peridotiten Chromit—ist, welches concentrirt wird. Es ist somit a priori anzunehmen, dass etwaige 'oxydische' Erzausscheidungen in den Graniten einen ähnlichen niedrigen TiO_2-Gehalt führen müssen wie die in dem Granit normal ausgeschiedenen (wohl namentlich Magnetit und Eisenglanz, untergeordnet Titaneisen.)"

The strongest evidence of the intimate relation between the granitic apophyses and the mineralization is based on field observation. Some of the small dikes are reproductions in miniature of the larger occurrences. Figs. 17, 18 and 19 are photographs of two of these small dikes.

Fig. 17 is of a specimen from the ore in the tunnel under West Five

[38] J. H. L. Vogt: Zur Classification der Erzvorkommen. *Zeitschrift für Praktische Geologie*, vol. 2, p. 394 (October, 1894).

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Mine. This is as deep-seated an ore deposit as has been exposed, and is the one most clearly independent of limestone. It is noticeable how entirely clear-cut is the contact between aplite and magnetite in the lower part of the picture. At the same time the other contact is less sharply defined and clearly shows some magnetite trapped in the aplite.

FIG. 18.—PHOTOGRAPH OF SPECIMEN FROM EAST MINE, SHOWING RIM OF GARNETS DEVELOPED BETWEEN APLITIC GRANITE (WHITE) AND DIABASE PORPHYRY (DARK). ACTUAL SIZE.

Fig. 18 is a photograph and Fig. 19 a microphotograph of a specimen from the East Mine. This specimen comes from the area of diabasic rocks, that lies between the ore in the East Mine and the ore in the pit above, known as North Mine. The specimen shows garnet, merging

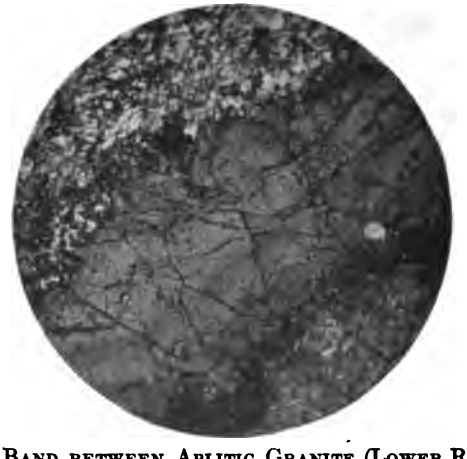

FIG. 19.—GARNET BAND BETWEEN APLITIC GRANITE (LOWER RIGHT) AND DIABASE PORPHYRY (UPPER LEFT). FROM EAST MINE. × 18.

on one side into diabase porphyry, on the other into aplite. The writer can see only one explanation for this specimen, and that is a change in the mineralizing solutions from intermixed mineralizers and acidic rock material to aplite.

It is on the basis of such occurrences, and for the reasons given, that

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the writer believes: that the aplite and ore are phenomena produced at approximately the same time; that both were formed from a mixture of mineralizers and rock material derived from the granitic magma; and that the iron oxides, lime and the major part of the silica, held in aqueous solutions above the critical temperature, diffused into the wall rock, leaving the residual rock material in the fissures as aplitic dikes.

VII. DETAILED DESCRIPTION OF THE ORE DEPOSITS

In order to test the hypothesis stated above in regard to the genesis of the ore deposits, it must be applied to the different occurrences of ore in the district. To do this, a detailed description of the more carefully studied mines will be given, and the mode of formation of the orebodies discussed.

Ocania Mine

The Ocania is the most westerly of the Juragua Iron Co.'s mines. It lies on a steep hillside, at an elevation of from 860 to 1,085 ft.

Fig. 20.—Specimen from Ocania Mine, Level 2. The Dark Material is Garnet. The Light Material is Wollastonite. The Structure is Attributed to Diffusion. Actual Size.

The wall rocks are coarse, diabase porphyry, volcanic agglomerates, and marble. The contact between the granitic and diabasic rocks lies on the south slope of the hill, at an elevation of about 700 ft. A quartz-bearing, andesitic dike is found in the mine itself. The ore forms a series of lenses that strike in a northwesterly direction, and have a steeply inclined dip to the southwest. These lenses are made up almost entirely of hematite and quartz, with little or no magnetite. There is considerable pyrite through the ore in various places, and, at the edge of the limestone, chalcopyrite and manganese oxide are found. Some fragments of diabase, with native copper in them, were observed near the limestone. Between the ore and the limestone are masses of garnet rock. Near the bottom of the orebody, as exposed on the southeast side of the mine, was

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found a rock (Fig. 20) which shows, in part, alternating layers of brown garnet, and a greenish mineral which the microscope shows to be wollastonite. As the accompanying microphotograph (Fig. 21) of the contact between the garnet and wollastonite shows, the two bands have an interlocking structure, perpendicular to the boundary. The black crystals in the wollastonite band are magnetite. The garnet does not show crystal edges. Toward the center it is very dusty, with undeterminable inclusions. The wollastonite forms an aggregate of poorly developed, prismatic crystals. A somewhat similar structure, made up of magnetite and actinolite, is shown in a specimen from the Juragua Mines described by W. Lindgren. He says:

"The texture of this specimen strongly suggests diffusion banding, such as might form in a material, a limestone perhaps, freely penetrated by hot iron-bearing solution."

FIG. 21.—CONTACT BETWEEN GARNET AND WOLLASTONITE IN ROCK SHOWING DIFFUSION STRUCTURE. FROM OCANIA MINE. × 18. Magnetite—black. Garnet —dark. Wollastonite—grey. Holes in thin section—white.

In the case of the Ocania specimen the solutions were probably lime-bearing and lower in iron. Whether the original rock was limestone or diabase porphyry, it is not possible to tell. It is true of this mine that the mineralization effects have spread further into the diabasic rocks than into the marble. The clean white marble is separated from the ore by only a few feet of garnet rock, while the diabase is garnetized, near the ore, and epidotized for tens of feet from the ore. The ore itself is so complete an alteration of whatever wall rock it replaces that there is no *prima facie* evidence of what it originally was. The pyritization is later than the principal period of ore deposition, the pyrite forming in fractures and cavities in the ore. The chalcopyrite accompanies the pyrite.

Surface action, producing partial hydration of the ore and considerable infiltration of secondary calcite, obscures the evidence.

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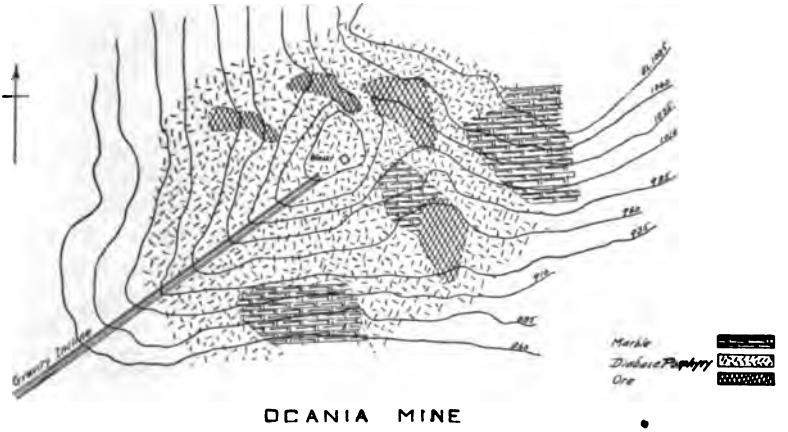
Northeast of the mine workings is the largest exposure of marble found in the district. It is cut by sills of an entirely epidotized, basic, igneous rock. It is separated from the orebodies by massive garnet rock, or highly garnetized and silicified diabasic rock. The map (Fig. 22) shows the relation of orebodies to wall rock. The commercial ore is sharply bounded, but the mineralization is not.

The striking features in this mine are the absence of magnetite and of highly acid wall rocks, and the clear evidence that the diabasic rocks are more diffusely mineralized than the marble. At the same time the resemblance of the blocks of ore to engulfed limestone blocks or to roof pendants, is noticeable.

If the Ocania Mine were the only one studied, it would be difficult for the writer to defend his hypothesis. There are no aplitic or pegmatitic



OCANIA MINE
FIG. 22.

rocks in evidence. Except for one dike of doubtful age, no quartz-bearing igneous rock is exposed in the mine. There is abundant limestone.

There are also some features that are not explained by any theory involving the diabase as the ore-bringing rock. The diffuse mineralization of the diabasic porphyry indicates a hydrothermal effect, later than the consolidation of the diabase. It is difficult to ascribe to the diabase porphyry a hematitizing effect on included limestone, or even a magnetitization free of chrome or titanium minerals, with a superimposed hematitization. The fact that the larger part of the marble has suffered no mineralization is also unexplained by any theory that makes the overwhelming of the limestone by the diabase the cause of mineralization.

On the other hand, if the solutions are supposed to come from the granite, the difficulties of explanation are not so great. A fissure in the diabase near the limestone, localized the ore and the mineralization in

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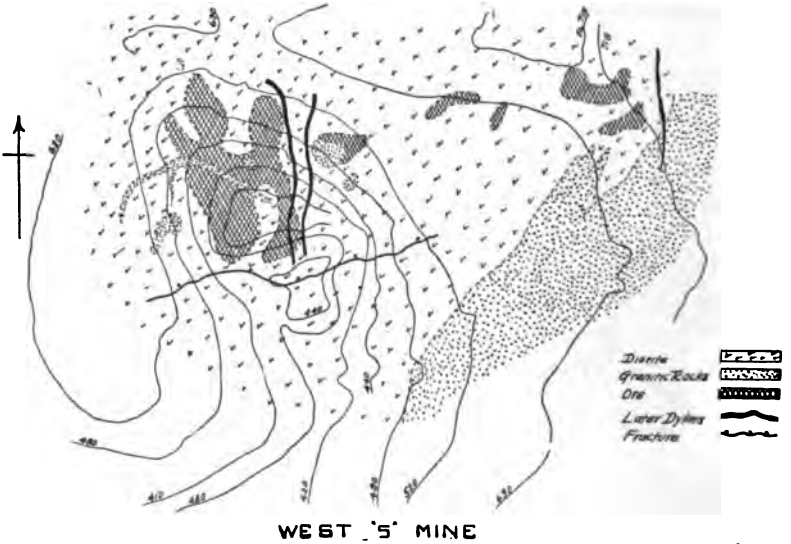
both formations. The distance of the ore exposed from the parent magma accounts for the preponderance of hematite over magnetite.

West Five Mine

This mine is the furthest west of the mines in the vicinity of Firmeza, and lies at an elevation of from 420 to 718 ft. There is a tunnel under the mine at a level of 299 ft.

The ore lies in flat or in steeply inclined lenses. It is almost entirely magnetite, the enclosing wall rock diorite, with considerable, more or less aplitic, granite through and near the ore. The diorite varies from



WEST "5" MINE
Fig. 23.

the even, granular type shown in Fig. 8 to a porphyritic type. No fragmental diabasic rocks occur, nor is there any trace of limestone.

The rocks from the aplitic dike, in the west side of the mine, were described under the granitic rocks. The connection between aplite and ore is shown on a small scale in the photograph (Fig. 17) of a small dike of aplite from the tunnel under the mine. The magnetite and epidote lying immediately next to the aplite, merge gradually into the granular diorite. There can be no question of any rocks being involved other than the diorite and the aplite. Their inter-relationship has been discussed in considering the question of genesis. Considerable garnet is found in the mine workings, and its occurrence is similar to the magnetite and epidote. In one small exposure the position of the garnet, as a contact effect in the diorite, next to the granite, is clearly shown. This

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the north side of the bottom level. The epidotization and silicifica-
are more widely diffused from the granite than are the magnetite
garnet. One specimen, from the lowest level, showed an inter-
h of apatite with magnetite, and garnet and epidote, cut by later
ts of epidote (Fig. 14). This is the only case in which apatite
nted to more than a very minor accessory.

n the south side of the mine is an east-west fault steeply inclined to
orth. It is obviously post-mineral, and appears not to be of very
throw.

mming up the evidence from this mine we find: no limestone;
aplitic or porphyritic granite; almost no specularite; but much
etite, garnet, epidote, quartz—all in diorite.

he ore deposits of this mine offer the clearest evidence that lime-
could not have been the dominant cause determining the locus of
deposition. There is no limestone in the immediate vicinity. The
tion from ore to diorite is gradual. The mineralogy shows none of
ell-crystallized calcite that is so abundant in the ore deposits that
obviously formed in limestone.

he intimate relation between ore and aplite is also well shown
The dike exposed in the west side of the workings has led other
vers to believe the granitic apophyses later than the ore, but as
n under the discussion of the various theories, the same phenomena
terpreted by the writer to indicate the contemporaneous formation
ore and the aplite.

l the phenomena can be explained on the hypothesis advanced.
is that the aplite, the magnetite, and the lime silicates are all the
of the one mineralizing period.

Loma Alta Mine

he Loma Alta workings lie at the top of a hill, just east of West
Mine. They are at an elevation of from 900 to 1,000 ft. As they
been practically abandoned for some time, weathering agencies
obscured much of the evidence. Cropping out on the east side and
ng under the workings at a flat angle, is a sheet-like mass of
yritic, pegmatitic granite. In the southwest corner of the workings
gnizable diabase porphyry. What is left of the ore is earthy hema-
ing on the granite with some kaolin and chlorite. All around the
ngs on the hill are rounded boulders of more or less pure magnetite.
ssociation of the granite, diabase and ore in this mine is suggestive,
he evidence is too obscured to be conclusive. The earthy hematite
ubtedly owes its origin to the weathering of the primary, crystal-
on oxides. There is no reason to believe that the process did not
e the formation of limonite and its dehydration.

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West Three Mine

An exposure in this mine, which lies on the same hill, south of and below Loma Alta Mine, is worth describing. It is a mass of magnetite, in diabase, directly next to a vertical pegmatitic granite dike. The rocks are all fairly unaltered, and the contact relationship well shown. There is no reason to suspect that any limestone was involved.

West Four Mine

This mine is located northeast of Loma Alta, at an elevation of from 645 to 935 ft. At the north end of the mine, in the upper levels, is a mass of clear white marble, entirely unmineralized. The wall rock in the upper levels is diabase porphyry; in the lower levels it is diorite. The diabase and diorite are both cut by somewhat porphyritic and pegmatitic granite. The ore is magnetite, and hematite and magnetite, and lies in lenses, whose major axes vary from a vertical to a horizontal position, on top of and next to the granite. These lenses occur mostly in the diorite, partly in the diabase, but not, so far as exposed, in contact with the marble. The entire series, diabase, diorite, granite, ore, is cut by a large dioritic dike. The mineralization effects are no different from those in West Five Mine. The chief difference is that there is marble exposed in the West Four workings. So far as can be determined it has had no direct effect on the formation of the ore deposits.

West One Mine

The West One workings are located north of Firmeza along the railroad tract. Exposed for a long time and not much worked of late years, they offer a poor field for study. Altered diabase and diorite, highly mineralized, are the wall rocks. Neither limestones nor granite are in evidence. From one of the cuts that is being worked, come the well-developed pyrite crystals, of which one is shown in Fig. 13. This exposure shows the pyrite all through the magnetite in cavities and along minute fractures. It establishes the age of the pyritization as definitely post-magnetitization.

Fig. 24 is a photograph of a drawing made with a camera lucida, from a thin section of a rock from West One Mine. The specimen is dark and dense, and shows patches of magnetite. In thin section the rock proves to be a diabasic porphyry, partly replaced by magnetite. The replacement is apparently independent of the mineralogy of the replaced rock. The magnetite occurs in patches in the groundmass, or in the phenocrysts, or partly in each.

West One Mine offers no new evidence in regard to the genetic problem. It does offer, however, an excellent field for the study of the pyritization.

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East Mine

ne East Mine itself lies east of Firmeza, at an elevation of from 440 0 ft., but it cannot be considered apart from the small cut just to orth at about 800 ft. elevation, known locally as the North Mine. ne large orebodies of this mine are made up of specularite and etite, with much epidote quartz and garnet. They lie in the lower of the mine around bosses of aplitic granite, in diorite and diabase yry. Fig. 18 is a photograph and Fig. 19 a microphotograph lime-iron garnet rim along a small aplite dike, in recognizable se porphyry. It shows the entire ability of the mineralizers that panied the aplite to garnetize the diabase.

. 24.—Photograph of a Drawing Made with a Camera Lucida, from a Section of a Specimen from West One Mine. It Shows a Diabase Por- Partly Replaced by Magnetite (Solid Black). The Phenocryst is oclase. The Lath-Shaped Crystals Are Plagioclase. The Ground- s Granular Magnetite and Chlorite.
the upper half of the drawing the interstitial magnetite has been left out.)

g. 26 is a microphotograph of a thin section of a completely epido- nd silicified diabase. The specimen comes from within 2 ft. to e shown in Fig. 15. The two show successive stages in the hydro- al metamorphism.

the upper levels, extending toward the North Mine workings, veral included sheets of completely marmorized limestone. The Mine workings, now abandoned, show remnants of orebodies. ore is different from any found in the mines so far described. Well- oped rosettes of specularite blades in garnet quartz matrix; well- llized epidote in abundance; coarse quartz; much recrystallized ; in short all the components are found of a typical contact-meta- ic deposit in limestone.

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In the East Mine, as in the others, later andesitic and dioritic dikes are in evidence. A northwest-southeast fault, pitching to the northeast, has cut the ore and aplitic granite as well as the older rocks. It has brought the most basic of the granitic rocks, the quartz-bearing diorite, in contact with the ore. That the specimen described as a wall rock of the East Mine by J. F. Kemp[39] came from this mass of quartz-bearing diorite seems probable. The rock in which the ore is found is of a much finer grain.

In the East Mine itself there is no evidence that limestone had any share in forming the orebodies. In the workings known as the North Mine, limestone is without doubt the host rock of the orebodies.

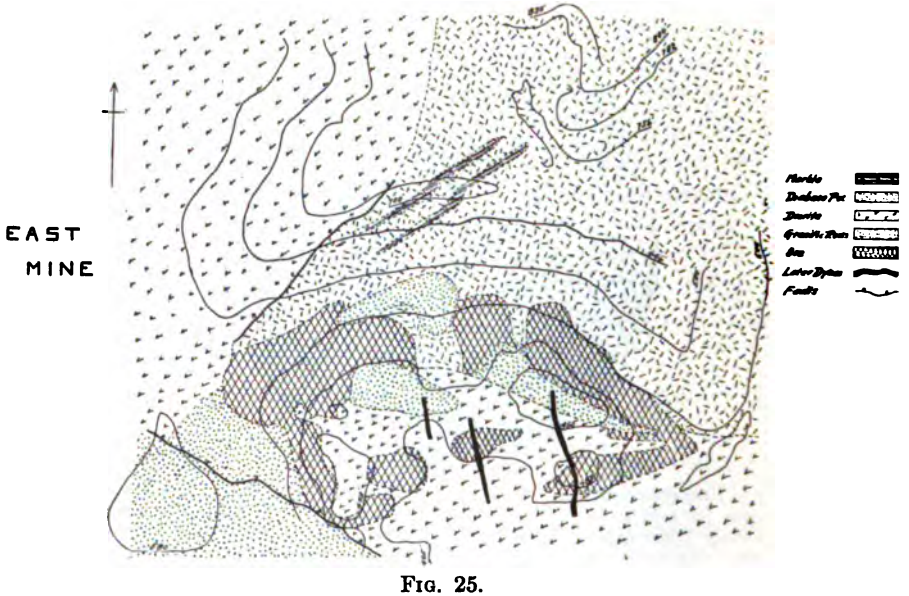

FIG. 25.

The ore in the main workings, with the associated granitic rocks, was formed at centers of intense mineralization. The ore deposits of the North Mine were formed at a greater distance from the centers of mineralization. The aplite dikes in the terrane between the two are the filling of the fissures through which the solutions traveled.

CHICHARRON (NORTH EAST) MINE

Located about ¾ mile northeast of the East Mine and now entirely idle, are the workings of the old Chicharron Mine. This mine is well north of the zone in which most of the mines lie, and in some respects differs from the others.

[39] *Op. cit.*, p. 15.

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The face of the mine is a cliff of diabase, over which the water of the nevolencia River falls. This diabase is more or less mineralized, and

. 26.—Diabase Porphyry Completely Replaced by Epidote and Quartz. From East Mine. × 18. Quartz—light. Epidote—dark.

laced by magnetite, wollastonite, and epidote. The brook bed follows ng the bottom of the cliff, and with an overgrown talus pile, separates cliff from a mass of marble. Cutting the marble is a dike of

Fig. 27.—Ore from Chicharron Mine. × 18. Magnetite—black. Wollastonite—white.

gnetite, with interstitial wollastonite in small amounts (Fig. 27). C. Spencer describes this occurrence as follows:[40]

'At this place the magnetite seems to be an intrusive dike cutting across a mass rystalline limestone into which it sends a short apophysis."

[40] *Op. cit.*, p. 80.

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The magnetite probably formed a dike, instead of a more diffuse forn deposit, because while the temperature was high, the pressure was As a result, the diffusive power of the mineralizers was not great, altho they could form high-temperature minerals.

In the Chicharron Mine, also, the later dikes are in evidence.

Estancia Mines

To the east of East Mine, in the mineralized zone, lies Estancia All around the hill at from 700 to 800 ft. elevation lie mine worki The ore in these workings is either magnetite and specularite repla diabase, or it is the hematite-garnet-calcite combination that is eloqu of contact-metamorphic limestone, though no limestone is left. Aro most of the hill, and all of the south side, is a railroad track at an eleva of about 500 to 550 ft., and it is nearly all cut in a porphyritic gra in which micropegmatite predominates. In the valley north of Esta Hill granite is exposed along the trail. The dump of an old tunnel into Estancia Hill for 600 ft. at an elevation of 695 ft. is granite, an W. Gaumer, the Juragua Iron Co.'s chief engineer, informed the w that all the tunnel was in the same rock. South of the railroad t runs the lower trail to Concordia, and it is all in diorite, or coarse diab which has been mapped as diorite.

All this evidence seems to indicate that a sheet of porphyritic gra intruded into diabasic rock with included marble, to form a sill. orebodies at the upper contact of the granite formed in that rock to w the mineralizers had access.

Concordia Mine

This, the most easterly of the Juragua Iron Co.'s mines, offers no evidence. It is the lowest of the mines, being at an elevation of from to 450 ft. The rock and ore association is magnetite in diorite granite near by.

The ore deposits are lenticular, and as a rule lie in an almost horizo position. No limestone is in evidence.

VIII. GEOLOGIC HISTORY

The evidence from which the geologic history of the Firmeza dis must be deduced is scanty. Almost all the field work was done in immediate vicinity of the mines, and little search was made in the rounding country for evidence which might give completeness to geologic history. Enough material was found to make possibl sequential arrangement of the geologic events that produced the pre

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topographic, petrologic, and mineralogic features of the district. These geologic events will be discussed in the order of their occurrence, as follows:

1. Sedimentation.
2. Igneous cycle (including ore deposition).
3. Uplift with tilting and erosion.
4. Deposition of coral limestone.
5. Emergence.
6. Submergence.

Sedimentation

The formation which appears to be the oldest in the district is the marble. As has been shown in the discussion of the rocks, this marble is Mesozoic, and probably Cretaceous in age. The constituents of the floor upon which the limestones now represented by marble were deposited, cannot be determined; nor is it possible to estimate the thickness of these early sediments, because of the lack of any continuous section. The limestone found at the highest elevation is a tuff-limestone. Because of its position it may be considered as the youngest of these early sediments, and because of the volcanic fragments embedded in it at the time of its deposition, it may be considered as representing the connecting link between the period of sedimentation, and the initiation of the igneous cycle.

Igneous Cycle

The igneous cycle began with a period of volcanic activity, to which the clastic igneous rocks of basic nature bear witness. These volcanic rocks form the southern slopes of the Sierra Maestra range, and the tops of some of the foothills. They contain remnants of interbedded limestones, and must have formed contemporaneously with the latter part of the period of sedimentation.

The next event was the invasion of these volcanic rocks and sediments by a magma of basic composition. Of this "parent" magma the present representatives are the diabasic and dioritic rocks. These rocks were left in their present position partly by the chilling of the magma, and to a lesser degree, by intrusion into the sediments and volcanic rocks.

The invasion of basic magma was followed by a long period of differentiation. During this period the liquid residue in the magma chamber became more acidic, and charged with mineralizers. This acidic liquid, with its burden of mineralizers, invaded the earlier, chilled, basic, igneous rocks in the form of a granitic batholith.

Further differentiation, resulting in a concentration of the mineralizers, ended in the injection of the mineralizers, with admixed rock material, into cooling cracks in the basic rocks above the batholith.

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There followed a separation of the highly heated mineralizers from t admixed rock material, which resulted in the formation of the ore dep and the aplitic and pegmatitic apophyses.

It is uncertain to what part of the igneous cycle the marmoriza of the limestone should be ascribed, because all the marble exposures near or in igneous intrusive rocks, which range from diabase to gra Any one of these may have been competent to produce the marmo tion. It seems probable, however, that because it was the earliest, invasion of the diabasic magma is responsible for the greater part of marmorization.

What events were taking place at the surface during the igneous c is a question to which no answer has been found. The formation of c ing cracks, however, in the basic igneous rocks, seems to indicate they were not very far from the surface at the time of ore deposition.

The last episode in the igneous cycle was the intrusion of basic d into fissures formed in all the igneous rocks and their associated deposits.

Uplift, Tilting, and Erosion

Because of the almost complete marmorization of the older limesto their bedding is difficult to determine. But the pitch of the extru and the few visible remnants of stratification in the marble, indica definite dip to the southeast. Evidence of uplift is found in the fact limestone occurs at elevations up to 1,400 ft. above sea level.

As has been shown in the discussion of the areal geology, there i conclusive evidence as to the exact time relation between the ign cycle and the period of uplift and tilting, but because pressure is certain extent a function of depth, it seems probable that the ore dep were all formed in an approximately horizontal plane. Their pre position along an inclined plane would, in that case, be some mea of the differential elevation. Meager and inconclusive as the evidenc it must be considered, and until evidence to the contrary is found, uplift and tilting should be regarded as later than the igneous activit

The coral limestones themselves show no evidence of tilting and fact places the tilting as older than the coral limestones—that is Pleistocene. Emergence has, however, continued.

Fragments of iron ore in the base of the later coastal limestones p that this period of uplift was accompanied by erosion sufficient to into the ore deposits.

So far as the ore belt is concerned, the later record is one of conti erosion. Whatever else may have happened to that part of the dist no record remains except one of superficial alteration and erosion. coral limestones and the valley topography show that dynamic fo were not entirely quiescent.

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Deposition of Coral Limestone

The deposition of the Pleistocene-Recent coral limestone began when the land was about 350 ft. lower than at present. Since then there has been oscillation with predominant emergence.

Emergence

The sea-cut cliffs in coral limestone, rising in terraces from the coast to an elevation of 350 ft., are evidence of the periodic emergence of the land since deposition of the limestone began. What the extent of the emergence was cannot be determined on the evidence in the district, because it has been masked by the most recent geologic event—submergence.

Submergence

A. C. Spencer,[41] judging from other parts of the island, places the amount of the submergence at from 40 to 70 ft. If the thickness of the fluviatile deposits in the east-west valley could be determined, it would give an accurate measure of the amount of submergence in this district.

For reasons that have been discussed in detail under the interpretation of the topography, the writer believes that this submergence represents a movement of the sea.

The geologic history of this district is a record of quiet activity. There is no regional metamorphism or profound fracturing, such as would be expected if vast diastrophic movements had taken place. The only metamorphism that has left a record was due either to igneous or to superficial causes. The only fracturing recorded is the result of the shrinkage of igneous rocks in cooling, and gives evidence of little or no local movement. Igneous forces, acting through long periods of time, dominated the rock formations. Slow uplift, alternating with periods of almost complete quiet, and accompanied by much erosion, formed the topographic features.

IX. ECONOMIC APPLICATION

Any geological investigation of more than a purely academic interest must answer some questions of a commercial nature. The present study of the ore deposits of the Firmeza district, by its inquiry into their origin and extent, is able to answer certain questions which are of economic interest. These questions refer to the continuation of the deposits in depth, a possible change in their composition if they do extend downward, and the favorable locus for the search for more ore.

[41] *Op. cit.*, p. 34.

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Continuation of the Orebodies in Depth

Since the mineralization came from below, from the granitic massif, there is no reason to believe that the ore will not extend at least to the granite. On the other hand, there is no reason to believe that the separation of the mineralizers from the granite was incomplete, so that no ore can be looked for in the granite. Where the ore lies on top of sheet-like granitic masses, like the Estancia and Loma Alta deposits, a definite limit in depth is in sight, but in workings where the massive granite is not in evidence, extensive development in depth seems probable.

Character of the Ore in Depth

The pyritization in the ores at Firmeza has made the owners of the ore deposits fear that the ores would change to pyrite in depth. However, the pyritization is later and independent of depth. There is no reason to believe that the ores found at greater depth will be any higher in sulphur than those now being mined in the lower levels of the larger mines. The lower sulphur content of the ores close to the surface is due to superficial oxidation of the pyrite. Below the level of surface oxidation the sulphur content may be expected to remain fairly constant.

The only change that seems probable is an increase in the proportion of magnetite to hematite, as the workings approach the deeper centers of mineralization.

Favorable Locus for Exploration Work

The problem of exploration work may be considered in two ways—positively and negatively. Both are of fundamental importance, although frequently the question of where to look for more ore is considered, while the equally important question of where not to look for ore is neglected. It is just as necessary to discover in which localities the search for ore will be fruitless and therefore any exploration work would be valueless, as it is to know where the search for ore will be profitable.

Obviously, rocks formed later than the ore, like the coastal limestone, cannot be expected to contain ore deposits. The granitic massif, from which the ore solutions are thought to have escaped, is also to be regarded as barren ground.

On the other hand, the granitic apophyses, aplites and pegmatitic granite are the best guides as to the location of orebodies. It is their upper surface which merits the most careful search.

Since the orebodies contain much magnetite, the magnetic survey is undoubtedly the best guide to their location. But a careful study of

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formations is necessary to interpret such a survey, and to decide
t areas of magnetic attraction are most valuable, and also what areas,
ite of a lack of this attraction, offer fields for development.
The economic application of geologic evidence will be successful only
ne interpretation of the geologic phenomena is correct. At the same
the ultimate test of any geologic theory is its application in the
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Effect of Sulphur on Low-Carbon Steel

BY CARLE R. HAYWARD,* S. B., CAMBRIDGE, MASS.

(New York Meeting, February, 1917)

SULPHUR has long been one of the banes of the steel manufacturer and often no effort and expense have been spared in order to reduce it to a small per cent. in the finished product. This condition is due to a general conviction that in many cases where steels have failed in service, sulphur has been the cause. But there has been a growing feeling in recent years that the verdict against sulphur has been unnecessarily severe. In cases of segregation it was present in augmented amounts along with other impurities, but it had not caused the segregation. High sulphur in pig iron is caused by poor furnace conditions and the sulphur is merely one indication of an iron that has not been properly reduced. No amount of subsequent treatment under oxidizing conditions in the open-hearth furnace can remedy the defects, although the per cent. of sulphur may be considerably reduced. In other words, the causes of bad steel can frequently be traced back to bad pig iron, and sulphur is merely one indication that the pig iron is bad. The writer recently visited a steel plant where a mass of evidence had been accumulated which substantiated this fact, and the superintendent was emphatic in stating that high sulphur was not harmful provided the steel was not otherwise poor due to insufficient reduction in the blast furnace.

The presence of a moderate amount of sulphur is desirable from the standpoint of the man who machines the steel. The low-sulphur material drags and the production of a smooth surface is very difficult. A slight increase in sulphur enables the machinist to produce a smooth surface without difficulty.

Since, therefore, such large quantities of steel are subjected to machining, it becomes highly important that the sulphur controversy should be settled, and if its presence is proved to be harmless the ban on it should be lifted.

Among the recent papers on the effect of sulphur on steel is one by Dr. J. S. Unger, Manager of the Central Research Bureau, Carnegie Steel

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Co.[1] The results of an exhaustive series of tests are given and the conclusion states: "The author does not advocate paying no attention whatever to sulphur content in steel but believes firmly that a steel containing less than 0.100 per cent. is not necessarily bad, and that it will show little, if any, difference in quality when compared with the same steel of much lower sulphur, other conditions being the same."

The present investigation was undertaken at the suggestion of A. H. Annan of the Rhode Island Tool Co., who coöperated by furnishing the steel and machining the specimens. The results are presented as a contribution to the general knowledge on the subject.

Steels Used

It was planned to use three steels of different sulphur content but with the other elements the same. The manganese was an exception, however, for with this element part is in the form of MnS existing free in the steel and the remainder is dissolved in the steel. It is evident, therefore, that the manganese should vary but that the amount in excess of MnS should be constant in the different steels.

The steels finally selected were in the form of ¾-in. round bars. Two bars of each grade were required to furnish sufficient specimens. The analyses are shown in Table 1.

TABLE 1

Mark	Carbon, Per Cent.	Total Manganese, Per Cent.	Excess Manganese, Per Cent.	Phosphorus, Per Cent.	Silicon, Per Cent.	Sulphur, Per Cent.
1	0.18	0.55	0.48	0.007	0.01	0.038
1A	0.18	0.57	0.50	0.009	0.02	0.041
2	0.17	0.67	0.52	0.008	0.01	0.086
2A	0.18	0.70	0.55	0.010	0.03	0.087
3	0.18	0.80	0.54	0.006	0.02	0.152
3A	0.17	0.80	0.55	0.011	0.03	0.148

Heat Treatment

In order to make a comparison of the steels under different conditions it was decided to heat all the specimens to a temperature just above the critical range, quench in water, and reheat different lots to 300°, 400°, 500° and 600°C. respectively. For this purpose the bars were cut into 7-in. lengths, which was sufficient for tensile specimens and specimens for microscopic examination.

The furnace used is shown in Fig. 1. The muffle is made of alundum

[1] *Iron Age*, No. 97, pp. 146–150 (1916).

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is 2 in. high, 9 in. wide and 16 in. long. It is wound with No. 15 ello resistance wire.

In order to obtain uniform heating of the specimens, they were sup- ted on an asbestos rack, as shown. Asbestos shields were placed on sides and ends of the rack so that the specimens were practically in uffle within a muffle.

A platinum platinum-rhodium thermocouple was introduced through back of the muffle into the center of the heating chamber and con- ted to a Siemens and Halske recording galvanometer.

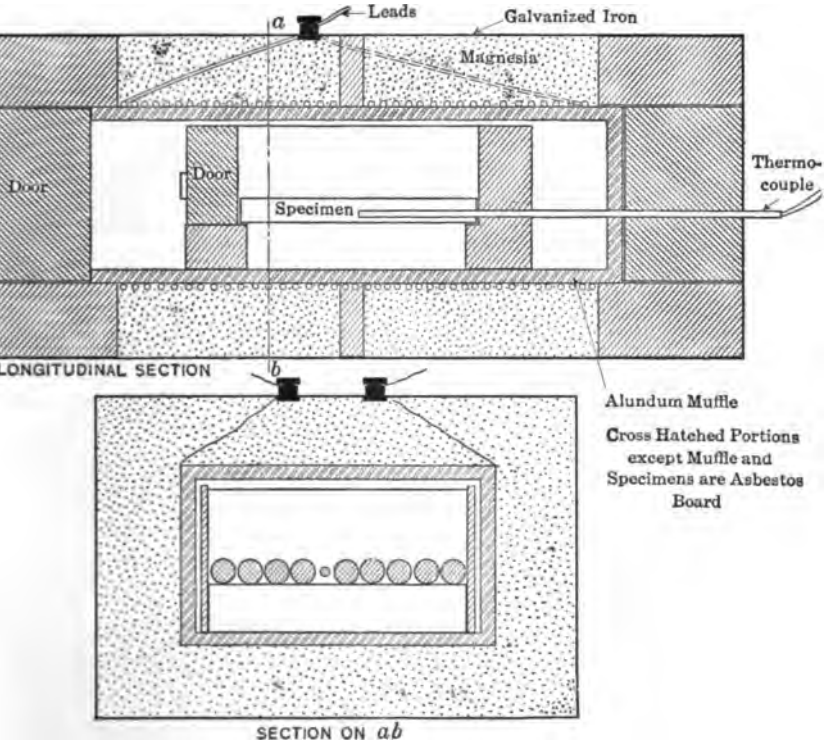

FIG. 1.—HEAT-TREATMENT FURNACE.

The procedure in heat treatment was as follows: The furnace was ted to 880° C. and nine specimens introduced. This caused the tem- ature to fall to 550° C. and it took about 40 min. to again reach 880°. en the latter temperature was reached it was maintained constant 15 min., after which the specimens, except as noted below, were nched in water and a new lot introduced into the furnace. This continued until all but 18 of the specimens had been treated. Nine hese, consisting of three high-, three medium- and three low-sulphur ls, were removed from the furnace and allowed to cool in air. The

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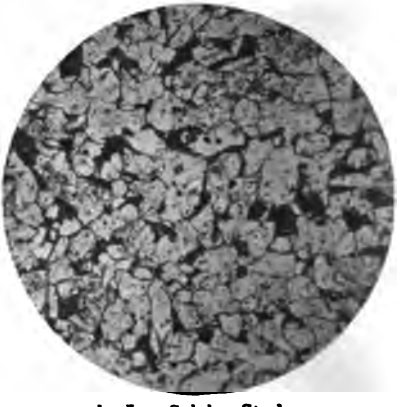
a.—High-Sulphur Steel.

b.—Low-Sulphur Steel.

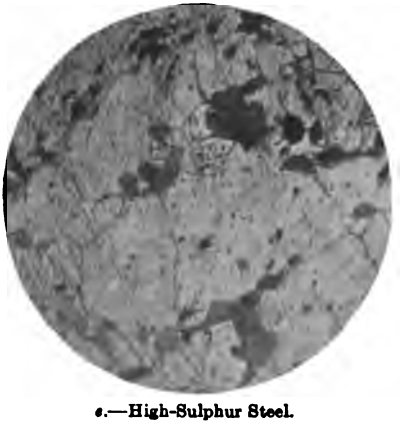
Quenched in Ice Water.

c.—High-Sulphur Steel.



d.—Low-Sulphur Steel.

Cooled in Air.



e.—High-Sulphur Steel.

f.—Low-Sulphur Steel.

Cooled in Furnace.

PLATE I.

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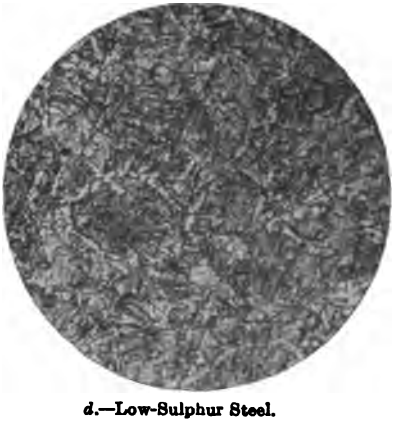
a.—High-Sulphur Steel.

b.—Low-Sulphur Steel.

Reheated to 300°.

c.—High-Sulphur Steel.



d.—Low-Sulphur Steel.

Reheated to 500°.

PLATE II.

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TABLE 2

No.	Mark		Yield Point, Lb. per Sq. In.	Breaking Load, Lb. per Sq. In.	Elongation, Per Cent. in 2 In.	Reduction of Area, Per Cent.
1	1A	Heated to 880° C. and quenched in ice water.	58,500	86,250	23.0	64.7
2	1A	Heated to 880° C. and quenched in ice water.	58,000	85,250	23.5	62.4
3	1A	Heated to 880° C. and quenched in ice water.	57,250	85,500	23.5	62.4
Average			57,900	85,600	23.3	63.1
4	2A	Heated to 880° C. and quenched in ice water.	53,000	79,750	26.0	69.3
5	2A	Heated to 880° C. and quenched in ice water.	56,250	77,000	29.5	71.4
6	2A	Heated to 880° C. and quenched in ice water.	51,500	76,250	27.5	69.3
Average			53,600	77,700	27.7	70.0
7	3A	Heated to 880° C. and quenched in ice water.	56,500	84,250	22.0	59.9
8	3A	Heated to 880° C. and quenched in ice water.	79,250*	89,250	22.5	59.9
9	3A	Heated to 880° C. and quenched in ice water.	57,300	84,500	22.0	62.4
Average			56,900	86,000	22.2	60.7
10	1A	Heated to 880° and cooled in still air	41,000	58,500	41.0	67.0
11	1A	Heated to 880° and cooled in still air	46,750	58,500	41.5	69.3
12	1A	Heated to 880° and cooled in still air	41,750	58,250	41.5	67.0
Average			43,150	58,400	41.3	67.8
13	2A	Heated to 880° and cooled in still air	45,750	58,000	41.0	67.0
14	2A	Heated to 880° and cooled in still air	46,250	58,000	40.5	69.3
15	2A	Heated to 880° and cooled in still air	39,750*	58,000	41.5	69.3
Average			46,000	58,000	41.3	68.5
16	3A	Heated to 880° and cooled in still air	40,000	59,500	38.0	62.4
17	3A	Heated to 880° and cooled in still air	41,500	59,500	39.5	64.7
18	3A	Heated to 880° and cooled in still air	46,250	59,250	39.5	64.7
Average			42,600	59,400	39.0	63.9
19	1A	Heated to 880° and cooled in furnace	38,000	52,500	40.5	62.4
20	1A	Heated to 880° and cooled in furnace	37,750	52,750	40.5	64.7
21	1A	Heated to 880° and cooled in furnace	33,750	52,500	40.0	62.4
Average			36,500	52,600	40.3	63.2
22	2A	Heated to 880° and cooled in furnace	37,000	53,000	40.0	62.4
23	2A	Heated to 880° and cooled in furnace	38,750	53,000	40.0	62.4
24	2A	Heated to 880° and cooled in furnace	34,500	53,000	41.0	59.9
Average			36,750	53,000	40.3	61.6
25	3A	Heated to 880° and cooled in furnace	37,500	55,000	37.5	59.9
26	3A	Heated to 880° and cooled in furnace	35,500	55,000	38.5	59.9
27	3A	Heated to 880° and cooled in furnace	32,250	55,000	38.0	69.3*
Average			35,080	55,000	38.0	63.0
301	1	Quenched at 880°. Reheated to 300°	53,750	76,000	25.5	69.3
302	1	Quenched at 880°. Reheated to 300°	58,000	76,000	26.0	69.3
303	1	Quenched at 880°. Reheated to 300°	56,000	75,000	28.0	71.4
Average			55,900	75,700	26.5	70.0
304	2	Quenched at 880°. Reheated to 300°	57,500	73,250	28.5	71.4
305	2	Quenched at 880°. Reheated to 300°	61,250	78,000	26.5	71.4
306	2	Quenched at 880°. Reheated to 300°	59,750	73,750	25.0	71.4
Average			59,500	75,000	26.7	71.4
307	3	Quenched at 880°. Reheated to 300°	57,000	80,250	23.5	62.4
308	3	Quenched at 880°. Reheated to 300°	56,500	83,000	21.5	59.9
309	3	Quenched at 880°. Reheated to 300°	59,000	85,000	19.5	59.9
Average			57,500	82,750	21.5	60.7
401	1	Quenched at 880°. Reheated to 400°	63,750	76,500	28.5	71.4
402	1	Quenched at 880°. Reheated to 400°	58,500	76,000	29.5	73.5
403	1	Quenched at 880°. Reheated to 400°	58,750	76,450	28.5	71.4
Average			60,300	76,300	28.8	72.1
404	2	Quenched at 880°. Reheated to 400°	62,750	75,000	26.5	69.3
405	2	Quenched at 880°. Reheated to 400°	63,750	80,000	24.0	69.3
406	2	Quenched at 880°. Reheated to 400°	61,250	74,750	28.5	71.4
Average			62,600	76,600	26.3	70.0
407	3	Quenched at 880°. Reheated to 400°	64,000	84,000	21.5	62.4
408	3	Quenched at 880°. Reheated to 400°	62,000	82,750	21.5	62.4
409	3	Quenched at 880°. Reheated to 400°	61,250	82,750	24.0	62.4
Average			62,400	83,150	22.3	62.4
501	1	Quenched at 880°. Reheated to 500°	55,250	70,750	33.5	73.5
502	1	Quenched at 880°. Reheated to 500°	58,250	73,250	31.5	73.5
503	1	Quenched at 880°. Reheated to 500°	57,500	74,000	31.0	73.5
Average			57,000	72,650	32.0	73.5
504	2	Quenched at 880°. Reheated to 500°	60,750	73,250	29.0	71.4
505	2	Quenched at 880°. Reheated to 500°	61,250	74,000	32.0	71.4
506	2	Quenched at 880°. Reheated to 500°	57,000	72,500	26.0	71.4
Average			59,650	73,250	29.0	71.4
507	3	Quenched at 880°. Reheated to 500°	61,500	78,750	26.0	67.0
508	3	Quenched at 880°. Reheated to 500°	60,500	77,500	26.0	67.0
509	3	Quenched at 880°. Reheated to 500°	60,750	77,750	26.0	67.0
Average			60,900	78,000	26.0	67.0
601	1	Quenched at 880°. Reheated to 600°	53,500	69,250	35.0	75.5
602	1	Quenched at 880°. Reheated to 600°	51,000	67,750	35.0	75.5
603	1	Quenched at 880°. Reheated to 600°	52,000	69,500	32.5	75.5
Average			52,150	68,850	34.2	75.5
604	2A	Quenched at 880°. Reheated to 600°	54,500	70,500	35.5	73.5
605	2A	Quenched at 880°. Reheated to 600°	52,500	72,000	32.5	73.5
606	2A	Quenched at 880°. Reheated to 600°	53,750	70,750	31.0	73.5
Average			53,550	71,100	33.0	73.5
607	3	Quenched at 880°. Reheated to 600°	55,000	74,250	31.0	71.4
608	3	Quenched at 880°. Reheated to 600°	55,000	73,750	30.5	71.4
609	3	Quenched at 880°. Reheated to 600°	55,500	75,750	30.0	69.3
Average			55,150	74,600	30.5	70.7

* Results abnormal. Not included in average.

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remaining nine, consisting of three of each sulphur content, were allowed to cool in the furnace.

In drawing the quenched specimens at the various temperatures, the procedure was as follows: The furnace was heated to 600° C. and nine specimens (three of each sulphur content) were introduced. The temperature fell and it required about 30 min. to come back to 600°. It was maintained here for 10 min. and then the specimens were withdrawn and quenched in water. The furnace was cooled to 500° C. and another set of nine specimens was introduced. When the temperature had regained 500° it was maintained constant for 10 min. and then the specimens were withdrawn and quenched. Following a similar procedure, sets of nine specimens were treated at 400° and 300° C. respectively.

Tensile Tests

After sawing off ¾ in. from each piece for microscopic examination, standard test specimens were prepared with 2-in. gage length, 0.505-in. diameter and threaded ends. These were pulled in an Olsen machine in the testing laboratory of the Massachusetts Institute of Technology.

The results obtained are given in Table 2.

For more ready comparison, the averages are retabulated in Table 3. The designations *L*, *M* and *H* refer to low, medium and high sulphur content.

TABLE 3

	Yield Point, Lb. per Sq. In.	Breaking Load, Lb. per Sq. In.	Elongation, Per Cent. in 2 In.	Reduction of Area, Per Cent.
L Heated to 860° C. and quenched in ice water	57,900	85,600	23.5	63.1
M Heated to 860° C. and quenched in ice water	53,600	77,700	27.7	70.0
H Heated to 860° C. and quenched in ice water	56,900	86,000	22.0	60.7
L Heated to 860° C. and cooled in still air	43,200	58,400	41.5	67.8
M Heated to 860° C. and cooled in still air	46,000	58,000	41.3	68.5
H Heated to 860° C. and cooled in still air	42,600	59,400	39.0	63.9
L Heated to 860° C. and cooled in furnace	36,500	52,600	40.5	63.2
M Heated to 860° C. and cooled in furnace	36,800	53,000	40.3	61.6
H Heated to 860° C. and cooled in furnace	35,100	55,000	38.0	63.0
L Quenched at 860° in ice water. Reheated to 300°	55,900	75,700	26.5	70.0
M Quenched at 860° in ice water. Reheated to 300°	59,500	75,000	26.7	71.4
H Quenched at 860° in ice water. Reheated to 300°	57,500	82,750	21.5	60.7
L Quenched at 860° in ice water. Reheated to 400°	60,300	76,300	28.8	72.1
M Quenched at 860° in ice water. Reheated to 400°	62,600	76,600	26.3	70.0
H Quenched at 860° in ice water. Reheated to 400°	62,400	83,200	22.3	62.4
L Quenched at 860° in ice water. Reheated to 500°	57,000	72,700	32.0	73.5
M Quenched at 860° in ice water. Reheated to 500°	59,700	73,300	29.0	71.8
H Quenched at 860° in ice water. Reheated to 500°	60,900	78,000	26.0	67.0
L Quenched at 860° in ice water. Reheated to 600°	52,200	68,800	34.2	75.5
M Quenched at 860° in ice water. Reheated to 600°	53,600	71,100	33.0	73.5
H Quenched at 860° in ice water. Reheated to 600°	55,200	74,600	30.5	70.7

There is not sufficient variation in the results to make an effective plot, but Table 4 summarizes the figures by giving the order in which they occur from high to low.

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TABLE 4

	Yield Point			Breaking Load			Elongation			Reduction of Area		
	L	*M*	*H*	*L*	*M*	*H*	*L*	*M*	*H*	*L*	*M*	*H*
Quenched in ice water..	1	3	2	2	3	1	3	1	2	3	1	2
Cooled in still air.....	2	1	3	2	3	1	1	2	3	2	1	3
Cooled in furnace	2	1	3	3	2	1	1	2	3	2	3	1
Reheated to 300°.....	3	1	2	2	3	1	3	1	2	3	1	2
Reheated to 400°.....	3	1	2	3	2	1	1	2	3	1	2	3
Reheated to 500°.....	3	2	1	3	2	1	1	2	3	1	2	3
Reheated to 600°.....	3	2	1	3	2	1	1	2	3	1	2	3
Totals.....	17	11	14	18	17	7	11	12	19	13	12	17
Order of totals.....	3	1	2	3	2	1	1	2	3	2	1	3

Photomicrographs

One specimen from each set was polished, etched in a solution of 10 per cent. HNO_3 in alcohol and examined under the microscope. The photographs show a representative spot on each specimen magnified 300 diameters.

The specimens were studied at higher magnifications than those given in the micrographs, but no further information was obtained. The lower magnifications gave a better idea of the general structure and, therefore, only these were reproduced in the paper.

Shock Tests

After concluding the tensile tests, it was thought desirable to determine the effect of varying sulphur content on specimens subjected to shock.

Through the kindness of the testing department of the Watertown Arsenal, the use of their Charpy machine was obtained for this purpose.

The remaining ¾-in. stock of bars marked 1, 2 and 3 was sawed into 2-in. lengths and the specimens subjected to heat treatment in sets of three under the same conditions that obtained in the case of the tensile specimens. After heat treatment, the specimens were machined to conform to the following specifications:

Length 55 mm., cross-section 10 by 10 mm., notch across one side, midway between the ends, 1 mm. wide. Radius of cutter edge, ⅔ mm.

The Charpy machine consists of a heavy pendulum which drops from a fixed height, strikes the specimen supported at each end and breaks

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TABLE 5

Treatment	No.	Mark	Breaking Shock, Ft.-Lb. per Sq. In.	Average
Heated to 480° and quenched in cold water	1	1A	494	
Heated to 480° and quenched in cold water	2	1A	545	
Heated to 480° and quenched in cold water	3	1A	540	526
Heated to 480° and quenched in cold water	4	2A	523	
Heated to 480° and quenched in cold water	5	2A	488	
Heated to 480° and quenched in cold water	6	2A	542	518
Heated to 480° and quenched in cold water	7	3A	432	
Heated to 480° and quenched in cold water	8	3A	507	
Heated to 480° and quenched in cold water	9	3A	470	470
Heated to 480° and cooled in still air	10	1A	356*	
Heated to 480° and cooled in still air	11	1A	460	
Heated to 480° and cooled in still air	12	1A	452	456
Heated to 480° and cooled in still air	13	2A	512	
Heated to 480° and cooled in still air	14	2A	521	
Heated to 480° and cooled in still air	15	2A	524	519
Heated to 480° and cooled in still air	16	3A	418	
Heated to 480° and cooled in still air	17	3A	398	
Heated to 480° and cooled in still air	18	3A	433	416
Heated to 480° and cooled in furnace	19	1A	258	
Heated to 480° and cooled in furnace	20	1A	284	
Heated to 480° and cooled in furnace	21	1A	295	279
Heated to 480° and cooled in furnace	22	2A	353	
Heated to 480° and cooled in furnace	23	2A	357	
Heated to 480° and cooled in furnace	24	2A	259*	355
Heated to 480° and cooled in furnace	25	3A	283	
Heated to 480° and cooled in furnace	26	3A	256	
Heated to 480° and cooled in furnace	27	3A	279	273
Quenched at 480°. Reheated to 300°	301	1A	633	
Quenched at 480°. Reheated to 300°	302	1A	447*	
Quenched at 480°. Reheated to 300°	303	1A	578	605
Quenched at 480°. Reheated to 300°	304	2A	562	
Quenched at 480°. Reheated to 300°	305	2A	537	
Quenched at 480°. Reheated to 300°	306	2A	530	543
Quenched at 480°. Reheated to 300°	307	3A	462	
Quenched at 480°. Reheated to 300°	308	3A	422	
Quenched at 480°. Reheated to 300°	309	3A	434	439
Quenched at 480°. Reheated to 400°	401	1A	613	
Quenched at 480°. Reheated to 400°	402	1A	597	
Quenched at 480°. Reheated to 400°	403	1A	582	597
Quenched at 480°. Reheated to 400°	404	2A	540	
Quenched at 480°. Reheated to 400°	405	2A	541	
Quenched at 480°. Reheated to 400°	406	2A	556	546
Quenched at 480°. Reheated to 400°	407	3A	474	
Quenched at 480°. Reheated to 400°	408	3A	447	
Quenched at 480°. Reheated to 400°	409	3A	429	450
Quenched at 480°. Reheated to 500°	501	1A	439*	
Quenched at 480°. Reheated to 500°	502	1A	694	
Quenched at 480°. Reheated to 500°	503	1A	670	682
Quenched at 480°. Reheated to 500°	504	2A	560	
Quenched at 480°. Reheated to 500°	505	2A	572	
Quenched at 480°. Reheated to 500°	506	2A	552	561
Quenched at 480°. Reheated to 500°	507	3A	495	
Quenched at 480°. Reheated to 500°	508	3A	456	
Quenched at 480°. Reheated to 500°	509	3A	469	473
Quenched at 480°. Reheated to 600°	601	1A	726	
Quenched at 480°. Reheated to 600°	602	1A	728	
Quenched at 480°. Reheated to 600°	603	1A	726	727
Quenched at 480°. Reheated to 600°	604	2A	584	
Quenched at 480°. Reheated to 600°	605	2A	604	
Quenched at 480°. Reheated to 600°	606	2A	602	597
Quenched at 480°. Reheated to 600°	494	3A	494	
Quenched at 480°. Reheated to 600°	502	3A	502	
Quenched at 480°. Reheated to 600°	521	3A	521	506

* Abnormal. Not included in average.

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it at the notch. The pendulum then continues its swing and the height it reaches is registered. Knowing the weight of the pendulum, the height it falls and the height it rises, a simple calculation gives the energy consumed in breaking the specimen.

The results obtained are given in Table 5.

Conclusions

Table 4, which expresses the summary of the tensile tests, shows that the high-sulphur steel has for each treatment the highest breaking load while the yield point ranks first for two treatments, second for three and third for two. From this we may conclude that the sulphur does not lower the tensile strength.

The figures for elongation and reduction of area show that there is little difference in ductility between the low- and medium-sulphur steels, but the ductility of the high-sulphur steel is slightly lower than the other two for most of the treatments.

The average figures for the shock tests, except for the air- and furnace-cooled specimens, are highest for each treatment in the case of the low-sulphur steels and lowest for each treatment for the high-sulphur steels. The widest difference appears in the steels which have been quenched and reheated.

It is difficult to draw definite conclusions from the results because of the newness of the shock test and the difference of opinion among engineers regarding its value. The tensile tests are not unfavorable to steels with moderate amounts of sulphur, while the shock tests show a decided falling off in strength as the sulphur increases. Until the interpretation of the results from the Charpy machine is more fully understood, it is impossible to say to which set of tests the most importance should be attached.

Further light might be thrown on the subject by making alternate stress or fatigue tests. It would be important to learn whether the results would confirm the tensile or shock tests. Unfortunately, however, the stock of steels used in the previous work was exhausted, and whatever the results of the fatigue tests there would be an uncertainty in their interpretation because of difference in stock. It was, therefore, decided not to include this series in the present investigation.

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[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Recrystallization after Plastic Deformation

BY HENRY M. HOWE, LL. D.,* BEDFORD HILLS, N. Y.

(New York Meeting, February, 1917)

THIS paper is a discussion of the extremely valuable one of Mathewson and Phillips, The Recrystallization of Cold-Worked Alpha Brass on Annealing,[1] which not only gives us a wealth of important data reached with great intelligence, but also shows both uncommon powers of imagination and a perfectly fair spirit.

Let me try to analyze some of their results, and to enunciate some of the laws to which these results point. For brevity I refer to them as "The Authors." The page numbers in parentheses refer to their paper.

1. *The Visible Aspects of Recrystallization.*—Plastic deformation in a single direction, as in cold-rolling, draws the grains out in that direction, that is, inequiaxes them, without destroying their apparent individuality. Each grain seems to endure the drawing out without changing either its volume or its orientation materially, for the etching tint of each grain seems to be as uniform after as before the drawing out, save for the "etch bands,"[2] shown in Figs. *A* to *D* of the Authors' Plate I.

As the drawn-out metal is heated progressively, on reaching a certain "disintegration temperature" these old grains seem to break up, probably into submicroscopic fragments, and this apparent disintegration is followed by a coalescence of these fragments, so that their number decreases and their size increases, till they may become very much coarser than before the drawing out. Ruder[3] coarsened the grains of silicon steel in this way till single grains reached an area of 50 cm. (7.75 sq. in.).

The deformation itself probably breaks each grain up into many very small fragments, but the fact that each grain continues to have a uniform etching tint different from that of its neighbors indicates that these fragments retain the initial common orientation of the grain. The apparent disintegration which we observe, then, is rather an abandonment of community of orientation for individuality of orientation, by the fragments formed earlier during the deformation. Hence we

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[1] *Bulletin* No. 109, p. 1 (January, 1916).

[2] Rosenhain calls these "slip bands," and I have called them "X bands," to avoid any preconception as to their nature (*The Metallography of Steel and Cast Iron*, p. 452).

[3] *Trans.*, vol. 47 p. 569.

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should speak rather of a "reorientation" than of a "disintegration" temperature. The reorientation and regrowth may jointly be called "recrystallization."

The recrystallization is accompanied by softening, that is to say, though the first raising of the temperature after the cold-rolling may increase the hardening which the cold-rolling caused, yet with further rise of temperature the hardness falls progressively to that of the normal metal.

While the Authors would explain the nature of disintegration and regrowth, their main objective seems to be to trace the connection between these two processes and this softening.

2. *Our natural conception of disintegration and grain growth* I take to be that the plastic deformation itself breaks each grain up into many fragments as already pointed out, of which the number increases and the size decreases as the deformation increases; that each fragment retains for the time the initial orientation of the grain from which it is broken, as is shown by the uniform etching tint of all these fragments of a given grain; but that each is sheathed completely with metal made amorphous by the friction during the act of deformation. This amorphous metal cements the whole together. On this hypothesis when the deformed metal is heated, five distinct changes occur side by side, though not necessarily simultaneously or in any fixed proportion to each other. These are, first, a progressive reabsorption of the amorphous sheaths by the crystalline fragments which they inclose; second, a suspected progressive but undescribed change in the condition of so much of these sheaths as has not yet become reabsorbed, or of the crystalline grain fragments thus sheathed, or of both; third, an apparent disintegration of the initial grains, which is really only a reorientation of the grain fragments already formed; fourth, their progressive growth by coalescence, with consequent lessening of their number; and fifth, the progressive release of stress as the mobility increases and the existing elastic limit falls correspondingly. The first three may be grouped together as "recrystallization." Let us in (3) to (7) glance at the evidence that these five agencies are at work.

3. *The reabsorption of the sheaths* with rising temperature is inferred from the observation that, by heating before etching, the etch bands, which seems to represent an accumulation of this same amorphous metal along the slipping planes, are made first fainter and finally undetectable.

4. *The reorientation of the grain fragments* is shown by the apparent disintegration which the grains undergo, for this disintegration means only that the several fragments of a given grain cease to have common orientation and hence to etch alike, and instead adopt each a new orientation, so that they etch differently and hence appear as distinct grains or grain fragments. It is not clear that this distinction between grains

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and grain fragments need be insisted upon in what follows, for indeed each fragment seems for all intents and purposes to be an independent grain, once its reorientation has occurred.

5. *Grain Growth.*—The progressive growth of the grain fragments is familiar to all through the coarsening that occurs on heating cold-rolled or otherwise plastically deformed metal.

6. *The progressive release of stress* could no doubt be proved by noting how much, after reheating to various temperatures, the specimen bends out of shape spontaneously on cutting away a fixed part of it, so as to destroy the balance of internal stresses remaining after the initial cold-rolling itself. We may assume confidently that the degree of this spontaneous bending decreases progressively as the temperature of annealing, prior to cutting away, rises.

7. *A progressive change in the condition of the amorphous sheaths* or of the crystalline fragments is neither shown nor known. I have inferred it as a probable cause of the perturbations which the cohesion, as indicated by the elastic limit and hardness of cold-rolled iron, both pure and cementite-bearing (mild steel), undergoes on gentle heating. On either slight heating or mere rest such iron may soften materially or may harden greatly, according to the intensity of the previous deformation.[4]

8. *In what Proportion is the Softening Caused by Annealing due to Each of These Five Influences?*—Though the present evidence does not enable us to give a full answer to this question, some inferences concerning it may be drawn. Let us see, in (9) to (14), what importance we are inclined to assign each of these five supposed agencies.

9. *Stress.*—The relief of stress is not likely to cause a very large fraction of the softening induced by annealing; first because it is incompetent to explain the great hardening and embrittlement which gentle heating causes in cold-worked iron; and second because the progress of the soften-

[4] *The Metallography of Steel and Cast Iron*, pp. 365 and 452 to 458. The Authors' explanation (*Bulletin* No. 109, p. 7 (January, 1916)) that the strengthening represents the removal of stress is not competent to explain either the tripling of the hardness which, as I find, mere rest after deformation may cause, or the simultaneous great loss of ductility. Thus Stromeyer (*Journal of the Iron and Steel Institute*, vol. 75, No. 3, p. 92 (1907)) finds that the ability of the outer surface of a nicked test-piece to elongate before rupture may fall from 60 per cent. to 6 per cent. on long rest, and in one case to 6 and in another even to 3 per cent. on heating to 100° for 15 min. Nor can we accept the explanation (Gulliver, *Journal of the Institute of Metals*, No. 2, p. 220 (1909)) that this hardening, strengthening, and embrittlement represent "a change from the instable Fe_3C, present in commercial steels, to the stable Fe_2C," because I find that practically carbonless steel, with only 0.01 per cent. of carbon, hardens like and perhaps even more than common mild steel of 0.147 per cent. of carbon, on heating to 100° after plastic deformation (*op. cit.*, p. 365). This strengthening and hardening which slight heating causes so markedly in iron occur only in a slight degree in brass, as is shown concordantly by the results of Bengough and the Authors (pp. 3 and 6). Indeed it is so slight as to have been overlooked by Grard (p. 3).

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ing, both absolute and relative, in the Authors' results is not such as would be expected to be caused by stress-relief. Thus, on the theory that stress-relief is the chief cause of the softening, it is hard to see why a progressive increase in the degree of cold-rolling should lead either to the progressive and marked lowering of the temperature of most rapid softening, or to the marked narrowing of the temperature range in which most of this softening occurs, as recorded by the Authors (Fig. 3), or why, after 8 per cent. reduction by rolling, the softening should be only 1.4 units between 350° and 450°, yet should rise to 2.1 units between 450° and 550° and to 2.5 between 550° and 650° (Table IV).

10. *Reorientation or Disintegration.*—If we are right in conceiving that visible disintegration represents the breaking away from the initial orientation and substituting a new one in each of the several grain fragments which thereby become visible, then before any large fraction of the fragments have thus changed, the change should become visible, substituting in the earliest of the reorienting grains finer uniformly etching areas for the initial coarser ones. On this conception, disintegration is not likely to be the chief cause of the softening, because after the Authors' 4 and 12 per cent. reductions more than one-third of the total softening occurs from 100° to 200° before the reorientation becomes visible (Table IV).

Again, it is hard to reconcile the belief that most of the softening is due to reorientation with the fact that, after their 35 per cent. reduction, 80 per cent. of the total softening occurs after reorientation has become visible, and successive softenings each of about 10 per cent. of the total are found at temperatures, 200°, 300°, and even 400° higher, though the micrographs suggest no such temperature-distribution of the reorientation.

11. *Grain Growth.*—If we conceive, as seems reasonable, that the grain fragments cannot coalesce with each other to any important degree before they reorient themselves and thereby cause visible disintegration, then the great softening which, after the 4 per cent. and 8 per cent. reductions, precedes visible reorientation, cannot be referred to grain growth. This disinclines us to refer the softening as a whole chiefly to grain growth, in spite of the very great softening that accompanies the replacement of finer by coarser-grained gold in the important experiments of Fahrenwald,[5] a softening so great that we hesitate to refer it solely to difference in grain size. The Authors clearly share this belief (p. 31).

Another indication that grain growth is not the chief cause of the softening is the discordance between the curves of grain size and of hardness in Fig. 4. To judge from Plate XI there is a great retardation of grain growth from the 350°–450° to the 450°–550° range, and Fig. 4

[5] *Bulletin*, No. 109, p. 122 (January, 1916).

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tells us that there is such a retardation from the 450°–550° to the 550°–650° range, yet in passing these ranges the softening undergoes no corresponding retardation, nor does the tensile weakening.

12. *Reabsorption of the Amorphous Sheaths.*—Thin as these sheaths seem to be, if we refer to them the great hardening that cold work is capable of giving we must infer that their total volume forms a very appreciable fraction of the whole mass, an inference which is the more probable in view of the marked changes which cold work causes in the solution pressure, density, and other physical properties of the mass as a whole.

It is reasonable to hold that this amorphous metal is harder than the crystalline, because it lacks the cleavages and partings which enfeeble every crystalline mass. Hence its progressive reabsorption on heating and rest would itself naturally cause a corresponding softening of the mass as a whole. It may be that the rate of reabsorption of this amorphous metal on heating and rest varies in such a way as to contribute greatly to the variations observed in the rate of softening of the whole mass.

13. *Change of Condition of the Amorphous Sheaths or of the Crystalline Fragments which they Inclose.*—The great hardening, strengthening, and embrittlement which pure iron undergoes on rest or on slight heating after cold deformation are not readily referred to any of the four agencies which we have just considered in (9) to (12). In order to explain them I attempt to refer them to unknown changes in the nature of either the amorphous sheaths or crystalline fragments which they inclose, or both. If we grant that either or both these agencies are capable of causing these very striking changes, we naturally infer that they may also contribute to a very important degree to the softening which follows this hardening in iron on higher heating, and occurs without prominent hardening in the other metals.

14. *To sum up*, stress-relief and grain growth are not likely to contribute greatly to the softening caused by heating overstrained metal. Reorientation may contribute greatly, but it is not likely to be the chief cause. The reabsorption of the amorphous sheaths inclosing the grain fragments, and a progressive change in the condition of those sheaths of the crystalline metal in those fragments, are possible important causes. Beyond this preliminary speculation the present data do not justify our going.

It is probable that at least two agencies are at work, with effects which differ greatly, because, in the case of iron, rest and gentle heating, while effacing one of the effects of the cold work, the etch bands, simultaneously increase another and simultaneous effect of that same cold work, the hardening which it causes.

15. *Correspondence between the Authors' Conjectured Progress of*

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Recrystallization and Their Observed Progress of Softening.—First we must understand the Authors' term "fragmental resolution," for which I venture to suggest "limiting grain size." They hold that, for each temperature, there is a certain " limiting grain size," "limiting" in the sense that all grain or grain fragments which are smaller than it start to grow toward it, but cannot grow beyond it, nor can grains of this or any greater size grow at this temperature. As far as I understand "fragmental resolution," it means this "limiting grain size," a

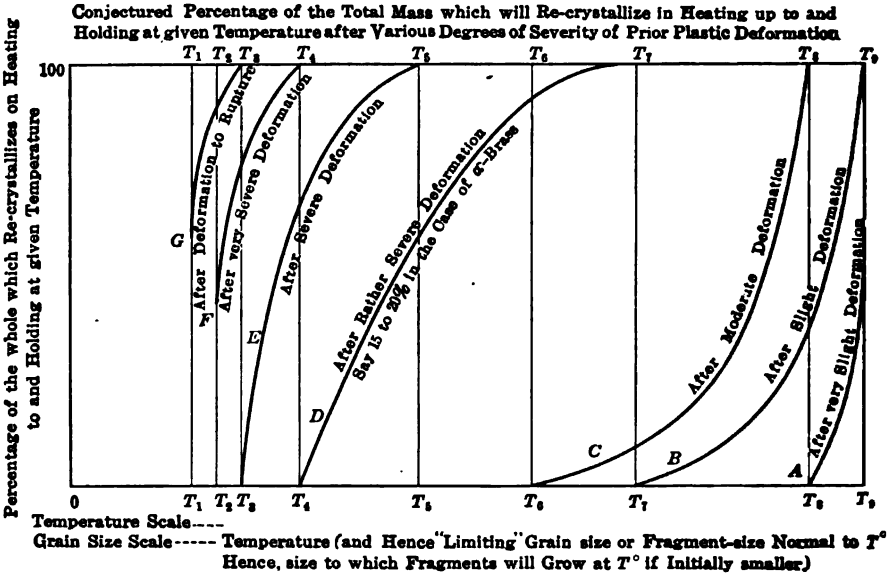

FIG. *A*.—CONJECTURED PROGRESS OF RECRYSTALLIZATION OF COLD-WORKED BRASS ON PROGRESSIVE HEATING.

term which I offer reluctantly and only because I find "fragmental resolution" confusing.

The Authors give in their Fig. 2 a most interesting generalization from their hypothesis. This, their conjecture as to the progress of recrystallization, that is of the return to the crystalline state of the metal made amorphous by the deformation, may perhaps be grasped more easily as relettered in my Fig. *A*. Here ordinates show what percentage of that part of the metal which has thus been made amorphous can recrystallize in rising to given temperature and staying there 30 min. Abscissæ measure both temperature and the limiting grain size for given temperature. Far from attempting quantitative accuracy, this sketch tries only to show the general family to which the curves are expected to belong.

According to this forecast, the greater the previous reduction by cold-rolling has been, (1) the lower is the temperature at which the curves leave the horizontal axis, that is the temperature at which recrystallization

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starts, and (2) the steeper is the beginning of the curve. Further (3), whereas after severe deformation the curves are convex upward, after slight deformation they are concave upward. Hence the curves for great reductions form a sheaf of which the top fans out to the right, whereas those of slight deformation form a sheaf of which the bottom fans out to the left.

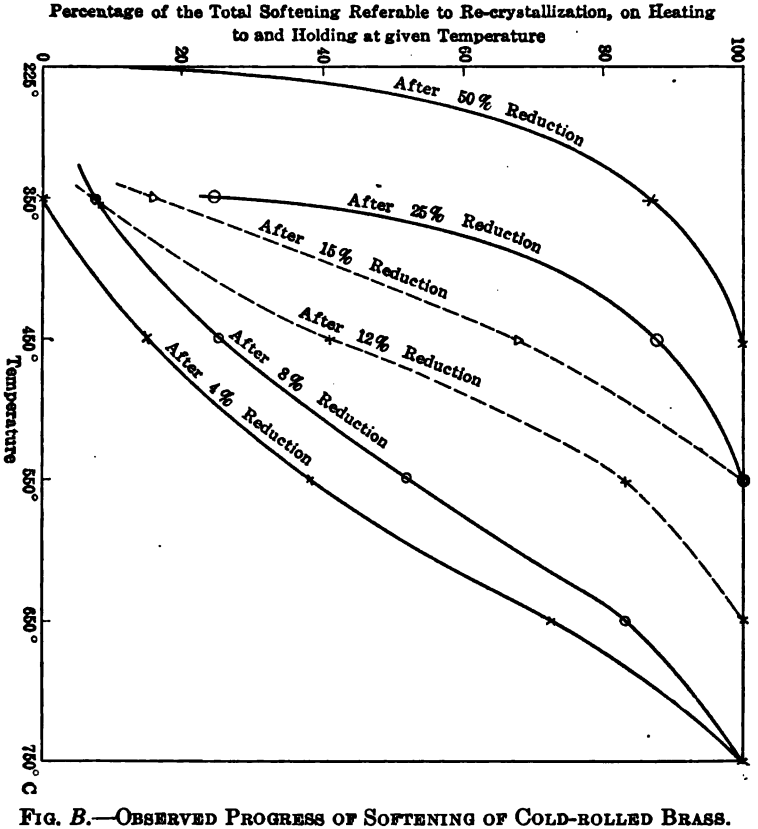

FIG. *B*.—OBSERVED PROGRESS OF SOFTENING OF COLD-ROLLED BRASS.

We have no data showing the true progress of this recrystallization, and hence we cannot check their forecast as to the relation of this progress to the temperature of annealing. But in my Fig. *B*, I attempt to check it indirectly by showing how the progress of that fraction of the softening which may be referred to recrystallization is related to the annealing temperature. Any such attempt must be very rough, because we do not know what fraction of the softening is due to recrystallization and what to grain growth. In view of the relatively small effect of grain size as such on hardness, I assume that the greater part of the softening is due to recrystallization, and for the purposes of this rough check I assume

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that the temperatures given in line 2 of my Table *A* are those at which recrystallization becomes complete, on the ground that these are the ones at which the softening is nearly complete.

Because the heatings underlying their Table IV were of 30 min., because a temperature of 255° was needed in their Table II to cause an appreciable softening after 40 per cent. reduction in about this time; and because 30 min. at 200° caused a slight hardening after this reduction; I set $T°_1$, the temperature needed to cause the first softening after 50 per cent. reduction, at 255°.

The curves of Fig. *B*, plotted on these assumptions, tend to support the second and third of the Authors' predictions, that the steepness of the beginning of the curve increases with the previous reduction, and that the curves for great reduction are convex upward whereas those for light reductions are concave upward.

Their first contention, that the temperature at which recrystallization starts is the lower the greater the deformation has been, has such good direct experimental support that it is to be accepted in spite of its not being clearly indicated by these curves. Note, for instance, that the shape of the 8 and 12 per cent. reduction curves suggests that they leave the horizontal axis at temperatures rather below than above those of the 15 and 25 per cent. curves.

16. *Is There an Equilibrium Grain Size for Each Temperature?*—The Authors' assertion to this effect (p. 18) may well be taken rather cautiously in our present ignorance as to the mechanism of grain growth. That this growth accelerates with rise of temperature, and retards itself as it proceeds, seems clear; but that it ceases completely, so that the agencies resisting coalescence come in time to arrest it completely, at relatively high temperatures, has not yet been shown, I believe. A marked retardation as the process proceeds would naturally occur on almost any theory of grain growth, but that need not imply arrest. If it is true as reported that meteorites often consist of a single enormous grain, that would indicate that, at least in this case, if time is almost unlimited coalescence becomes almost unlimited, and that here the anti-coalescence agencies never become strong enough to cause complete arrest.

17. *Certain Laws of Annealing for Alpha Brass.*—The Authors' data point to the following laws:

(*A*) As the degree of previous deformation (cold-rolling) increases from a reduction in thickness of 4 per cent. to one of 40 or 50 per cent.,[6] (1) the temperature at which in subsequent heating reorientation becomes visible under a magnification of 85 diameters falls progressively from 650° to between 275° and 300° (Table III); (2) the temperature of most

[6] Both the 40 per cent. maximum reduction of Table III and the 50 per cent. reduction of Table IV and Plate XI are correct.

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rapid softening falls progressively from between 550° and 650° to below 350° (Table IV); and (3) the range of temperature in which the greater part of the softening occurs becomes progressively narrower (Table IV).[7]

(*B*) From this it follows that the annealing temperature needed to cause nearly complete softening is the lower the greater the previous reduction has been, and the longer the annealing period. For a 30-min. annealing the relation is about as follows:

TABLE *A*

1. Percentage of reduction by cold-rolling.....	4	8	12	15	25	50
2. Temperature needed for nearly complete softening, degrees Centigrade.....	750	750	650	550	550	450
3. Percentage of total softening at that temperature.....	90.4	98.8	90.0	90.6	79.6	79.2
4. Lowest temperature of visible disintegration, degrees Centigrade.....	650	550	550	450	350	350

(*C*) The temperature-range of most rapid softening merges into a higher range of progressively retarded further softening, which continues in every case even to beyond 750°, suggesting that the softening is due to at least two superposed changes, of which one, probably recrystallization, occurs rapidly at a relatively low temperature—the lower and narrower the greater has been the cold-rolling—and the other, probably grain growth, is spread out.

(*D*) *Decrease on Heating of the Excess of Hardness Caused by Greater Deformation.*—Here we may leave out of account the 2 per cent. reduction, because its effects are so slight compared with the accuracy of the Shore method, and the 50 per cent. reduction because this refers to a different alloy.

The excess of hardness caused by 25 per cent. reduction over that caused by 4 per cent., which is 14.7 Shore units before annealing, falls to 10.9 units on annealing at 350° and to 1.7 units on annealing at 450°. We may question whether this excess remains measurably great after annealing at 550° or any higher temperature, for though, if we consider the extreme cases, the 4 and the 25 per cent. reductions, a slight excess persists even on heating to 800°, as shown in line 4 of Table *B*, yet for

[7] This agrees with Muir's observation that the greater the hardening caused by overstrain, as shown by the rise of the yield point in steel, the lower is the temperature at which softening sets in (*Philosophical Transactions of the Royal Society of London*, vol. 193 A, p. 22, 1900), and with Chappell's that the greater the deformation the lower the temperature at which the recrystallization (disintegration and subsequent coarsening) of overstrained steel occurs (*Journal of the Iron and Steel Institute*, vol. 89, pp. 471, 496, 1914).

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TABLE B

1. Annealing temperature, degrees C....	Unannealed	350	450	550	650	750	800
2. Hardness after 25 per cent. reduction..	27.4	23.6	13.7	11.8	10.2	8.2	7.8
3. Hardness after 4 per cent. reduction...	12.7	12.7	12.0	10.9	9.3	8.0	7.5
4. Difference, Shore units[8].....	14.7	10.9	1.7	0.9	0.9	0.2	0.3

temperatures above 450° this excess hardly exceeds the limits of error. Moreover, if we compare all five of the reductions, we find great anomalies. For instance, the specimens reduced by 4 per cent. are actually reported as harder than those reduced by either 8 or 12 per cent., after the 650°, 750°, and 800° annealings severally.

Again, for each of the four annealing temperatures, 550°, 650°, 750°, and 800°, there are five reductions that we are considering, those of 4, 8, 12, 15 and 25 per cent. If for each temperature we compare the hardness after any one reduction with that after the next greater reduction, we have four such comparisons at each temperature, or 16 for the four temperatures. Though in eight of these 16 cases the hardness given by the greater reduction exceeds that given by the less, yet in seven cases the reverse is true, and in the remaining case the greater and the less reduction give the same hardness.

(*E*) *The Temperature at which Disintegration First Becomes Visible may Suffice to Efface the Excess of Hardness Given by Greater Reduction.*— This refers to the temperature at which the reorientation becomes visible under a magnification of 85 diameters. For instance, this temperature is 550° for the reductions of 8 and of 12 per cent., and at this temperature the residual hardness is actually greater after the 8 than after the 12 per cent. reduction, though the difference hardly exceeds the limits of observational error.

(*F*) As the temperature rises, the softening accelerates greatly and continuously up to a maximum rate. For instance, a softening amounting to 3 Shore units is made 6.4 times as rapid by a 50° rise, from 225° to 275°, and about 1,000 times as rapid by a 100° rise from 225° to 325° (Table II, p. 7).

[8] By a "Shore unit" is meant a difference of 1 in the unit place of the Shore scale, hardness 14.3 being 2 units greater than 12.3.

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Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. HOWLAND, printed in *Bulletin* No. 111, March, 1916, pp. 627 to 650.

W. H. BLAUVELT, Syracuse, N. Y. (communication to the Secretary*).—I have read Mr. Howland's paper on Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces with much interest, and believe that his method of analysis of the subject will be of material assistance in our study of the action of coke in the blast furnace.

Mr. Howland quotes Professor Richards' statement that from the standpoint of the generation of the maximum quantity of heat in the furnace, Grüner was right in saying that all the carbon burned in the furnace should be first oxidized at the tuyères to CO and all reduction of oxides above the tuyères should be caused by CO, which thus becomes CO_2. It is clear that the maximum quantity of heat is generated in the furnace when the greatest amount of carbon is burned to CO_2 irrespective of whether it is burned at the tuyères or further up the furnace. So it would appear that Grüner's theory is sound regarding the maximum production of heat only because if all the carbon is burned to CO at the tuyères the reduction of the ores will have to be effected by the further oxidation to CO_2. Whereas, if the reduction of the ores is effected in part by solid carbon, part of this carbon might be oxidized only to CO, resulting in a higher percentage of CO in the escaping gases than from Grüner's "ideal conditions." But in studying the combustion of coke in the furnace, it is clear that the production of the maximum quantity of heat is not of the first importance in blast-furnace operation, or in the utilization of the fuel charged into the furnace. To my mind the production of a high thermal head at the tuyères is of the first importance, and the best coke is that which reaches the tuyères in proper condition to produce the highest temperature at the tuyères, and in just sufficient quantity to do the amount of work required there under the conditions produced by this maximum temperature. The combustion of a much larger amount of fuel at the tuyères, under conditions that will fall short of producing the highest possible temperature, cannot produce as good results, either in fuel economy or output. If this statement is correct, does it not correspond with Mr. Howland's statement that too much wind is inimical to low coke consumption? Nothing is more fatal to obtaining the highest temperatures than an excess of air for combustion. In the blast furnace an excess of air dilutes and cools the products of combus-

* Received June 30, 1916.

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tion, reducing the maximum thermal head at the tuyères, and the larger volume carries the high-temperature zone too high in the furnace.

There has been a great deal of discussion as to what are the qualities which make the best furnace coke. The wider use of the byproduct oven has brought the control of the coke closer to the furnace man, and makes it possible to modify its structure in many ways, so that the question "what are the best specifications?" has become a very live one. It will probably be generally admitted that furnace coke should be of nearly uniform size, and many furnace managers are eliminating all coke below ¾ in. and above 4 or 4½ in. Also that the best coke is that which is sufficiently strong to resist undue abrasion and crumbling by attrition with the stock, and of an open porous structure that will permit the most rapid combustion when it reaches the tuyères. Many large users agree that the coke should never be overcoked, beyond the point of producing a sufficiently strong structure, as overcoking quickly reduces the combustibility.

If Grüner's ideal gives the best furnace operation, we should want a coke that is resistant to the oxygen in the ore but easily combustible at the tuyères, which is a contradiction of qualities. If my argument is correct, that the furnace man wants the greatest thermal head at the tuyères rather than the production of the greatest quantity of heat in the furnace as a whole, then he is willing to sacrifice some coke by solution in the oxidizing gases in the upper part of the furnace, provided he can obtain a sufficient quantity of coke at the tuyères, of a quality that will permit rapid combustion with the minimum amount of air, thereby giving him the maximum thermal head.

The Rifling of Diamond-Drill Cores

Discussion of the paper of WALTER R. CRANE, printed in *Bulletin* No. 113, May, 1916, pp. 823 to 833.

H. M. ROBERTS, Minneapolis, Minn. (communication to the Secretary*).

—The rifling of drill cores is a frequent source of interest to men engaged in diamond drilling. Previous to the appearance of Dr. Crane's paper, there has been little record of connected observation or rigid speculation as applied to the cause of rifling. Replying to one of Dr. Crane's letters of inquiry, I ventured my opinion that the cause of rifling was a complex problem in physics, which I had not attempted to solve up to that time, except to ascribe it to the general cause of rotation and vibration of the rods. The question is worth attention not only as a matter of curious scientific interest but also for the reason that it stimulates thought on

* Received June 24, 1916.

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the mechanical action which takes place at the end of a diamond bit. In discussing Dr. Crane's paper, I still recognize the complexity of the problem and merely note a few observations of fact with the inferences which may be drawn from them. I am indebted for suggestions from W. J. Mead, F. F. Fredlund and James W. Hunter among others on the staff of the E. J. Longyear Co.

It seems clear that the cutting medium which causes uniform threading must be the diamond bit itself. The deep, regular rifling of such dense rocks as granite and norite for long intervals admits of no other reasonable assumption.

Rifling occurs in many different rocks, both hard and soft, but in all instances noted there is one feature in common: the rock is homogeneous over the extent of the rifling.

Wall of Drill Hole Rifled

The wall of the drill hole itself is rifled, as well as the core. This has been observed in the shaft of the Isabella Mine in Northern Michigan, which was sunk on a 2-in. hole in diabase.

FIG. 1.—SPECIMEN FROM SHAFT OF ISABELLA MINE, CASCADE RANGE, MICH., SHOWING RIFLING ON THE WALL OF AN "N" DRILL HOLE.

Fig. 1 shows a specimen taken from the walls of this hole. It is reasonable to suppose, therefore, that rifling of the core is also accompanied in most instances by rifling of the wall of the hole. This leads to the inference that perhaps the immediate end of the bit is responsible.

The Diamond Bit

A consideration of the great pressures which bear upon a bit during the drilling makes it quite certain that all the stones work and engage the rock together. Examination of a worn bit shows that all the stones

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have played their part. In this connection it is of interest to note that while diamond setters place outside stones very accurately to gage, they seldom use a gage in setting inside stones, which perhaps admits of one stone working alone on the inside for some interval of time. Those portions of the stones which project from the immediate interior and exterior edges of the face of the bit are no doubt responsible for the threading. It is apparent that any play in the bit would act with greatest effect at the extreme edges and would permit of greater penetration than the overset of that portion of the stone which extends up from the edge on the inside and outside of the bit. During the drilling operation the protuberance of the stones at the edges is much greater than when first set up, owing to the wearing away of the metal. Fig. 2 shows two worn diamond bits which have produced rifled core. There is no evidence to

FIG. 2.—WORN DIAMOND BITS WHICH HAVE PRODUCED RIFLED CORE AND A SPECIMEN OF QUARTZITE CORE SHOWING LEFT-HAND THREADS. THIS CORE WAS PRODUCED BY A BIT REVOLVING TO THE RIGHT.

show that a bit which has produced rifling was different in any material respect when first introduced to the hole than a bit which has produced smooth core. There is decided evidence to the effect that carbon wear and loss of metal are increased in any bit which has produced rifling. Thus the cause of rifling must lie in some force which acted upon the bit during the time when the rifled core was produced and which did not act during the time when the smooth core was produced.

Different Kinds of Threads and Varying Conditions

The depth of the hole does not seem to be a governing factor. Deeply rifled cores have been found in norite drilled with a 2-in. bit 10 ft. from the machine. Threads of different size, different pitch, and of varying numbers to the turn are to be found on cores of the same material from one hole drilled with the same bit and with the machine making the same

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number of revolutions per inch of advance in the bit. The threads are both left hand and right hand, although as far as my observation goes right-hand threads predominate. Fig. 1, showing the rifled wall of a drill hole, reveals both left-hand and right-hand threads in the same specimen. The bit producing these threads revolved to the right. I have counted both even and odd numbers of threads, varying from three to twenty, after which they become too fine to count; in fact, a smooth core always shows small variations from a true cylindrical shape.

Vibration of Rods Prevalent When Rifled Core is Produced

It is an observed fact that vibration of the rods is prevalent in drilling formations where regular rifling occurs. This obvious relation led to the replies which drill operators gave to Dr. Crane's question. The production of threads on a vibrating pipe in a lathe bears a distinct analogy and renders the relation between the vibration of the drill rods and rifling of the core quite certain. Let us examine this relation further, since we are pursuing the inquiry largely as a matter of pure interest.

Harmonic Motion of the Bit

The presence of threads over any interval of core indicates a constant play to and fro of the bit relative to the axis of the hole during its advance. Uniformity of the threading over any interval of core indicates that the swing of the bit operates according to some law. The fact that the number of threads and their character change from interval to interval, indicates that the factors causing the swing of the bit vary in considerable degree. We are evidently dealing with some type of harmonic motion. Possibly the homogeneous character of the rock found in most rifled cores is one of the principal factors which permits of harmonic motion in the bit. In considering the instance where many threads of low pitch appear on a cross-section of core, it is possible at first sight to conceive that the threads are due to spiraling which is directly proportional to the advance of the bit, but in an instance where corrugations of steep pitch are produced, as No. 8, Fig. 1, of Dr. Crane's paper, with perhaps 600 revolutions of the bit to an advance of 1 in., it is evident that the process is extremely complicated. The production of left-hand core by a bit revolving to the right is also a difficult matter to explain (see Fig. 2).

Consider the forces that operate on a flexible line of drill rods which are revolving rapidly. First, compression; second, torque. The play of these two forces sets up waves which have their lengths parallel but twisted with respect to the drill rods and with their crests and troughs at right angles to the axis of the hole. The twist in the rods is perhaps

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not as great as might be supposed, for when a bit is blocked under great pressure, and the engine is stopped and released, the chuck seldom revolves back more than a turn or a turn and a half. During this process the bit is presumably fast on bottom.

As the rods revolve, transverse waves advance down the rods at every azimuth, like waves of light. That waves of this type are produced is shown by the whipping out of uncased holes when drilling soft iron formation. The samples are often vitiated in this manner. Anyone who will climb the tripod and look down on the water-swivel when the rods are vibrating will see there a distorted but nevertheless fairly true picture of what is taking place at the bit on the other end of the rods. It is possible to count distinct beats in the play of the swivel to and fro as the rods vibrate.

The regularity of the grooving in the threaded core indicates that the waves producing it have a definite time interval with a fixed relation to the revolution of the rod. The fact that all the stones are working at the same time does not present any particular difficulty when it is remembered that in any one instance of threading the stones are always at fixed intervals with respect to the axis of the bit and therefore have a fixed relation to any wave that affects the bit and a fixed relation to the period of revolution. In the instance cited by Dr. Crane where pentagonal core is formed, the time of vibration must be nearly a definite divisor of the time of revolution. That is, during one revolution of the bit there were approximately five major wave motions which produced a portion of five threads on the core for a longitudinal distance of less than $\frac{1}{100}$ in. In all other instances where the pitch is flatter the ratio must be more intricate. When right-handed threads are produced, the period of vibration is incommensurable and in this instance thereby deferred slightly each time with respect to the core as the rod revolves. When left-handed threads are produced, the period of vibration is also incommensurable, but in this case occurs just an instant earlier with respect to the core during each succeeding revolution.

Overtones

In wave motions of this kind it is conceivable that there are overtones; that is, if the period of one wave length be T, then there are wave lengths of $\frac{1}{2}\,T$, $\frac{1}{3}\,T$, etc. The braided threads on many pieces of core are suggestive of secondary action of this kind (see Fig. 2 of Dr. Crane's paper). All of the laws of resonance and interference in wave motion would enter into the amount of play in the bit as it revolves. Again, when relatively smooth core is produced, it is probably the result of the impact of countless numbers of periodic waves offsetting each other.

There are many other apparently simple phenomena in nature which

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are due to a combination of periodicities, as a beam of light or the general strike of a complexly folded iron formation.

The net result of the wave motions down the rods operating under the laws of resonance is translated into a wave whose length is nearly parallel to the circumference of the core and whose amplitude is along the radius, thus producing uniform threading. This is recorded in the rock like the mark on the indicator card of an engine. Only in rare instances do all the factors combine under the laws of resonance so that a deep definite tracing of the wave motion is formed.

Mathematical Interpretation

We may attempt to arrive at a rough mathematical interpretation of wave motion at the end of a diamond bit as expressed by threads in the core.

Let R = the number of revolutions of the bit during any interval of advance.

N = the number of threads appearing on the cross-section of core

$F(p)$ = some continuous or discontinuous function of the pitch of the threads, p having values from negative to positive infinity, $F(p)$ approaching the values -1 and $+1$ at these ends, and approaching the value zero as p approaches zero, the value of p being related to the value of N.

Let W = the number of major wave motions during any interval of advance of the bit, as indicated by the threads.

Then $W = R\,N\,F(p)$

Thus in the instance where pentagonal core is formed, and N equals 5, p approaches infinity and $F(p)$ approaches the value 1, if it be assumed that 600 revolutions of the bit have been made during the delivery of 1 in. of core; then $W = 5 \times 600$ or 3,000, which is indicative of the number of major vibrations of the bit with respect to the axis of the core during its spiral advance of 1 in.

When the pitch p is left-handed, $F(p)$ may be considered as varying from 0 to -1; when the pitch is right-handed $F(p)$ varies from 0 to $+1$. As p becomes small in either case the value of N increases and W increases approaching the case where many fine threads of low pitch are produced; W increases greatly, which is to say that a cylindrical core is formed when there are a large number of exceedingly small wave motions of the bit to and fro. Considering an instance when N remains constant and the threads ultimately coalesce while "p" decreases in value, then the value of W decreases. The ultimate end of this process is the production of a smooth core with slight wave motion of the bit.

Inspection of this equation shows that as R increases, W must increase.

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This expresses the violence of vibration when harmonic motion of the resonant type is set up during a high speed of revolution.

This mathematical statement is merely an attempt to express some of the relationships which appear, by speculating on the nature of the phenomenon. It is doubtful whether it is possible to determine rigidly the value of $F(p)$ in any instance or to tell why p should be left-handed or right-handed or to devise any means of telling why N is 5 in one case or 9 in another. The strength of the rods, the number and character of the carbon, the way they are set with respect to each other, the water pressure in the hole as determined by the force of the pump, the clearance of the bit, etc., the amount and character of the deformation in the metal of the bit at any particular time, the kind of drilling machine, and the manner of its set up, the depth drilled, the nature of the rock, the presence of soft horizons above in the hole which may affect the vibration of the rods, the speed of rotation and the pressure on the rods—the factors which influence these values are numerous and therefore difficult to analyze into their true proportions.

Practical Considerations

Dr. Crane's practical application of the results of his research is discouraging in these days of keen competition for drilling contracts. I refer to the suggestion that it is possible to reduce "the vibratory action of the drilling mechanism by reducing the speed of rotation of the rods." An old drillman in commenting on this says that he can break up the vibration by increasing the speed of rotation. This suggestion might better be stated thus: The violence of vibration can be broken up by varying the speed of rotation.

This destroys one of the constant conditions essential to harmonic motion. The greasing of the rods dampens the wave and sets up interference by releasing friction at antinodes of the waves, thus destroying another constant. This may be inferred by examining the line of greased rods when pulled from the hole and by observing that the grease is worn clean at definite intervals.

In making this comment on Dr. Crane's conclusion I would not be understood as casting doubt on the utility of this type of research. An analysis of all the conditions which govern the advance of a diamond bit might enable operators to control the direction of diamond-drill holes. At present the hole usually takes its own direction.

Some practical considerations have occurred to me while dealing with the subject. They are no doubt commonplace to many old drillmen, but seem worth recording: When drilling deep holes in homogeneous formations, it will reduce the carbon wear and increase the footage to use stiff new rods and to use a blank bit which has the smallest possible excess

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in diameter over the size of the rods. The use of eight stones in a bit rather than four or six will reduce the possibility of one stone carrying the burden of the work. The bit should be reset frequently so that deformation of the weakened steel in long-used bits will be avoided; thus the gage of the hole may be kept more accurately. These precautions will set up conditions adverse to the development of resonant wave motion to and fro in a bit and thus tend to keep it working straight ahead, which is the true business of a diamond bit.

The Emerald Deposits of Muzo, Colombia

Discussion of the paper of JOSEPH E. POGUE, printed in *Bulletin* No. 113, May, 1916, pp. 799 to 822.

EDGAR T. WHERRY,* Washington, D. C. (communication to the Secretary†).—Dr. Pogue's presentation of the facts concerning the emerald deposits is very clear and convincing, and the only addition that I can suggest is a summary of previous theories of origin. He makes it evident that the pegmatite theory is the only one capable of explaining the existing relations, but upon certain details there may be some difference of opinion. If I understand the term pneumatolytic, it does not imply that all the elements concerned in a given deposit were transported as gases, but rather that the crystallization of these elements into the various minerals was favored by the presence of certain gaseous substances, notably H_2O, CO_2 and HF. It is highly improbable that the oxides of glucinum, aluminum, chromium, and silicon, which enter into the composition of the mineral emerald could have been transported in the gaseous form. The same is true of the metallic constituents of the parisite and other associated minerals. The explanation suggested, that solutions separated into liquid and gaseous portions, the latter ascending and forming the emerald in the upper portions of the rock only, therefore, seems to me untenable.

When two formations exist side by side and one, *A*, is mineralized while the other, *B*, is barren, the possible explanations may be classed as (1) chemical, and (2) physical.

1. Some chemical feature of *A* not found in *B* might have caused crystallization of certain minerals in the former, which did not appear in the latter. In the present instance both rocks appear to be so similar chemically that no such effect can be looked for. Pogue mentions carbon as a possible precipitating agent, but describes both formations as carbonaceous, so that the difference in the minerals of the two can not be thus explained.

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† Received June 28, 1916.

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2. The physical condition of *A* might have permitted or encouraged the passage of the solutions, while that of *B* retarded or prevented it. In the present deposit some mineralization occurs in both formations, calcite veins, albite, and pyrite being found in both, whereas emerald, parisite, and a number of minerals of minor importance occur only in the upper, *A*. It seems to me that this difference may have been produced by a change in the composition of the solutions during the progress of mineralization; at first these brought in only the constituents of albite and pyrite and deposited them with calcite dissolved from the wall rock, in both formations; the openings in *B* became completely filled, while the more numerous or larger ones which would naturally have developed in *A*, since it was the uppermost formation, remained partially open. Then, when during later phases of the mineralizing activity the constituents of emerald and parisite appeared, they were deposited only in *A* because *B* had become impermeable.

History of the Flotation Process at Inspiration

Discussion of the paper of RUDOLF GAHL, printed in *Bulletin* No. 117, September, 1916, pp. 1627 to 1681.

DAVID COLE, EL PASO, TEXAS (communication to the Secretary*).—I have read with great interest Dr. Gahl's painstaking paper giving us the details of development of flotation at Inspiration, and it seems to me that he has covered the ground so completely that there is little to discuss or criticize in connection with his subject. While it was in operation, the test mill in which the campaign was carried on was the Mecca of mill men and metallurgists. There was much to interest students of milling methods in addition to the flotation experiments, and readers of Dr. Gahl's paper will miss the information they hoped to get when the "pilot" mill results were finally compiled, and I hope that the following remarks outlining from a distance some of the things that happened there will bring forth the rest of the story of the performances recorded in these important experiments.

Prior to the use of flotation methods in concentration it had long been recognized that the "unavoidable" losses of sulphides were in the slime which is inevitably produced in the grinding operations required to free the minerals to be separated. Classifying the feed prior to its final stage of treatment had long been in style in milling. This assisted the sand-handling machines and resulted in lowering the tailings made by them, but at the expense of the slime-handling department; and while little was expected of the latter, the wisdom of complicating the process by hydraulic classification was being seriously questioned. Indeed, at the

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time that the gravity method flowsheets were being developed for Inspiration ores, there was a minor revolution in milling methods impending involving the elimination of the hydraulic classifier. This change gave no promise of higher extractions but did promise greater simplicity and much less floor space per unit of capacity. Such mills promised to be less costly to build, less expensive to operate, and equally efficient. Wholesale concentration in relatively small space would get as good results as piecemeal concentration had been getting in the multiple operations of the spreadout plants into which the porphyry treatment mills had degenerated.

The one great drawback was the slime. The desiderata of the engineer and the manufacturer was, on the one hand, to provide grinders which would produce the minimum of slime, and, on the other hand, improved "slimers" which would give the maximum recovery from the finest products. The technical press of the period reflected this state of things in the advertisements of those having the latest novelties to sell. There was much revamping of old ideas with some refinement, but nothing new. Tube-mill grinders were "taboo" for concentration, because they had a bad reputation as slimers. Automatic canvas plants were being exploited and very ingenious multiple-deck table devices were being offered as the remedy—the only possible remedy.

Looking backward no further than 1912, when Inspiration began to study its milling problems, we now see that we were without effective resource in treating real sulphide slimes. Mr. Callow's investigations and experiments had apparently demonstrated that some departure from the usual gravity practice would be advantageous and that a high recovery for that method would be possible on the granular material, but he threw small light on slime treatment. This was the situation when the Inspiration company was endeavoring to determine its mill treatment scheme.

After reviewing from every angle the results of experiments on the ore and other information available, a modified flowsheet was finally crystallized by Mr. Burch and adopted by the management. Mill plans were drawn for what was to be a most highly developed 7,000 tons per day gravity process plant, and work was immediately inaugurated to carry out these plans. The mill site was selected, and much active work had been prosecuted before flotation (by this time being hastily tried in the old experimental plant) had so far won its battle that results could be viewed otherwise than probably too good to be true.

The apparent promises of flotation were extremely attractive, because the process would be simple and would solve the all-important slime problem. The process would have a greatly reduced number of stages; the plant would be much smaller per unit of capacity; the cost of construction per unit of capacity would probably be very much less; the

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use of water would be minimized; grinding would have to be carried further than usual, and would be the main item of milling cost, but this was not a very great handicap because sliming did not matter. What system of grinding would be best to use and what is the best machine? Would it be possible to parallel the small test-mill metallurgical results on a full-tonnage basis, and finally would it not be too hazardous to accept so revolutionary a process with so much experiment in its makeup?

The mine preparation program which had been settled upon would produce about 600 tons daily of freshly broken ore directly from the headings. This happened to be the rated capacity of a full-sized Minerals Separation unit which the flotation people were urging as a means of improving their extraction. If this ore from the headings were put in stock pile in the usual way, pending the completion of the milling plant, the ore would oxidize to some extent and besides would involve reclaiming expense later on. Why not mill it all as fast as produced, in a real "pilot" mill, wherein not only flotation problems, but grinding problems, power consumption, use of water, preparation of sticky concentrates, and other vital questions which might come up could be definitely threshed out under what would be regular commercial conditions on full-sized machines? In this size of "pilot" operations, labor would be used economically and production would almost, if not quite, pay all of the current expenses, except the mining cost.

This program was adopted. That the decision to carry it out was a wise one is shown in Dr. Gahl's paper, and that it paid its way is shown by Mr. Mills' annual report for 1915, in which he says: "Contrary to the usual experience, this test mill paid the cost of its construction, its operation expense, the present average mining cost on ores treated, and something besides, and has been written off the books."

Anybody with a real idea applicable to the problem could get a hearing and a tryout in this extremely practical commercial sized laboratory where sampling was in the hands of engineers and the results were compiled in a way to make them comparative and valuable. Thousands of dollars were spent by the company, by inventors, and manufacturers in demonstrations. Expense was subordinated. Heavy shipments by express were made when necessary to hasten the work, and much more than flotation was developed in this "pilot" plant.

Four different types of Symons crushers and pulverizers were tried. Three of these were marked and very interesting departures from ordinary practice. The company had purchased the Hardinge mill patent rights for Arizona and four forms of this mill were installed to determine the best to use, and these were kept busy during nearly the whole campaign. A long parallel tube mill was installed and run in competition with the Hardinge mill. A high-speed Huntington-type grinder at one time attracted much interest. Hammer pulverizers of two different makes

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were also tested. Various linings and grades of flints were tried in the pebble mills. Steel balls in place of pebbles were advocated and a carload purchased.

In the latter part of the testing period the Marcy type of ball mill, especially designed for using iron balls larger than usual, and adapted to crush from breaker sizes to 48-mesh in closed circuit in one operation, was installed and perfected. This grinder proved capable of a greater range of reduction than had previously been thought possible, from an initial feed as coarse as 3-in. cubes. It is a ball mill pure and simple, having large capacity in small space. It makes use of a perforated diaphragm to keep the balls and charge inside of the mill until the latter will pass a $\frac{3}{16}$-in. opening, and it has the equivalent of a peripheral discharge. An overflow classifier determines the finished size and the oversize is continually returned to the grinding chamber. This mill uses little water in the grinding chamber, so that its charge of ore is mortar like in consistency. It was quite successful. There was nothing in the Hardinge equipment to parallel it because the Hardinge mills were built for pebble mills and did not have feed scoops or openings adapted to handle as coarse a feed, and the linings would not stand up under the ball load. Would the Hardinge machines when built as a ball mill with the required strength, type of lining, size of feed and discharge openings, do as well? To wait for a mill to be made over or a new one manufactured would take too long, so the Marcy type was adopted and, contrary to what I think is the popular impression, the conical-type steel-ball mill *vs.* the Marcy-type steel-ball mill did not receive a tryout.

Recording electric instruments were installed so that accurate power records could be continually made while the various machines were being operated.

Several varieties of drag and rake classifiers and two types of vacuum filters were installed and records made. The efficacy of high-reduction herringbone gears driving ball and tube mills became a matter of interest on account of the troubles that developed in them, and the reasons for these troubles, which would make a story by itself when discovered.

This testing work grew into a process of elimination and the scrap pile grew steadily. Some blasted hopes may be buried there, but it does not follow that all of the machines or materials that were returned to the sponser or that found their way to the scrap heap were entirely unfit. It was necessary to choose and to discard, and that there is no acrimony in connection with the matter speaks well for the type of justice and judgment that prevailed. Doubtless the use of some discarded things might have answered as well, but it would be hazardous indeed to say that anything vital on the score of cost or recovery of values failed to receive recognition in the final selection. One of my mental offspring was among the fallen. It held out for a long time and I greatly appre-

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ciate the favorable mention which Dr. Gahl has made of it. I would have been pleased to pursue its development further, but I can not at this time see where its adoption would have saved any more copper or any more money in getting the copper than the ones that were selected, and I think this will apply to all the "late lamented" concerned in the campaign that was carried on. In other places and on other ores the conditions would not be the same. The ratio of concentration, or the crushability factors particularly, may so change conditions as to indicate the use of ideas that were discarded at the Inspiration test mill.

When the comprehensive plan of testing on a large scale was decided upon, I think Dr. Ricketts and Mr. Mills recognized that the records made might be extremely valuable to engineers and metallurgists, and I believe the matter of writing the story of the campaign was considered and that records were started with that ultimate end in view. Experience added to theory is more valuable than theory alone; the scrap heap more eloquent than the machine promotor, and the wornout unit in the "bone yard" is often more interesting than the one shown on the original blue print. It is seldom indeed that experiments are possible under such auspices and on such an extensive scale as those carried out in the Inspiration "pilot" mill, and I hope that Mr. Burch or Dr. Gahl, or both of them, will, with the approval of the Inspiration company, find time and inclination at an early date to prepare a paper or papers which will extend, through the *Transactions*, the value of these experiences to our members.

Referring to the final flowsheet adopted, I note that hydraulic classification had no place in the 600-ton "pilot" mill experiments which Dr. Gahl has so interestingly described, and I note that he has not referred to the reason for retaining this remnant of the old system of concentration in the flow sheet.

I note that 3 tons of water per ton of ore handled is required in the flotation operation and that 3 tons more is added in the subsequent table treatment which, of course, includes the hydraulic classification operation practiced, and I presume that something more than one-half of the last 3 tons is added in the classifiers themselves; and since the water is clarified and returned, the addition of unnecessary water would entail expense for clarification and pumping which would not be required if less water were used.

In my work in the concentration of ores I have not been able to become very enthusiastic over hydraulic classifiers, and since flotation has come to the rescue of the slimed sulphide I find myself less enthusiastic than ever over their use.

When I was a lad in the Black Hills I was fascinated in watching a certain mountain spring in which polished micaceous particles glistening in the sunlight would be caught in the current rising from an orifice in the bottom of the sand funnel and be flirted to the surface, sail across the

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crater and fall upon the conical sides to be again methodically returned over the same route. It was interesting to watch the disturbance caused by dropping a handful of foreign sand and silt into the funnel and have it "classified" and washed clean, a new form of crater finally being established with the changes of average sizes retained. My first contact with hydraulic sizing in concentration was studied from that foundation. The spring took its time to do a good job; it worked the charge over repeatedly. All the silt went out quickly and the fines gradually went overboard with a rapidly decreasing ratio, until the crater would settle down again to its regular work of turning the mobile contents over and over in a new condition of equilibrium. But I soon learned that the beauty was all taken out of the process in its commercial application.

The process witnessed in the action of the spring was balanced, precise, and definite, and quite at variance with what we witness in watching through glass the operations going on in a "teeter chamber" of the metallurgical hydraulic classifier, which seems to me to be of little value except in its office of washing out the slimes which used to be the main source of the loss in treating unclassified material upon concentrating tables.

Following the thorough combing out of the slimed values as effected by the splendid flotation treatment that the pulp has previously received in the Inspiration final flowsheet, I am inclined to question the value of the subsequent classification by hydraulic means. I note that the flotation tailing is split into slime and sand at the drag belts, that there is very little slime left in the sand portion, and that what little there is (on account of the previous frothing) is completely devoid of value which the tables can save, as indicated by the fact that the main slime overflow of the drag-belt separators is discharged to waste without further treatment. It seems to me that it ought to be possible to eliminate the classifiers, and I would like to ask Dr. Gahl if there has been any trial to determine what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation.

Comparative Friction Test of Two Types of Coal-Mine Cars

Discussion of the paper of P. B. LIEBERMANN, printed in *Bulletin* No. 114, June, 1916, pp. 1057 to 1065.

EDWIN M. CHANCE, Wilkes-Barre, Pa. (communication to the Secretary*).—I have read with great interest Mr. Liebermann's paper reporting dynamometer tests of mine cars and wish to express my appreciation of this investigation. The coal-mining industry needs just such precise study of the problems with which it is confronted.

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* Received June 24, 1916.

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It would seem, however, that the writer has demonstrated that a high-grade grease is superior to black oil as a mine-car lubricant rather than that roller-bearings are superior to plain. While it is reasonable to believe that roller bearings will cause less friction than plain when used on the trucks of mine cars, still this point can not be conceded on the strength of the tests quoted, for in these tests grease was the lubricant of the roller bearings, while the usual black oil, of low lubricating value, was the lubricant of the plain bearings. Under these conditions, a case is made for the lubricant rather than for the bearings.

Some time ago certain of the coal mining companies with which I am associated, called upon me to have a high-grade grease prepared for them for use in plain mine-car bearings. This problem was successfully solved and this lubricant has now been in use for some years. The fact has been established that the ordinary plain mine-car journals are especially susceptible to such a lubricant and that its use greatly decreases the drawbar pull of these cars. The fact must be borne in mind, however, that the exigencies of this service require a special grease and that the use of the ordinary grades of cup grease will surely spell disaster.

It would be indeed interesting to have tests of the same degree of precision as those reported made upon cars with plain bearings lubricated with black oil and special mine-car grease and to compare the latter with tests under like conditions made upon cars equipped with roller bearings.

CHARLES LEGRAND, Douglas, Ariz. (communication to the Secretary*).—Mr. Liebermann's paper on friction tests of two types of coal-mine cars is very interesting. It is a pity that the dynamometer car was not designed for use on tracks of 18 to 24 in., as this is the most usual gage in metal mines in this part of the country.

I have tried to make tests by means of an ordinary spring dynamometer and find, as stated by Mr. Liebermann, that this method is very unsatisfactory, as the conditions of mine tracks are too variable to produce a steady pull on the dynamometer. It would be very valuable to the mining industry to have accurate tests taken on narrow-gage track, to determine not only bearing friction, but the total track resistance. Such tests would convince the mine managers that it pays to lay heavy tracks and keep them in good condition.

Personally, I have always been in favor of the roller bearing, not only on account of its lower friction but also on account of its smaller maintenance and lubricating costs. To be successful, however, this type of bearing has to be dustproof. On a narrow-gage car this is usually done by inclosing the whole axle in a casting which also supports the two bearings. This casting acts as an oil reservoir and permits the car to be run a whole month with one oiling.

* Received July 5, 1916.

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I hope that Mr. Liebermann will continue his work and make tests of the narrow-gage cars.

JOHN PHILLIPS, Pittsburgh, Pa. (communication to the Secretary*).—I am familiar with the test made by the Greensburg Coal Co. and described in Mr. Liebermann's interesting paper, although I was not personally present. Mr. Liebermann's results on the Hockensmith plain-bearing wheels compare closely with tests which have been made elsewhere and show a considerable saving in favor of the roller bearings. However, as the grade increases, the ratio in favor of the roller bearings decreases until, on a 4 per cent. grade, their efficiency is only 11 per cent. greater than that of the plain bearings, as shown by the following official figures of the Hyatt company:

Test, Oct. 3, 1915

	Pounds
Weight of car	2,650
Weight of coal	4,500
Total	7,150

Weight of 20 cars loaded, 2.4 per cent. grade average.. 71.5 tons.
Correction..... 48 lb. per ton.

$71.5 \times 48 = 3{,}432$ lb.

4,280		5,190	
4,350		5,070	
4,410		5,250	
3)13,040		3)15,510	
4,347	Hyatt	5,170	plain
4,347		5,170	
3,432		3,432	
915		1,738	

Frictional resistance for Hyatt, $\frac{915}{71.5} = 12.8$ lb. per ton.

Frictional resistance for plain bearings, $\frac{1{,}738}{71.5} = 24.3$ lb. per ton.

Drawbar, Pounds per Ton		Grade, Per Cent.	Per Cent. Saving	Percentage Cars Possible to Add in Case of Roller Bearings
Roller Bearings	Plain Bearings			
12.8	24.3	level	47⅓	90.0
32.8	44.3	1	26.0	35.0
52.8	64.3	2	17.9	21.8
72.8	84.3	3	13.6	15.8
92.8	124.3	4	11.0	12.4

It will be noticed that all the readings were taken on a straight track on an average grade of 2.4 per cent. Had any of the readings been taken on a curve, we are confident that the drawbar pull on both the roller

* Received July 10, 1916.

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bearings and the plain bearings would have been considerably higher when the train was rounding the curve, because the axles of the Hockensmith trucks are of straight cold-rolled steel without collars, and in rounding the curve the car naturally shifts against the inside wheel. The rear hubs of the wheels are necessarily large in order to admit the bearings, and the large hub grinding against the box creates considerable friction, which would increase the drawbar pull on a curve.

Records of three tests conducted by the Engineering Department of the Pittsburgh Coal Co., on three different types of wheels, are given below. At Essen No. 3 mine, the Jarvis Adams roller bearings were used; at the Champion mine, Eureka wheels and angle-bar trucks (the wheels being similar to the plain-bearing wheels used in the Greensburg test), and at the Dickson mine, our open-cap wheel trucks. Inasmuch as the drawbar pull on the Hockensmith plain-bearing wheels at Champion agrees very closely with the results on the Hockensmith plain bearings at Greensburg, the results may be taken as approximately correct, and show the following comparison:

Hyatt roller bearings at Greensburg	12.8 lb. per ton.
Jarvis Adams roller bearings at Essen No. 3	15.14 lb. per ton.
Open-cap trucks, Dickson mine	18.48 lb. per ton.
Hockensmith plain bearings, Champion mine	24.40 lb. per ton.
Hockensmith plain bearings, Greensburg	24.30 lb. per ton.

The figures on the Hyatt bearings were lower than we anticipated, as we had always supposed that the Jarvis roller bearings when new and in good condition were the easiest running on the road. Our open-cap truck costs about the same as the Hockensmith angle-bar truck, but runs about 25 per cent. easier. The cost of the roller-bearing truck is about $10 to $12 per car more than our truck, and on the basis of the Greensburg figures runs about 30 per cent. easier.

From the above data, I think that it is safe to say that it would be well for Mr. Liebermann, or some person equally well qualified to carry out this class of tests, to pursue it much further, both as to different kinds of wheels, and to the character of roads, which would include different grades and varied curvature, before a thorough comparison could be made and a conclusion arrived at that would be entirely satisfactory.

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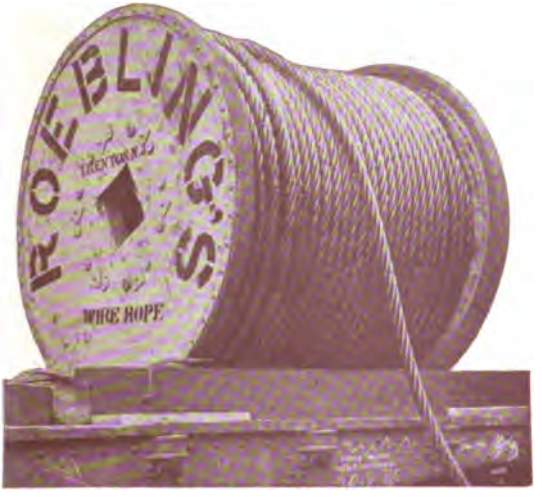
ROEBLING WIRE ROPE



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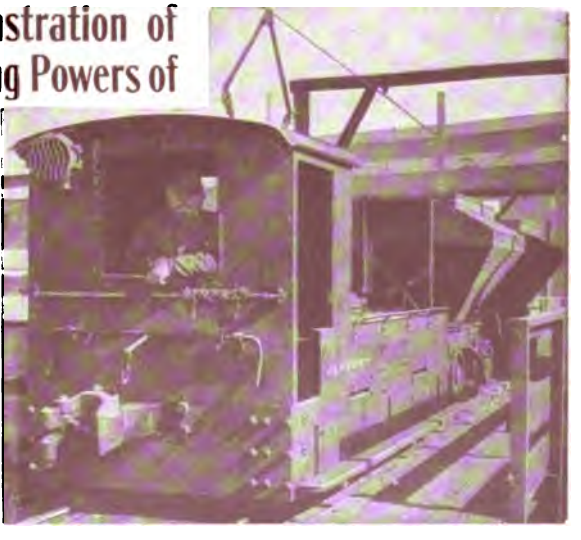

A 10-Ton Jeffrey Electric Locomotive hauling cars of Hot Calcine from Wedge Roaster to Reverbatory Furnace, at the plant of the Kansas City Smelting & Refining Co., El Paso, Texas.

Bulletin No. 115-2 describes this installation, and pictures other Jeffrey Locomotives in similar service. Send for copy

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NOVEMBER, 1916

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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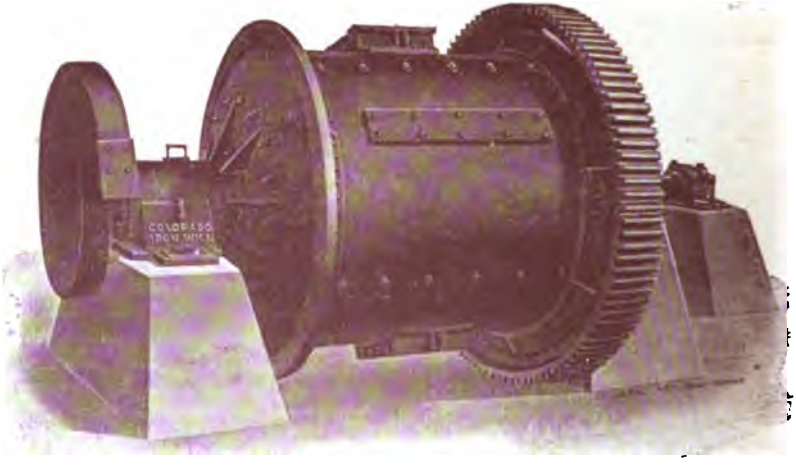

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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

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THE ARIZONA MEETING

A full account of the wonderful Arizona meeting will be published in the December *Bulletin*. For the strenuous, interesting, and altogether happy week, we express our hearty thanks to those marvels of energy and thoughtfulness, the members of the local committees in Texas, New Mexico and Arizona.

In this number we publish the discussion resulting from the technical session on "Flotation." Because of the timeliness of the subject, this session's discussion was transcribed first and sent to the contributors, and, thanks to their promptness in revising, we are able to print it in this issue.

MANUSCRIPT CLOSING DATE FOR THE NEW YORK MEETING, 1917

The 114th (New York) Meeting of the Institute will be held in the third week of February, 1917. The committee on Papers and Publications has set Dec. 1, 1916, as the closing date for the receipt of manuscripts intended for this meeting.

Manuscripts must be received by the Secretary on or before this date. If they are sent through members of the technical committees they must be sent to those committees enough in advance of the time to be forwarded and in the hands of the Secretary by the time specified.

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While Dec. 1, 1916, is the final closing date, it is a great advantage to get papers in as much before that time as possible as it enables them to be submitted in proof form to their authors. Papers received on or before Nov. 1, 1916, will, as far as possible, be published in the January *Bulletin* or earlier, thus giving good distribution among the members in advance of the meeting.

The Institute's records show that the discussion of papers is very much better when they are distributed somewhat in advance of the meeting instead of in the *Bulletin* which goes out immediately preceding the meeting.

COMMITTEE ON INDUSTRIAL PREPAREDNESS

The work of the Committee on Industrial Preparedness* is well summed up in the following letter written by Thomas A. Edison, Chairman of the Naval Consulting Board, after President Wilson had signed the army appropriation and army reorganization bills:

"The Committee on Industrial Preparedness of the Naval Consulting Board, of which committee Howard E. Coffin is chairman, after a five months' campaign in many ways unique in vigor and vision, has played a very definite part in the measures for the national defense which have just become law. It is responsible for having laid the foundation for a true industrial preparedness in this nation.

"Five months ago the committee, wholly non-partisan in make-up, acting with the civil, mechanical, mining and electrical engineers and the chemists of the country, members of five great scientific bodies, began its work in every nook and cranny of the land. The purpose was to prepare American industry efficiently to support the armed forces of America in the event of war.

"The public, the business men and the legislators at Washington knew little regarding the demands on industry in connection with modern war. No department at Washington was charged with the duty of coördinating the needs of the army and navy with transportation, industry, and kindred functions. The purchasing, quartermaster's and ordnance departments of the army and navy were unable to buy military supplies except by competitive offering in the open market. No one had considered the need, and no legislative machinery was in existence, for preventing the presence of skilled workers on the fighting front when they would in time of war be needed in the factories, mills and mines of the country. Neither at Washington or elsewhere was there even a list of the industrial resources of the nation in such shape as could be used for practical purposes in case war should come.

"The three things, then, that the committee headed by Mr. Coffin set out to do and that now have been brought about were as follows: First, to mobilize concretely the business men of America and their output against a day of need. This meant chiefly to discover in fullest detail just what equipment our manufacturers possessed that might be swung over to supply the army and navy swiftly and bountifully from the hour the colors were raised. In a few weeks this initial task, which has been a truly colossal one, will be at an end. A complete report of the equipment of practically every concern of substantial size in the country is now in the hands of the committee. Directly in charge of the great inventory has been W. S. Gifford, Supervising Director of the Committee.

"The second step was to procure legislation making possible the placing by the war and navy departments of annual orders for munitions in small quantities to American manufacturers now not producing such things, so that quietly, efficiently and thoroughly they could learn in time of peace how to supply the government in time of war. The new laws make all this possible. It should be said here that this practice will bring about a geographical diffusion of munition plants all over our country, instead of affording us only a few such concerns for the most part near the exposed seaboards, as at present. It is obvious that this system will also throttle all talk of a munitions trust.

"Further to teach manufacturers how to produce munitions, the army appropriation bill permits the purchase of 'gauges, dies, jigs, tools, fixtures, and other special

* See *Bulletin* No. 113, p. vi (May, 1916).

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aids and appliances, including specifications and detailed drawings necessary for the manufacture by the government and by private manufacturers of ammunition necessary for the use of the land forces of the United States in time of war.'

"The third step was to cause skilled labor in America to be enrolled in an industrial reserve. For, as Mr. Coffin has repeatedly told the country, it takes but ten or twelve months to turn out a soldier, but years to make a tool-maker. All this is provided for in the army reorganization bill just signed.

"At the same time there has been created the Council of National Defense, composed of the secretaries of war, navy, interior, agriculture, commerce, and labor, and seven civilians, each eminent in his respective field, and each to serve the government of the United States without pay, as we on the Naval Consulting Board have been doing.

"Part of the new act reads—and every American citizen should know about it—That it shall be the duty of the Council of National Defense to supervise and direct investigations and make recommendations to the President and the heads of executive departments as to the location of railroads with reference to the frontier of the United States so as to render possible expeditious concentration of troops and supplies to points of defense; the coördination of military, industrial and commercial purposes in the location of extensive highways and branch lines of railroads; the utilization of waterways; the mobilization of military and naval resources for defense; the increase of domestic production of articles and material essential to the support of armies and of the people during the interruption of foreign commerce; the development of seagoing transportation; data as to amounts, location, method and means of production, and availability of military supplies; the giving of information to producers and manufacturers as to the class of supplies needed by the military and other services of the Government, the requirements relating thereto, and the creation of relations which will render possible in time of need the immediate concentration and utilization of the resources of the Nation.

"The work of the Committee on Industrial Preparedness will almost surely be turned over to the new Council, the committee's remarkable constructive programme being in effect already an accomplished fact. With this accomplishment comes not merely a better understanding between the business men of America and their government but it marks almost dramatically the entrance of the trained, non-partisan engineer, doing his job on the sole basis of efficiency, integrity and Americanism, into the affairs of government.

"But the greatest services which the committee has rendered to our people and government has been to drive home the vital, irresistible truth that in war as now waged, battles are won not alone by fighting men but by the fighting industries of a nation."

The prominent business men and engineers composing the Committee on Industrial Preparedness are: Chairman, Howard E. Coffin, Vice-President of the Hudson Motor Car Co.; Thomas Robins, President of the Robins Conveying Belt Co., and Secretary of the Naval Consulting Board; B. G. Lamme, Chief Engineer of the Westinghouse Electric and Manufacturing Co.; William L. Saunders, President of the Ingersoll-Rand Co., and Second Vice-Chairman of the Naval Consulting Board; Lawrence Addicks, Consulting Engineer; William LeRoy Emmet, Mechanical Engineer of the General Electric Co., Schenectady, N. Y., and Benjamin B. Thayer, Vice-President of the Anaconda Copper Mining Co.

ENGINEERS COMBINE WITH SCIENTISTS IN ORGANIZATION OF NATIONAL RESEARCH COUNCIL

Arrangements have been completed in New York whereby the resources of The Engineering Foundation, under the auspices of the four principal national engineering societies, are placed at the disposal of the National Research Council, which was appointed by the National Academy of

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Sciences* at the request of President Wilson. The object of the Council is to coördinate the scientific research work of the country in order to secure efficiency in the solution of the problems of war and peace. The Council was without funds until The Engineering Foundation, established to further scientific and engineering research, offered to place its resources at the Council's disposal, including the services of its secretary, Dr. Cary T. Hutchinson, to act as secretary of the Council. The offer was accepted and plans for immediate activities have been placed in the hands of an executive committee.

In indicating how thoroughly every branch of science and engineering is represented, Dr. George E. Hale, Director of the Mt. Wilson Solar Observatory, and Chairman of the Council, when he was in New York attending to the details of the arrangement, called attention to the personnel of the body, saying that it is the purpose to enlist the coöperation, in the solution of our industrial and military problems of a scientific character, of every possible established agency. Medicine, for example, is represented on the Council by Dr. William H. Welch, President of the National Academy of Sciences, Brigadier General William C. Gorgas, Surgeon General of the United States Army, Dr. Simon Flexner, Director of the Rockefeller Medical Institute, and Dr. Victor C. Vaughan, Past President of the American Medical Society; biological science by Dr. Edwin G. Conklin, Professor of Zoology, Princeton University; chemistry by Dr. A. A. Noyes of Massachusetts Institute of Technology and Dr. L. H. Baekeland; physics by Dr. A. A. Michelson of the University of Chicago; and electricity by Professor M. I. Pupin, of Columbia University.

These branches, with the exception of medicine, are in the realm of pure science. Recognizing, however, that the practical applications of the principles which the pure scientists discover rest largely with engineers, there is a strong representation from the great engineering societies. Clemens Herschel, President of the American Society of Civil Engineers; John J. Carty, Chief Engineer of the American Telephone & Telegraph Co.; Gano Dunn, President of the J. G. White Engineering Corporation; C. E. Skinner, Director of the Research Laboratory of the Westinghouse Co. and Dr. W. R. Whitney, Director of the Research Laboratory of the General Electric Co., are among those who will represent the engineering side of the Council's work.

The important military aspects will be presented to the Council by Major General William Crozier, Chief of Ordnance of the U. S. Army, by Lieutenant Colonel George O. Squier, Chief of Aviation of the U. S. Army, and Chief Constructor David W. Taylor, U. S. Navy. Other branches of the government are represented by Dr. S. W. Stratton, Director of the National Bureau of Standards; Van H. Manning, Director of the Bureau of Mines and Prof. Chas. F. Marvin, Chief of the United States Weather Bureau.

* "During our civil war the need of scientific advice was often felt by our Government. Accordingly, the National Academy of Sciences was chartered in 1863 by act of Congress, which stipulated that 'the academy shall, whenever called upon by any department of the Government, investigate, examine, experiment, and report upon any subject of science or art. * * *' During the war, and frequently in later years, the academy has been consulted by Congress, by the President, and by various members of his Cabinet. It is thus the agency naturally chosen for the organization of the National Research Council." From a letter written by Dr. George E. Hale to the *New York Times*, in July, 1916.

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It will thus be seen that the Council includes representatives of all of the important scientific activities bearing on military or industrial problems. The executive committee's plans are of wide scope; the support already pledged will insure immediate action where the need is greatest.

Our English brethren by a similar organization have been of immense assistance to their government during the war. The American organization now being perfected should be of equal value to its government, but it is hoped that the benefits industrially will be no less important than the military ones.

The members of the Council not already mentioned are Dr. John A. Brashear, Pittsburgh; Dr. W. F. M. Goss, Dean of Engineering, University of Illinois; Dr. William H. Holmes, Curator, United States National Museum; Dr. W. W. Keen, President, American Philosophical Society; Prof. E. C. Pickering, Director, Harvard College Observatory; Mr. Charles F. Rand, President, United Engineering Society; Prof. Theodore W. Richards, Harvard University; Prof. R. A. Millikan, University of Chicago; Mr. Ambrose Swasey, Cleveland; Dr. Elihu Thomson, Swampscott, Mass.; Dr. C. R. Van Hise, President, American Association for the Advancement of Science; Dr. Charles D. Walcott, Secretary, Smithsonian Institution; Dr. J. M. Coulter, Professor of Botany, Princeton University; Prof. R. H. Chittenden, Dean of Sheffield Scientific School, Yale University; Prof. Raymond Pearl, Biologist, Maine Experiment Station, Orono, Me.; M. T. Bogert, Professor of Organic Chemistry, Columbia University.

Other members of the council will be appointed from time to time as the needs of the work dictate.

INTERESTING EXCURSION TO A SUBMARINE BASE

New London, Conn., Saturday, Nov. 11, 1916

The American Society of Mechanical Engineers extends an invitation to members of the Institute to participate in an exceedingly interesting excursion on Saturday, November 11.

Permission has been secured for a visit to the plant of the Electric Boat Co. at New London, Conn. There will be run a special test on a high-power 8-cylinder, 4-cycle Diesel engine. A large number of submarine and other engines in various stages of construction will be seen, also the boring of torpedo tubes, cylinders and other equipment for submarines. The drafting rooms and foundry will also be open for inspection.

The party will have an opportunity to see at close range the submerging of several submarines, and an attack on a vessel on the surface, an exhibit which is being arranged by courtesy of Rear Admiral A. W. Grant, who is in command of the submarine forces of the United States.

The Boston members will leave South Station on train No. 7 at 8.30 A. M. Those from Providence will board this train at 9.36 A. M. The Worcester members will leave on train No. 745 at 8.36 A. M.

The New York contingent will leave the Grand Central Terminal at 8.31 A. M. on train No. 8. This train will leave New Haven at 10.30 A. M.

Upon arrival at New London, each group will go directly to the Mohican Hotel, where a "get-acquainted" luncheon will be served

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promptly at 11.45 A. M., so that the party will be able to leave at 1 P. M. for the plant of the Electric Boat Co.

Returning, the following train connections can be made:

For Boston and Providence:

4.07 and 6.00 P. M., arriving at Boston at 7.00 and 8.43 P. M. respectively.

For New Haven and New York:

3.57 and 5.41 P. M., arriving in New York at 7.11 and 8.45 P. M. respectively.

For Worcester:

4.20 P. M., arriving at Worcester at 8.00 P. M.

Members desiring to join this excursion and to participate in the "get-acquainted" luncheon, please notify the Secretary promptly.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at the Institute headquarters during the period Sept. 10, 1916 to Oct. 10, 1916.

Arthur K. Adams, Chanaral, Chile.
Hiram W. Belnap, Washington, D. C.
E. A. Bridgman, Seattle, Wash.
D. W. Buckly, Wallace, Idaho.
G. L. Carlisle, Jr., New York, N. Y.
Ernest H. A. Cohen, Johannesburg, So. Africa.
A. Faison Dixon, Camp Rio de Oro, Venezuela.
Charles E. Dutton, Goldfield, Nev.
D. F. Hewitt, Washington, D. C.
James E. Howard, Washington, D. C.
A. O. Ihlseng, Joplin, Mo.
I. Ishikawa, Niagara Falls, N. Y.
Montrose L. Lee, New York, N. Y.
V. H. McNutt, Billings, Mont.
Benjamin L. Miller, So. Bethlehem, Pa.
C. F. Moore, Salt Lake City, Utah.
L. D. Moore, Providence, R. I.
George A. Morrison, Chicago, Ill.
Pablo Ortega, Havana, Cuba.
L. W. Paige, New York, N. Y.
Charles Petzel, New York N. Y.
C. A. Pierce, Patagonia, Arizona.
W. E. Pomeroy, El Paso, Texas.
George S. Rice, Washington, D. C.
G. J. Schofield, Seattle, Wash.
Warren D. Thompson, New York, N. Y.
M. M. Valerius, Tulsa, Okla.
G. B. Waterhouse, Buffalo, N. Y.
Joel H. Watkins, Washington, D. C.
H. P. Wherry, Chicago, Ill.

Arthur K. Adams has left for Chile, where he will act as geologist for the Andes Copper Co., a subsidiary of the Anaconda copper company.

H. H. Colley, formerly assistant superintendent of the Copper Queen smelter, has succeeded L. O. Howard as superintendent of the Old Dominion mill and smelter at Globe, Ariz.

Robert E. Cranston has been appointed consulting engineer to the Mining Associates, Ltd., operating the Rawley mine near Salida, Colo. He is developing and equipping this property for production.

G. H. Cunningham has resigned his position as superintendent of construction with the Anaconda Copper Mining Co., to become chief engineer for the Electrolytic Zinc Co. of Australia, at Hobart, Tasmania.

C. Mason Farnham, chief geologist for the Cerro de Pasco Mining Co., is engaged in special geologic work at Harvard University and the Massachusetts Institute of Technology.

James Gayley, formerly vice-president of the United States Steel Corporation, is the president of the newly organized Alaska Mines Corporation.

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H. R. Hanley, until recently general manager of the Bully Hill Copper Co., has been appointed superintendent of the new electrolytic plant that the U. S. Smelting Co. is to erect near Kennett, Cal.

W. L. Honnold is in New York, as director in America of the Committee for Belgian Relief.

L. O. Howard, formerly superintendent of the Old Dominion smelter and mill at Globe, Ariz., is now in charge of the International Smelting Co.'s works at Miami.

W. J. Lakeland has resigned his position with the Burma Mines and has joined the Indian Army Reserve of Officers.

Professor Francis Church Lincoln, director of the Mackay School of Mines, University of Nevada, Reno, Nev., has recently returned after spending the summer in mining work in Bolivia and Peru.

E. P. Mathewson has resigned as manager of the reduction works of the Anaconda Copper Mining Co. at Anaconda, to accept the position of general manager for the British America Nickel Corporation at Sudbury, Ont., with headquarters at Toronto.

Frank W. Parker has resigned as superintendent of the Rainbow Mine, in eastern Oregon. His address for the present will be 1619 Hays St., Boise, Idaho.

Charles A. Randall has resigned his position as mill superintendent with the Tough Oakes Gold Mines, Ltd., to accept a position with the Messrs. Simonds and Burns, mining engineers of New York, as superintendent of construction and operation of two cyanide plants in Cuba.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute of members or other persons introduced by members.)

Member, Cornell M. E., graduate, aged 31, married. Will soon complete 2 years' engagement in the Russian oil fields. Eight years' practical experience in drilling for and producing oil. Can speak Russian fluently and some French and German. Open for engagement. Interview after August 15, in the United States. No. 299.

Metallurgist, four years' experience in electric furnace operation on steel, ferro-alloys and smelting of ores, both practical and research, desires position as metallurgist or superintendent. No. 314.

Member, technical mining graduate. 5 years' experience in coal and metal mines and construction work. Married. Aged 27. Employed at present. Interview in Salt Lake City or Denver. No. 316.

Member, American, aged 39, married, desires position as superintendent or mine superintendent. 14 years' experience in opencut and underground work. 12 years' Spanish, Portugese, Cuban and Mexican labor. Fluent Spanish. Good training in first aid and safety first. Specialty of high explosives and steam-shovel work. Good costs. Go anywhere, but tropics preferred. References. Interview, New York or Philadelphia. No. 318.

Member, aged 29, unmarried, technical graduate. 7 years' varied experience in metal mining; 3 years' as draftsman, surveyor and assayer;

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4 years' operating experience as foreman, superintendent, and manager. Good health and habits. References. At present employed, available on short notice. Can handle men efficiently and get results. Employment with reputable company desired, preferably as foreman or superintendent. No. 319.

Member. Graduate Columbia School of Mines. Single. Thirty years old. Six years' practical experience, 5 years in Mexico and Central America. Fluent Spanish. Good organizer, able to handle men. Desires position as superintendent or assistant in metal mine in U. S. or foreign country. No. 320.

Young technical graduate who has had one year's post-graduate work in geology, would like to get into a geological department or to work as a mining engineer in engineering or mining departments. Willing to go anywhere. No. 321.

Member, technical graduate, aged 32. Course of studies including mining, metallurgy, chemistry and geology. Previous practical experience in erection and operation of machinery. Have had about 5 years' varied engineering experience. At present mining engineer for large copper company, but desire change with any substantial company desiring a reliable man. References furnished as to ability and character. No. 322.

Member. Experienced mining geologist and engineer open for long or short engagement. Familiar with exploration and development of both large and small mines. New York interview. Can supply own instruments and experienced assistant for surveys, etc. No. 323.

Mining engineer, technical graduate, with 5 years' experience, desires position. At present employed as superintendent of gold mine in Alaska. Free after November 1. No. 324.

Member, Columbia E. M., married, aged 32, 10 years' experience in gold-silver mining and milling in Mexico and United States, including management, desires position as manager or assistant manager, or with examining engineers. No. 325.

Member, graduate Colorado School of Mines. Have had considerable and valuable experience in mine surveying and mine sampling. Desire a position along those lines. Available on three weeks' notice. No. 326.

Member, technical graduate, aged 25. 2 years' experience in copper milling, with particular reference to flotation. At present superintending construction of a mill and doing assaying and surveying. Desire position as superintendent or assistant superintendent of a mill. Can furnish references. No. 327.

Member, aged 26, married, technical graduate, 4 years' experience in testing, experimental and research work at large copper smelters and refineries, at present engaged in South America, desires to communicate with operating company in Russia or Africa where such experience would be of advantage. No. 328.

Member, graduate of the Columbia School of Mines. Drifting and shaft sinking a specialty. 7 years' practical experience in operating and developing work. Can open up new property or take entire charge of mine, skilled in up-to-date methods. At present employed, desire change.

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Able to handle organization of men, excellent results. Will go anywhere. Moderate salary. References. No. 329.

Member, aged 33, professor of geology in eastern technical institution, who has had 3 years' post-graduate work in geology, desires position on geological staff of mining or oil company in the United States. Will consider secondary position if prospects are good. Available on short notice. Interview if desired in New York or Chicago. No. 330.

AFFILIATED STUDENT SOCIETY NOTES

The Mining Society at the **Massachusetts Institute of Technology** has elected the following officers:

G. A. Roper, President,
P. M. Rowe, Vice-President and Treasurer,
R. W. Rogers, Secretary.

The Student Branch of the **Ohio State University** held its initial meeting of the school year on Oct. 4, electing the following officers:

President, W. R. Collette,
Vice President, R. C. Lawrence,
Treasurer, H. T. McMillen,
Secretary, A. W. Seabright.

H. E. Nold, Assistant Professor, Department of Mine Engineering, gave an illustrated lecture on several of the mining fields in the United States.

Arrangements are being made for a series of talks on mining and metallurgical subjects to be given during the school year.

ALBERT W. SEABRIGHT, *Secretary.*

DEDICATION OF THE CERAMIC ENGINEERING BUILDING OF THE UNIVERSITY OF ILLINOIS

The new Ceramic Engineering Building of the University of Illinois is to be formally dedicated on Nov. 20 and 21. The occasion will be made one of great interest to the clay-workers of the country. It is expected that the exercises will be attended by many representatives of the architectural, structural, mining, geological, chemical, and manufacturing interests. In connection with the dedication exercises, an industrial conference will be held, in which a number of topics of current interest to the ceramic engineer, the clay-worker and the manufacturer will be discussed by well-known experts.

The number of students registered in the College of Engineering of the University of Illinois eleven days after the beginning of registration for the new year, is as follows:

Architecture, 116; Architectural Engineering, 167; Ceramic Engineering, 46; Civil Engineering, 190; Electrical Engineering, 266; Mechanical Engineering, 273; Mining Engineering, 25; Municipal and Sanitary Engineering, 26; Railway Engineering, 34; total, 1,143.

The enrollment of students in the whole University, not including the College of Medicine, the College of Dentistry and the School of Pharmacy, which are located in Chicago, is 5,214.

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LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS
AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

BLACK DAMP IN MINES. Bull. No. 105, U. S. Bureau of Mines, Washington, 1916.
HEALTH CONSERVATION AT STEEL MILLS. Tech. Paper No. 102, U. S. Bureau of Mines, Washington, 1916.
HEAT TREATMENT OF TOOL STEEL. Ed. 2. By Harry Brearley. New York, 1916.
HISTORY AND DEVELOPMENT OF GOLD DREDGING IN MONTANA. Bull. No. 121, U. S. Bureau of Mines, Washington, 1916.
ORE SAMPLING CONDITIONS IN THE WEST. Tech. Paper No. 86, U. S. Bureau of Mines, Washington, 1916.
SAFE PRACTICE AT BLAST FURNACES. Tech. Paper No. 136, U. S. Bureau of Mines, Washington, 1916.
WORKS OF DMITRI KONSTANTINOVICH TSCHERNOFF, published by the Russian Metallurgical Society (In Russian). Petrograd, 1915. (Gift of D. Tschernoff.)
THE FLOTATION PROCESS. By Herbert A Megraw. McGraw-Hill Book Co., New, York. (Gift of Publishers.) Price $2.50 net.

[NOTE.—In this volume the author aims to present in concentrated form the results of practical and theoretical investigations carried on by many men in many different fields. It represents a compilation of information gathered from many different sources, arranged in a manner to enable those interested to learn the present status of the art of concentrating ores by flotation without wading through a great mass of

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published material. The author has endeavored to eliminate largely that portion of the literature which is purely speculative in character, retaining only the parts that have been definitely proved to be correct, or at least show a reasonable basis for that assumption. In the chapter on Patent Record of Flotation only those patents have been included that seem to be of particular importance. Other chapters cover the following: The Theory of Flotation; Oils and Their Uses; Flotation Processes and Apparatus; Testing Ores for Flotation; Testing at the Anaconda Mill; The Applications of Flotation; Examples of Flotation Practice; Flotation Operating Plants; Flotation Concentration at Anaconda; Flotation in Practice; The Practice of Flotation; The Place of Flotation in Metallurgy. B. A. R.]

THE MINERAL INDUSTRY, ITS STATISTICS, TECHNOLOGY AND TRADE DURING 1915. Edited by G. A. Roush, Vol. 24 supplementing Vols. 1 to 23. McGraw-Hill Book Co., New York. (Gift of Editor.) Price $10 net.

[NOTE.—This publication, the first volume of which appeared in 1893 under the editorship of Richard P. Rothwell, has become so well known to those interested in mining, metallurgy, and allied subjects as to need no extended review. Its value as a record of current events and progress in the field covered is recognized by those who have had occasion to consult the pages of the different volumes.

The present volume shows the status of the mineral industry in 1915 by means of statistics showing prices and production, and by technical articles by specialists on the developments in metallurgical practice in the treatment of the various metals. Flotation, the latest tool of the metallurgist, is treated in the regular reviews and in a special article. B. A. R.]

Geology and Mineral Resources

ANTIMONY DEPOSITS OF ALASKA. Bull. No. 649, U. S. Geological Survey, Washington, 1916.

BIBLIOGRAPHY OF NORTH AMERICAN GEOLOGY FOR 1915. Bull. No. 645, U. S. Geological Survey, Washington, 1916.

CHISANA-WHITE RIVER DISTRICT, ALASKA. Bull. No. 630, U. S. Geological Survey, Washington, 1916.

COAL RESOURCES OF THE CLINTWOOD AND BUCU QUADRANGLES, VIRGINIA. Bull. No. XII, Virginia Geological Survey, Charlottesville, 1916.

ECONOMIC GEOLOGY. Ed. 4. By Heinrich Ries. New York, J. Wiley & Sons, 1916. (Gift of Publishers). Price $4 net.

[NOTE.—The third edition was published in 1910, and advances in the knowledge of economic geology since that date have been so great as to require quite complete revision. In addition, a description of the more important mineral deposits of Canada have been added. The work is notable for the completeness of the lists of references given at the end of each chapter, for the excellence of its numerous illustrations, and for its extensive index. W. P. C.]

LIGNITE FIELD OF NORTHWESTERN SOUTH DAKOTA. Bull. No. 627, U. S. Geological Survey, Washington, 1916.

SUL MODO DI FORMAZIONE DEI GIACIMENTI PETROLIFERI E SOLFIFERI. By P. Toso. Roma, 1916.

SUL MODO DI FORMAZIONE DEI PRINCIPALI GIACIMENTI METALLIFERI. By P. Toso. Roma, 1913.

Mining Law

ABSTRACTS OF CURRENT DECISIONS ON MINES AND MINING, REPORTED FROM JANUARY TO APRIL, 1916. Bull. No. 126, U. S. Bureau of Mines, Washington, 1916.

MINING STATUTES OF NEW MEXICO AND OF THE UNITED STATES, etc., Bull. No. 1, New Mexico State School of Mines, Socorro, 1916.

Highway Engineering

CONSTRUCTION OF ROADS AND PAVEMENTS. By T. R. Agg. New York, 1916.

HANDBOOK FOR HIGHWAY ENGINEERS. Ed. 2. By W. G. Harger & E. A. Bonney, New York, McGraw-Hill Book Co., 1916.

Company Reports

BROKEN HILL PROPRIETARY COMPANY, LTD. Reports and Statements of Account for half year ending May 31, 1916. Melbourne, 1916. (Gift of Company.)

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BROKEN HILL SOUTH SILVER MINING COMPANY. Reports, Statements of Accounts, etc., for half year ended June 30, 1916. Melbourne, 1916. (Gift of Company).

Trade Catalogs

AMERICAN BLOWER Co., Detroit, Mich. Sirocco Service, Vol. 5, No. 11.
AUTOMATIC EQUIPMENT Co., New York City. The Autoprojector.
CHICAGO PNEUMATIC TOOL Co., Chicago, Ill. Bull. E. 43. Duntley Universal Electric hammer drill. Sept. 1, 1916.
COLUMBIAN BRACE FOUNDRY, Freeport, L. I. Propellers in a Nut Shell.
DU PONT DE NEMOURS, E. I. & Co., Wilmington, Del. Du Pont Magazine, September, 1916.
GENERAL ELECTRIC Co., Schenectady, N. Y. Bull. 44004A. Railway Line Material for direct suspension. Aug., 1916.
INGERSOLL-RAND Co., New York City. Form 3033. Imperial type "XPV" Duplex steam driven compressors. Ed. 2. April, 1916.
Form 4122. Leyner drill sharpener. Ed. 2. June, 1916.
Form 8112. "Crown" Pick type 56-H. June, 1916.
Form 9024. Steam condensing plants. Beyer barometric type. June, 1916.
JEFFREY MFG. Co., Columbus, Ohio. Bull. 183. Mine Ventilation.
KOVEN, L. O. & BRO., New York City. Marine Catalogue.
MARINE OIL ENGINE Co., Inc., New York City. Bull. 101. Skandia Marine Oil Engines for crude or fuel oils.
MILBURN, ALEXANDER Co., Baltimore, Md. Oxy-Acetylene Welding and Cutting Apparatus.
MOTOR COMPRESSOR Co., Newark, N. J. The Perfect Starter.
NATIONAL TRANSIT PUMP & MACHINE Co., Oil City, Pa. Bull. 107. Steam pumps. Bull. 201. Horizontal power pumps.
PYRENE MANUFACTURING Co., New York City. Ten reasons why Pyrene is superior to the three gallon chemical extinguisher.
Pyrene Extinguisher for motion picture theatres.
SAMSON ELECTRIC Co., Canton, Mass. "Perfex" waterproof ignition for motor boats. 1916.
SNOW AND PETRELLI MFG. Co., New Haven, Conn. Joe's high power reverse gears. 1916.
THE SPERRY GYROSCOPE Co., Brooklyn, N. Y. Neptune foiled by the Sperry ship's stabilizer.
TALBOT BOILER Co., New York City. Talbot crude oil burning steamers.
TRAYLOR ENGINEERING & MFG. Co., Allentown, Pa. Bull. G3. Gyratory crushers. 1916. Bull. J1. Jaw crushers. 1916.
WHEELER & SCHEBLER, Indianapolis, Ind. The Model "R" Schebler carburetor.

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MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Sept. 10, 1916, to Oct. 10, 1916.

BLOOMFIELD, EDWIN CHARLES, Min. Engr., 1012 Standard Bank Bldg., Vancouver, B. C., Canada.
BRINGS, HENRY, Min. and Met. Engr., Chief Engr., and Geol., Mollet-Mesrine, 1212 Mills Bldg., El Paso, Texas.
CHASE, MARCH F., Vice-Pres., Commercial Acid Co., 1906 Boatmens Bank Bldg., St. Louis, Mo.
CUMMINGS, ALVIN M., Asst. Mine Supt., Witherbee, Sherman & Co., Inc., Mineville, Essex Co., N. Y.
DOUGLASS, GEORGE M., Genl. Safety Inspector, Amer. Smelt. & Ref. Co., 120 Broadway, New York, N. Y.
EMRICH, CLARENCE T., Chief Chemist, Old Dominion Copper Min. & Smelt. Co., Box 1163, Globe, Ariz.
GARDANIER, SUTTER ALBERT, Assayer, Mgr., Douglas Assay Co., Douglas, Cochise Co., Ariz.
GRISWOLD, CHARLES E., Concentration Sampling, Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
GULICK, HORACE S., Met. Engr., More-Jones Brass & Metal Co., 3132 North Broadway, St. Louis, Mo.
HASBRONCK, BERNARD, Petroleum Geol. and Engr., The Carter Oil Co., 708 So. Boulder Ave., Tulsa, Okla.
HEIMROD, ALBERT A., Met. Engr., Research Corpn., 63 Wall St., New York, N. Y.
HOLMAN, C. VEY, Mining Lawyer & Geol., Prest., Caribou Gold Mines, Holman Oaks, Rockland, Me.
HUMPHREY, THOMAS ZENAS, Washoe Reduction Works, Anaconda Copper Min. Co., Anaconda, Mont.
INGLIS, WILLIAM WALLACE, Genl. Mgr., Coal Mining Dept., Delaware, Lackawanna and Western R. R. Co., Scranton, Pa.
ISHIKAWA, TOMEKICHI, Min. Engr., Kubara Min. Co., Saikokwa Hitachi Mine, Ibarakiken, Japan.
JARVIS, PHILLIPS F., Mgr., Sullivan Machinery Co., 2006 Railway Exchange Bldg., St. Louis, Mo.
JENKINS, CHARLES VERNON, Business Mgr., Nevada Cons. Copper Co., McGill, Nev.
JOHNSTON, IRVING H., Dist. Mgr., Pittsburgh Testing Laboratory, 58 Lafayette Blvd., Detroit, Mich.
KERFE, WILLIAM HENRY, JR., Engr.....Chase Metal Works, Waterville, Conn.
LE ROUX, GEORGE N., General Sales Agent, The Manhattan Rubber Mfg. Co., 309 Felt Bldg., Salt Lake City, Utah.
LONG, LEON ROOT, Assayer.....Stoddard Min. & Mill. Co., Stoddard, Ariz.
LYNTON, EDWARD D., Min. Engr., Mammoth Copper Mining Co., Kennett, Shasta Co., Cal.
MCCOY, WILLIAM ARTHUR.....Chief Chemist, Nevada Cons. Copper Co., Box 752, McGill, Nev.
MARSH, THOMAS A., Chief Engr.....Green Engineering Co., East Chicago, Ind.
MAYNE, RICHARD W., Genl. Foreman, Old Dominion Cons. Min. & Smelt. Co., Globe, Ariz.
MITCHELL, CLARK G.....P. O. Box 569, Victor, Colo.
MOLINA, OLEGARIO, Retired....Linea No. 57, entre A y B, Vedado, Havana, Cuba.
MONTGOMERY, WILLIAM BRUCE, JR., Engr. of Mines, Grasselli Chemical Co., Cleveland, O.
MOORE, JONES GUERRY, Genl. Supt. of Coal Mines, Sloss Sheffield Steel Co., 923 So. 17th St., Birmingham, Ala.
PERKINS, ENOCH, Surveyor and Draftsman, Alaska Gastineau Min. Co., P. O. Box 114, Juneau, Alaska.
POWELL, GUY M. Supt. of Coal Mines....Central Iron and Coal Co., Holt, Ala.
RAIT, DONALD MCKINNON, Chief Engr., Calumet & Arizona Min. Co., Warren, Ariz.

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RICH, CHARLES HENDERSON, Met. and Chemist, Alan Wood Iron and Steel Co., Conshohocken, Pa.
RISER, GEORGE C., JR., Supt. of Concentrator, Nevada Cons. Copper Co., McGill, Nev.
SMETTEM, BLAKE WILLIAM, Min. Engr.....Belknap Coal Co., Newcastle, Texas.
STEWART, JOHN S., Supt. of Lead Smelter and Refining, Irtysh Corpn., Care Kirgis Mining Co., Prov. of Semipalatinsk, Pavlodar, Ekibastus, Siberia.
STOCK, KURT, Met. Engr.....Bartlesville Zinc Co., Bartlesville, Okla.
STONE, SIDNEY MARVIN, Mgr. of Mine and Mill, Chas. Butters & Co., Ltd., Virginia City, Nev.
TALMAGE, RANDOLPH D., Min. Engr.....Cunningham Bldg., Joplin, Mo.
THOMPSON, JOHN FAIRFIELD, Supt., Monel Metal Dept., The International Nickel Co., Bayonne, N. J.
WILLIAMSON, FRANCIS O.....545 Old Colony Bldg., Chicago, Ill.
WILLSON, HAROLD EDWIN.....407 W. Illinois St., Urbana, Ill.
WILSON, EDWARD TAYLOR, Pres., The Continental Oil Co., 621 McPhee Bldg., Denver, Colo.
WINGFIELD, GEORGE..... Pres., Goldfield Cons. Mines Co., Reno, Nev.
WOO, ZUNG TSE KIEN, Genl. Supt., Han-Yeh-Ping Iron and Coal Co., Ltd., Hanyang Iron & Steel Works, Hanyang, China.

Associate Members

COLEMAN, WALTER H., Millman.....Big Pine Cons. Min. Co., Prescott, Ariz.
GAGE, WILLIAM PAUL, First Vice-Pres. and Genl. Supt., Lone Star Gas Co., 800-1st. National Bank Bldg., Fort Worth, Tex.
PLATT, JAMES MCCLURE, Min. Engr.....Care W. W. Platt, Alamosa, Colo.
SOWERWINE, ELBERT O., Accountant, Chief Clerk, International Smelt. Co., Tooele, Utah.

Junior Members

ARMSTRONG, JAMES A. B.....347 Manhattan Ave., New York, N.Y.
BRUNEL, LOUIS J., Engineering Staff, Nevada Cons. Copper Co., P. O. Box 362, Ruth, Nevada.
KLINE, HARRY D., Mine Sampler, Nevada Cons. Copper Co., P. O. Box 362, Ruth, Nevada.
LEE, HUNYET, Post Graduate Student.....Columbia Univ., New York, N. Y.
MCCRAY, H. EDGAR.....Elks Home, Denver, Colo.
MILLER, WILLIAM B., Engr., Boundary Red Mountain Min. Co., Sardis, B. C., Can.
MINISTER, HOWARD LESLIE, Min. Engr., The Empire Zinc Co., Kelly, New Mexico.
POND, WALTER FRANKLIN, Student, Mass. Institute of Technology, Boston, Mass.
ROPER, GEORGE, JR.....Student, Mass. Institute of Technology, Boston, Mass.
SULLY, KENNETH M., Assayer, Hanover Bessemer Iron & Copper Co., Fierro, New Mexico.
TRAVER, WILL MERTON, JR., Engr., Empire Zinc Co., 703 Symes Bldg., Denver, Colo.

Total Membership, Oct. 10, 1916..... 5,800.

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period Sept. 10, 1916, to Oct. 10, 1916, for election as members of the Institute. Their

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names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Harvey S. Adkins, Hazard, Ky.
Proposed by Ivor Livingston, Walter B. Miller, Henry Groos.
Born 1890, Wrigley, Ky. To 1907, Preparatory education. Three years' work on engineering party, and correspondence course while doing it. 1911–12, Kentucky State Univ. 1907–09, Morehead and North Fork R. R. Co. 1910, North Fork Commel Coal Co. 1913, North Fork Commel Coal Co. and The Lee Coal Co. 1914–15, Chief Engr., Blue Grass Coal Corpn.; Daniel Boone Coal Co.; Kentucky-Jewel Coal Co.; Harvey Coal Co.
Present position: Same as above.

Philip Henry Argall, Denver, Colo.
Proposed by Burr A. Robinson, Charles A. Chase, D. W. Brunton.
Born 1877, Oroca, Ireland. 1903, Univ. of Colorado, B. S. 1904, Univ. of Colorado, M. A. 1904, Chemist, Anaconda Copper Min. Co. 1904–06, Chemist, Selby Smelt. & Lead Co. 1906–08, Asst. Supt., Selby Smelt. & Lead Co. 1909–15, Resident Mgr., Stratton's Independence, Ltd., Victor, Colo. 1904, Author, *Western Mill and Smelter Methods of Analysis*. 1913, Member, Institution of Mining and Metallurgy, London.
Present position: 1910 to date; Cons. Mining and Met. Engr., Philip Argall & Sons.

David Avery, Melbourne, Aust.
Proposed by Guy C. Riddell, C. M. Warner, Gilbert Rigg.
Born 1871, Ballarat, Vic. 1888–95, Melbourne Univ., M. Sc., Chemistry and Physics with special attention to Metallurgy. 1895–98, Engaged in cyanide work on tailings for Melbourne Syndicate. 1898–1910, in charge of Chemistry Dept., Melbourne Technical School. Designed and equipped chemical laboratories.
Present position: 1910 to date; Cons. Chemist and Met., specialising in treatment of lead ores including flotation work, by-products and plant economics.

Marc Bailey, Bisbee, Ariz.
Proposed by Arthur Notman, Y. S. Bonillas, G. F. G. Sherman.
Born 1882, Phanute, Kans. Parsons Business College. 1900–05, Colorado School of Mines, Kansas Univ., Lehigh Univ., Michigan College of Mines; B. S. and E. M. at latter. 1905, Surveyor, Santa Fe R. R. 1906, Engr., Kansas Natural Gas Co. 1907, Engr., Dolores Mines Co. 1908, Mgr., Aurora and Sirena Mines. 1909, Director and in office City Investment Co. 1910, Examining Engr., U. S. S. R. & M. Co. 1912, Draftsman, Repath & McGregor. 1913, Chief Engr., Shattuck Arizona Copper Co.
Present position: Supt., Wolverine & Arizona Min. Co.

David Franklin Baker, Hamburg, N. Y.
Proposed by C. G. Atwater, L. O. Crewe, R. H. Sweetser.
Born 1889, Sparrows Point, Md. 1913, Mass. Inst. of Technology, B. S. 1913, Youngstown Sheet & Tube Co., Youngstown, O. 1914–16, Charge of all concreting of foundations for steel works, Broken Hill Proprietary Co., Ltd., Newcastle, N. S. W., Aust. Asst. to Supt. of Blast Furnace during construction of Blast Furnace. Shift Genl. Foreman on Blast Furnace.
Present position: Foreman, Blast Furnace Dept., Lackawanna Steel Co.

Bennett R. Bates, Chicago, Ill.
Proposed by H. L. Hollis, D. P. Hynes, Francis G. Fabian.
Born 1884, Brighton, Colo. 1908, Univ. of Cal., College of Mining, B. S. 1908–10, Engr., Jesus Maria y Anexas, San Jose de Gracia, Sinaloa, Mex. 1910–12, Supt.,

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Rosario Extension Syndicate, Ltd., Guadalupe y Colvo, Chih., Mex. 1912–13, Auditor and Engr., El Rayo Min. & Dev. Co., Santa Barbara, Chih., Mex. 1913–14, Special work, F. W. Bradley, San Francisco and Treadwell, Alaska.
Present position: 1914 to date; Staff, H. L. Hollis.

Leo Richard Bell, La Paz, Bolivia, So. Amer.
Proposed by Francis Church Lincoln, J. G. Scrugham, J. C. Jones.
Born 1891, New York, N. Y. 1899–1906, Grammar School No. 14, Richmond, N. Y. 1906–10, Curtis High School, Richmond, N. Y. 1910–14, Univ. of Illinois, Champaign, Ill., B. S. in Min. Engrg. 1912–14, Calumet & Hecla Min. Co., Bisbee, Ariz. 1913, Doe Run Lead Co., Flat River, Mo. 1914, Ray Cons., Ray, Ariz. 1914–16, Incaoro Mines Co., Bolivia and New York.
Present position: South American Electric Smelt. Co. Syndicate.

Floyd Shaw Benjamin, Duluth, Minn.
Proposed by Stacy H. Hill, Edwin J. Collins, W. G. LaRue.
Born 1887, Romeo, Mich. 1905, Romeo High School. 1906, Univ. of Michigan. 1908–09, Michigan College of Mines. 1910–11, Min. Engr., Duluth Engrg. Co., Duluth, and Min. Engr., Inland Steel Co., Crosby, Minn. 1912–16, Technical, Pluto Powder Co.
Present position: District Mgr., Aetna Explosives Co., Inc.

George Howard Birch, New York, N. Y.
Proposed by George M. Esterly, C. N. Crary, George C. Hazelet.
Born 1882, Peekskill, N. Y. Until 1897, Local Public and Private Schools. 1897–1906, engaged in banking business for the larger part of which time was in the employment of Messrs. Strong, Sturgis Co., New York. 1906–09, Columbia Univ., School of Mines as a special student. 1906, Asst. with Herman A. Keller, on examination work in Alaska. 1909, Asst. to Fred L. Morris, Guggenheim Explor. Co., on examination work in North Carolina. 1909, Trip of inspection of mines throughout western part U. S. and Canada. 1909–10, Nevada Cons. Min. Co., McGill, Nev. 1910, Placer examination work, Guggenheim Explor. Co., Siberia. 1910, took charge of construction work and operations of hydraulic plant, Supt. and Mgr., Dan Creek Min. Co., Dan Creek, Alaska.
Present position: Vice-Pres. and Genl. Mgr., Dan Creek Min. Co.

Elmer Bird, Wallace, Idaho.
Proposed by Rush J. White, William J. Hall, S. A. Crandall.
Born 1888, Mansfield, Ill. 1900, Grad., Billing High School. 1904, Student, Missouri Univ. 1907, Grad., College of Idaho, no science degree. 1900–05, summers and odd times, survey helper and levelman. 1905–07, summers, Transitman. 1907–09, Irrigation work, U. S. Rec. Service. 1909–10, Chief Engr., Bloine County Irri. Co. 1910–14, City Engineer, Consulting Engr., engaged in private practice and mining engineering. 1914–16, Engaged in private practice and mining engineering at Boise, Idaho. 1914–16, specializing in concrete underground. 1916, Asst. Engr., Bunker Hill Smelter Construction.
Present position: Asst. at Bunker Hill and Sullivan Smelter, Kellogg, Idaho.

Lionel E. Booth, Anaconda, Mont.
Proposed by Albert E. Wiggin, W. N. Tanner, Arthur E. Wells.
Born 1892, Ashland, Wis. 1910–11, Preparatory course, So. Dakota School of Mines. 1911–15, South Dakota School of Mines, Rapid City, South Dakota, B. S. in Met. Engrg. 1916, Solution man, St. Paul Montana Min. Co., Maiden, Mont. 1916, Testing Dept., Anaconda Copper Min. Co.
Present position: Met. Engr., Concentration Dept., Anaconda Copper Min. Co.

Milton P. Botsford, Duluth, Minn.
Proposed by Stacy H. Hill, Edwin J. Collins, W. G. LaRue.
Born 1885, Hancock, Mich. 1907, Michigan College of Mines, Houghton, Mich., E. M. and B. S. 1904–08, Min. Engr. and Foreman, Pickands Mather Co. 1909, Min. Engr., Oliver Iron Min. Co. 1910, Drilling and exploration on the Cuyuna Range, Paul D. Willard & Co. 1912–15, Drilling and exploration. Reporting on mining properties, Aitkin Exploration Co.
Present position: Technical man, Aetna Explosive Co., Inc.

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Joseph Markley Bradley, Florence, Colo.
Proposed by L. V. Emanuel, H. H. Utley, Charles W. Comstock.
Born 1876, St. Louis, Mo. 1901, Grad., Colorado School of Mines, E. M. 1905–06, Surveying at Ely, Nev. 1907, Engr., Austin Manhattan Min. Co., Austin, Nev. 1911, Asst. Supt., May Day Gold Min. Co.
Present position: Supt., Shenandoah-Dives Mines.

Guillermo Brockmann, New York, N. Y.
Proposed by E. Girault, Willard S. Morse, H. R. MacMichael.
Born 1870, Guanajuato, Rep. of Mexico. Mining School of Guanajuato and Government Schools of Mexico City. 1894, Organizer of Cia. Minera Esperanza y Anexas (Esperanza Min. Co.); Organizer of Compania Minera Las Dos Estrellas, S. A. of El Oro, Mex., of which I was the Treasurer and a member of the Board of Directors in the City of Mexico until 1914, when I left the City of Mexico to establish myself in business in New York City. Organizer of Société d'Affinage des Métaux (Sociedad Afinadora de Metales) with French and Mexican capital for the establishment of a refining plant in the City of Mexico. Member of the Board of Directors of this company since its incorporation. Organizer and controlling partner of Compañia Minera Real de Arriba, Temascaltepec, Mex. Connected in different capacities with several other mining concerns in Pachuca, Lower California, Sonora and Jalisco States, Mexico.
Present position: Partner in the banking firm of G. Brockmann & Co.

Edmund Ernest Campbell, Anyox, B. C., Can.
Proposed by A. J. Bone, William J. Hamilton, Wade L. Wetmore.
Born 1880, Belmont, P. E. I. 1898, Country school. 1898–1901, Prince of Wales College, Charlottetown, P. E. I. 1904–08, McGill Univ., Montreal, B. Sc. in Min. Engrg. 1911, McGill Univ., M. Sc. in Geology, Mineralogy and Ore Deposits. 1901–04, Coal miner, Crows Nest Pass, B. C. 1908–10, varied experience in Cobalt and Gowganda, Ontario; Lead, S. D.; Phoenix, B. C. 1910–11, Engr., Granby Mines, Phoenix, B. C. 1911–14, Field Engr., Granby Cons. Min. Smelt & Power Co., Ltd. 1914–15, Supt. Granby Co., Valdez, Alaska.
Present position: 1915 to date; Supt. of mining operations in Anyox District.

Ralph R. Carter, Audenried, Carbon Co., Pa.
Proposed by C. F. Huber, Douglas Bunting, Walter Fahringer.
Born 1878, Mauch Chunk. Lawrenceville. 1898, West Jersey Academy. 1902, Lafayette College. 1902, Chainman, Noteman and Transitman, Lehigh and Wilkes-Barre Coal Co., Wyoming Division. 1907, Transitman, Division Engr., Lehigh and Wilkes-Barre Coal Co., Honey Brook Division.
Present position; Division Engr., Lehigh and Wilkes-Barre Coal Co.

Herbert Edward Collbran, Denver, Colo.
Proposed by D. W. Brunton, Irving T. Snyder, H. S. Munroe.
Born 1879, London, England. 1886–88, Ohio Public Schools. 1889–95, Colorado Public Schools. 1895–98, Jarvis Hall Military Academy, Denver. 1901–02, Univ. of Chicago. 1899–1901, Assayer, Punjom Mines, Pahang, Straits Settlements. Asst. Mgr., Kechau Gold Mines, Pahang, Straits Settlements. 1902–10, Seoul, Korea Railway and General Contracting with Collbran-Bostwick Dev. Co.
Present position: 1910 to date; Secretary and Treasurer, Seoul Min. Co. and Collbran Bostwick Dev. Co., operating mines in Korea and Mexico.

William H. Cooper, El Paso, Tex.
Proposed by Walter M. Drury, David Cole, R. F. Manahan.
Born 1878, Meaford, Ont., Can. 1886–96, Grammar and High School, Meaford and Stratford, Ont. 1902–04, Student, International Correspondence Schools, Mining Course. 1898–1901, Millman, Detroit Copper Min. Co., Morenci, Ariz. 1901–03, Millman, Montezuma Copper Co., Nacozari, Son., Mex. 1903–07, Mill Foreman, Green Cananea Copper Co., Cananea, Son., Mex.
Present position: 1907 to date; Milling Supt., Minas Tecolotes y Anexas.

Alfred Nicks Detweiler, Springfield, Ill.
Proposed by Otto Rissmann, H. A. Wheeler, H. A. Buehler.
Born 1884, Lebanon, Mo. Common Schools. Normal Schools in Missouri. Business College education. 1906–10, Missouri School of Mines, Rolla, Mo., B. S. in Min. Engrg. 1910–11, Chemist and Yard Foreman, Altoona Zinc Co., Altoona,

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Kans. 1912, Chemist, Lanyan-Starr Smelt. Co., Bartlesville, Okla.; National Zinc Co., Bartlesville, Okla.; three different times for short periods, twice as Chemist and last as Asst. Supt.
Present position: 1913 to date; Supt., National Zinc Co.

Percy Williams Donovan, Minneapolis, Minn.
Proposed by E. J. Longyear, John E. Hodge, Francis W. Collins.
Born 1879, St. Charles, Minn. 1898–99, Minnesota School of Mines. 1904–05, Grad., Columbia School of Mines, M. E. 1899–1900, Assayer and Chemist, National Gold Red. Co., Florence, Colo. 1901–03, Foreman, Precipitation and Ref. Depts., Union Plant, U. S. Red. & Ref. Co., Florence, Colo. 1905, Engr., E. J. Longyear, Hibbing, Minn. 1906–16, Supt. of exploration on Cuyuna Range, Brainerd, Minn., for E. J. Longyear and E. J. Longyear Co.
Present position: Supt. of Contract Drilling, E. J. Longyear Co.

Boyd Dudley, Jr., State College, Pa.
Proposed by H. O. Hofman, Carle R. Hayward, W. R. Crane.
Born 1887, Hamilton, Mo. 1908, Missouri School of Mines, B. S. 1910, Missouri School of Mines, M. S. 1912, Mass. Inst. of Technology, M. S. 1915, Missouri School of Mines, Met. E. 1908–11, Instructor in Metallurgy and Ore Dressing, Missouri School of Mines. 1911–12, Grad. Student, Mass. Inst. of Technology. 1912–13, Asst. Supt., Asbestos Shingle Co., Nashua, N. H.
Present position: Asst. Prof. of Met., Pennsylvania State College.

Paul Emile Dulieux, New York, N. Y.
Proposed by E. Saladin, John B. Porter, Frank D. Adams.
Born 1881, St. Etienne, France. Complete college education in Lyons and Paris, France. 1901–03, Student (after an entrance competition) at Polytechnic School of Paris. 1904–07, Grad., High School of Mines of Paris. 1907–14, Prof. of Min. Engrg., Laval Polytechnic School, Montreal, Canada. Provincial analyst to the Quebec Bureau of Mines, Cons. Min. Engr.
Present position: Lieut. in the French Army, with the French Commission in New York.

John Somerville Dymock, Warren, Ariz.
Proposed by Alexander V. Dye, Ira B. Joralemon, Leon Feuchère.
Born 1886, Calumet, Mich. 1906, Grad., Calumet High School. 1913, Grad., Pittsburgh School of Mines, Univ. of Pittsburgh.
Present position: 1913 to date; Min. Engr., Calumet & Arizona Min. Co.

Ira William Farrand, Butte, Mont.
Proposed by J. H. Warner, G. G. Vivian, Oscar Rohn.
Born 1889, Ireton, Iowa. 1910–13, Univ. of Wisconsin. 1913–14, Anaconda Copper Co. (mines). 1914–15, Asst. Min. Engr., East Butte Copper Min. Co.
Present position: 1915 to date; Shift boss, East Butte Copper Min. Co.

Naximo Macambyra Monte Flores, Bahia, Brasil, So. Amer.
Proposed by F. Lynwood Garrison, H. M. Chance, R. H. Sanders.
Born 1889, Bahia, Brasil. 1910, Graduate in Railroad Engineering, Escola de Engenharia, Rio de Janeiro, Brasil. 1913, Graduate in Mining Engineering, I. C. Schools, Scranton, Pa. 1910, Employed as Surveyor in railroad location and exploratory work, Brazilian Federal Government. 1914, Examining prospects for the Empresa Bahirna de Mineracas (Bahia Min. Co.). 1916, American Consulate Office. Few days in the hinterland of Bahia as an assistant to Mr. Garrison.
Present position: Examining prospects.

Stuart W. French, Douglas, Ariz.
Proposed by Grant H. Dowell, Forest Rutherford, H. O. Hammond.
Born 1867, Dansville, N. Y. 1889, Amherst College, B. S. 1900–06, Asst. Supt., Copper Queen Cons. Min. Co. 1906–10, Asst. Genl. Mgr., Copper Queen Cons. Min. Co. 1910–16, Genl. Mgr., Copper Queen Cons. Min. Co.
Present position: Genl. Mgr., Phelps, Dodge & Co., Inc.

Thomas Herndon Garnett, Tucson, Ariz.
Proposed by R. P. Paul, Cony T. Brown, S. Ringlund.
Born 1885, Sharon, Mo. 1905, High School, Pueblo, Colo. 1911, Grad., Colo-

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rado School of Mines, M. E. 1911–12, City Engineer's Office, Butte, Mont. 1912 to present, Engr., Empire Zinc Co.; Supt., San Xavier Mine of Empire Zinc Co.
Present position: Supt., San Xavier Mine.

Anton Martin Gronningsater, Kristiansand S., Norway.
Proposed by V. Hybinette, R. W. Deacon, E. A. C. Smith.
Born 1880, Hjirandfjord, Norway. 1896–1900, Grad., Technical School in Trondhjem, Norway, Chem. Engr. 1900–04, Asst. in Chemical Laboratory, same school. 1903, Studied at the Royal Mining Academy, Freiberg, Saxony. 1905, Employed in smelting works in Canada. 1909, Smelter Supt., Refinery Supt., and Genl. Mgr., Eoji Nikkilverk and Kristiansands Nikkelraffineringsverk, Kristiansand, Norway.
Present position: Genl. Mgr., Kristiansands Nikkelraffineringsverk.

J. Glenn Haigh, Silver City, N. Mex.
Proposed by S. Ringlund, R. B. Paul, Cony T. Brown.
Born 1884, Fairview, Kans. 1906, Ottawa Univ., Ottawa, Kans., B. Sc. 1906 to date, Empire Zinc Co. Two years' office work in Denver and Cañon City. Two years Asst. Chemist, and four years Chief Chemist at Cañon City, Colo. Two years' mill work at Cañon City, and at the Cleveland Mine, Silver City, N. Mex.
Present position: Mill Supt., Cleveland Mine.

William S. Hall, Miami, Ariz.
Proposed by C. E. Mills, Rudolf Gahl, Guy H. Ruggles.
Born 1886, Winfield, Kans. 1905–09, Missouri School of Mines and Metallurgy, B. S. in E. M. Entered college from Kansas City Manual Training High School, Kansas City, Mo. 1909–12, Met., Mill Supt., cyanide work, U. S. Smelt. Ref. & Min. Co., Pachuca, Hgo., Mex. 1912, Shift boss, Hollinger Mill, Timmins, Ont., Canada. 1912–13, Designing Engr., Res. Mgr., St. Anthony Mine, Sturgeon Lake, Ont., Canada. 1913–15, Shift boss, No. 2 Mine, Ray Cons. Copper Co., Ray, Ariz.
Present position: Flotation Operator, Inspiration Cons. Copper Co.

Ray Weston Hart, Cananea, Son., Mex.
Proposed by Morris J. Elsing, Walter A. Richelsen, George Kingdon.
Born 1885, Minden, Nebr. 1907–10, Univ. of Wisconsin. 1911–12, Univ. of Wisconsin, C. E. 1914–16, graduate work in geology, Univ. of Wisconsin. 1910–11, Civilian Engr., Corps of Engineers, U. S. A. 1912–14, Civilian Engr., Corps of Engineers, U. S. A. 1916, Geol., Cananea Cons. Copper Co., Cananea, Son., Mex.
Present position: Geol., Cananea Cons. Copper Co.

Terrell Gaines Hawkins, Jr., Hillsboro, Tex.
Proposed by A. J. McQuatters, W. J. Deavitt, William D. Gordon.
Born 1885, Hillsboro, Tex. 1891–98, Public Schools, Hillsboro, Tex. 1899–1902, Private Schools, Hillsboro, Tex. 1903–04, Carlisle Military Institute, Arlington, Tex. 1905–06, Texas State Univ., Austin, Tex. 1907, Sturgis National Bank, Hillsboro, Tex. 1908–12, Mine Surveyor and Asst. Mine Supt., The Alvarado Min. & Mill. Co., Parral, Chih., Mex. 1912–14, Supt. of Mines, The Alvarado Min. & Mill. Co., Paral, Chih., Mex.
Present position: 1914 to date; Asst. Mgr., The Alvarado Min. & Mill. Co., Parral, Chih., Mexico.

Harry Duff Hunt, Miami, Ariz.
Proposed by B. B. Gottsberger, Frederick W. Solomon, R. B. Yerxa.
Born 1886, Baltimore, Md. 1900–04, Baltimore Polytechnic Institute. 1905–09, Colorado School of Mines, E. M. 1906, summer, Baltimore Copper S. & R. Co., Baltimore, Md. 1907–08, Baltimore Copper S. & R. Co., Baltimore, Md. 1909–11, Millman, Utah Copper Co. 1910–11, Chemist, Garfield Smelt. Co. 1911–15, Chief Chem., Miami Copper Co.
Present position: Research Engr., Miami Copper Co.

Harry G. Hunter, Corinth, Vt.
Proposed by John H. Allen, H. H. Knox, William H. Shearman.
Born 1881, Andalusia, Pa. 1898, Manual Training High School, Philadelphia. 1899, Banks Business College. 1900–02, Night courses mechanical drawing and engineering in evening high school and Franklin Institute. 1899–1905, Pencoyd Plant, American Bridge Co., Mechanical and Yards and Buildings Dept. 1905–06,

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Knox and Allen, New York. 1906–07, Supt., Pike Hill Mines. 1907–09, Knox and Allen, New York. 1909–11, Mech. Engr., American Felt Co. 1911–15, Member firm M. W. Allen Construction Co., Walpole.
Present position: 1915 to date; Mgr., Pike Hill Mines.

Hermann Kildal, Cripple Creek, Colo.
Proposed by John D. Hawkins, Fred W. Crosley, G. A. Swanquist.
Born 1874, Norway. Grad., Christiania Technical School, C. E., Norway. Last 10 years general mining engineering. Held commission U. S. Min. Surveyor. At times engaged in mining.
Present position: Min. Engr.

Oscar Lee, Anaconda, Mont.
Proposed by C. R. Wraith, Charles D. Demond, Enoch A. Barnard.
Born 1892, Minneapolis, Minn. 1916, Grad., Minnesota School of Mines, E. M. 1913, two months, Sampler, Schenango Mine, Chisholm, Minn. 1914, on valuation survey work, Great Northern Ry. Co. 1915, summer work at Washoe Smelter, Anaconda, Mont.
Present position: Testing Dept., Washoe Smelter, Anaconda Copper Min. Co.

William Wells Logan, Ely, Nev.
Proposed by E. E. Carpenter, E. S. Cunningham, H. M. Alley.
Born 1881, Banff, Scotland. 1899–1901, Oakland High School, Oakland, Cal. 1902–07, Univ. of Cal., B. S. 1907–10, Field man for the City of Tucson, Ariz.; Deputy County Surveyor, for the county of Pima, Ariz.; Field man for the engineering firm of Dietrich & Goetz. 1910–11, Custom surveying. 1911–13, Min. Engr. and Asst. Supt., Atlas Wonder Min. Co., Wonder, Nev. 1913–16, Millman, Churchill Mill. Co., Wonder, Nev.
Present position: 1916 to date; Asst. Supt., U. S. Tungsten Corpn.

E. Lynden Longmore, Timmins, Ont., Canada.
Proposed by P. A. Robbins, A. R. Glove, L. B. Eames.
Born 1884, Ernesttown. 1901, matriculated after regular public and high school course. 1912, Queens Univ., Kingston, Ont., B. Sc. in Min. Engrg. 1911, Asst. Engr., Bankhead Coal Mines, Ltd., Bankhead, Alta. 1912, Asst. to R. G. McConnel, Geol. Survey of Canada.
Present position: 1912 to date; Chemist, and general metallurgical work, Hollinger Cons. Gold Mines.

Clyde Stanley Longyear, Minneapolis, Minn.
Proposed by E. J. Longyear, John E. Hodge, E. C. Harder.
Born 1891, Duluth, Minn. 1911–13, Williams College. 1914–15, Univ. of Wisconsin. 1912, summer, Field Geology, E. J. Longyear Co. 1913, summer, Field Geology, E. J. Longyear Co. 1914, Manufacturing Dept., E. J. Longyear Co. 1916, Geol., E. J. Longyear Co., Sudbury, Ont.
Present position: E. J. Longyear Co.

James Osborn Lord, Gary, Ind.
Proposed by Harry E. Nold, Franklin A. Ray, George H. Morse.
Born 1891, Columbus, O. 1909, North High School, Columbus, O. 1915, Ohio State Univ., B. of Chemical Engrg.
Present position: 1915 to date; Met. Inspector, Illinois Steel Co.

E. H. Lundquist, Superior, Ariz.
Proposed by J. M. Callow, W. H. Aldridge, W. C. Browning.
Born 1874, Salt Lake City, Utah. 1894, Grad., Public Schools, Salt Lake City, Utah. 1899–1900, Special course, Univ. of Utah. 1900–01, Assay offices, Salt Lake City. 1901–02, Charge of Sheba Mine, Humbolt, Nev. 1902–08, Chemical work and charge of development work, Utah Cons. Mine, Bingham, Utah. 1908–12, Supt. and Mgr., South Utah Mines and Smelters, Beaver Co., Utah. 1912–14, Asst. Supt., Inspiration Cons. Co., Miami, Ariz.
Present position: 1914 to date; Supt., Magma Copper Co.

Bernard M. McAtee, Miami, Ariz.
Proposed by Guy H. Ruggles, C. E. Mills, Rudolf Gahl.
Born 1879, St. Louis, Mo. 1888–98, Grammar and High Schools, Denver, Colo. 1899–1903, Univ. of Cal., Min. Dept., B. S. 1903, U. S. Coast & Geodetic Survey, Alaska. 1903–04, Surveyor, Bay Cities Water Co., San Jose, Cal. 1904–06, Amal-

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gamator, Gold King Mines, Silver City, Colo. 1906–08, Mill night foreman, Gold Prince Mines, Silver City, Colo. 1908–10, Mill shift foreman, Florence Mine, Goldfield, Nev. 1910–12, Mill Supt., Arizona Parral Co., Parral, Mex. 1912–13, Mine Supt., Penoles Min. Co., Santa Barbara, Mex. 1913–14, Mill operator, Detroit Copper Co., Morenci, Ariz. 1914–15, Flotation operator, Inspiration Cons. Copper Co.

Present position: 1915 to date; Flotation Foreman, Inspiration Cons. Copper Co.

James Homer McCormick, Marysville, Mont.

Proposed by C. W. Goodale, George T. McGee, Rudolph Emmel.

Born 1871, Adams Co., Ill. 1885–88, High School, Princeton, Ill. 1902–03, Scranton Correspondence School. 1894, Steam Engr., Salt Lake Copper Co. 1895–96, Mill Foreman, U. S. Economic Red. Co. 1897, Mill Foreman, Holy Cross M. & M. Co. 1898–99, Mill Foreman, Arequa Red. Works. 1900–01, Mill Foreman, Central Montana M. Co. 1902, Mine Foreman, Barnes King Dev. Co. 1903–05, Supt., Gold Reef M. Co. 1905–06, Mill Foreman, Crooked River M. & M. Co. 1907–08, Mill Foreman, Skidoo Mines Co. 1909–10, Mill Foreman, Barnes King Dev. Co. 1911–12, Maginnis Lease, Maiden, Mont. 1914–15, Supt., Barnes King Dev. Co.

Present position: Supt., Piegan-Gloster Property, Barnes King Dev. Co.

Felix McDonald, Miami, Ariz.

Proposed by C. E. Arnold, George R. Lehman, C. E. Mills.

Born 1875, Dalton-in-Furness, Lancashire, England. Public school education in Lancashire, England, and in Wisconsin. 1889–1903, Mining and bossing in Michigan, Wisconsin and Utah. 1903–06, Mine Supt., Bingham Copper Gold Min. Co., Bingham, Utah. 1906–13, Mine Supt., Ohio Copper Co., Bingham, Utah.

Present position: 1913 to date; Mine Supt., Inspiration Cons. Copper Co.

George Mieyr, Bisbee, Ariz.

Proposed by G. F. G. Sherman, Arthur Notman, Charles A. Mitke.

Born 1870, Calumet, Mich. 1884–90, Apprentice and Machinist, Calumet & Hecla Min. Co. 1890–94, Shop Foreman, Boston Montana Min. Co., Butte, Mont. 1897–1901, Machinist, Copper Queen Cons. Min. Co. 1902–05, Master Mechanic, Calumet & Arizona Min. Co.

Present position: 1905 to date; Master Mechanic, Copper Queen Cons. Min. Co.

Stanley B. Moore, Bacchus, Utah.

Proposed by C. W. Stimpson, J. B. Ambler, Ernest Gayford.

Born 1877, Philadelphia, Pa. 1895, Public Schools of Philadelphia including Central High School. 1897–1901, Univ. of Pennsylvania, B. S. in E. E. 1895–97 and summer vacations to 1901, Electric Storage Battery Co., Philadelphia, in various capacities. 1901–03, Helios-Upton Co. of Chicago, Storage Batteries. 1903–13, E. I. duPont de Nemours Powder Co. and subsidiaries in various locations.

Present position: 1913 to date; Supt., Bacchus Works, Hercules Powder Co.

John More, Port Colborne, Ont., Can.

Proposed by Robert B. Stanton, John F. Thompson, W. A. Bostwick.

Born 1870, Scotland. General education. 1888–1916, holding various positions, International Nickel Co., Bayonne, N. J.

Present position: Genl. Mgr., International Nickel Co. of Canada, Ltd.

George Morris, DeLancey, Jefferson Co., Pa.

Proposed by Thomas A. Furniss, John McLeavy, James B. Cook.

Born 1877, Westmoreland Co., Pa. Common School education. One term at Western Pennsylvania Classical and Scientific Institute. Five years firing boilers and running stationary engines; one year as trapper; two years as oiler, South West Coal & Coke Co., Alverton, Pa. One year as Chainman, H. C. Frick Coke Co., Scottdale, Pa. Two years, Transitman and Draftsman, Keystone Coal & Coke Co., Greensburg, Pa. Two years, Min. Engr., Horword Gas Coal Co., Greensburg, Pa. Three years, Engr. and Supt., Penobscot Coal Co., Greensburg, Pa. Three years, Supt., Jamison Coal & Coke Co., Greensburg, Pa. Two years, Min. Engr., Quemahoning Coal Co., Somerset, Pa.

Present position: Supt., Adrian, Florence and Elk Run Shaft, Rochester and Pittsburg Coal & Iron Co., Punxsutawney, Pa.

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Herbert George Moulton, New York, N. Y.
Proposed by Robert M. Raymond, Walter H. Aldridge, Lucius W. Mayer.
Born 1883, Bellevue, Idaho. 1905, Univ. of Oregon, B. S. 1909 to date, Min. Engr., Eugene Meyer, Jr., & Co., New York, N. Y. 1915 to date, Cons. Min. Engr., The Public Service Commission for the First District, State of New York.
Present position: Consulting Engineer.

John Francis Murphy, Minneapolis, Minn.
Proposed by Rukard Hurd, Edward P. McCarty, W. R. Appleby.
Born 1880, Litchfield, Minn. High School, Litchfield, Minn. 1906, Univ. of Minn., Civ. Engr. 1906–07, Structural Steel work, American Bridge Co., Ambridge, Pa. 1908–13, Min. Engrg., Oliver Iron Min. Co., Duluth, Minn. 1913–16, Instructor in Mining, Minnesota School of Mines, Univ. of Minnesota, and Min. Engr. for the Minnesota Tax Commission.
Present position: Instructor in Mining, Minnesota School of Mines, Univ. of Minnesota.

Henry Clinton Nelson, Metcalf, Ariz.
Proposed by R. B. Earling, P. B. Scotland, A. L. Ferris.
Born 1889, Empire, Colo. 1906, High School, Georgetown, Colo. 1906–10, Miner, East Argentine Min. Co., Georgetown, Colo. 1911, Sampling Dept., Ray Cons. Copper Co., Ray, Ariz. 1912, Mine and Sampling Dept., Ray Central Copper Co., Ray, Ariz. 1913–14, Miner, Empire Tunnel Co., Empire, Colo. 1914–15, Shift Boss and Foreman, King Mine, Arizona Copper Co., Metcalf, Ariz.
Present position: Foreman, King Mine, Arizona Copper Co.

Domingo Nogues, Washington, D. C.
Proposed by Max W. Ball, W. A. Williams, Van H. Manning.
Born 1868, Mercedes (Prov. of Buenos Ayres), Argentine. 1887, Toulouse Univ., B. Sc. Attended Lycée Saint Louis, Paris, where I was a member of the class of Mathématiques Speciales. Entered on competitive examination, the Ecole Nationale Supérieure des Mines, in Paris, and received in 1894 the Diplome Supérieur d'Ingénieur Civil des Mines. Traveled in Western France and Belgium to examine the Douges Coal Mines. For two years was superintendent of the flour mills in Mercedes, Argentina. Until 1897, was a member of the advisory commission of the Banco de la Nación Argentine, Mercedes Branch. I afterward prospected for the Government in Argentina and drilled for water in Catamarca Province. I continued my prospecting for gold in Puna, Jujuy Province, and carried on work along this line for several companies. I also spent some time in examining fields for gold dredging in Tierra del Fuego. For several years I was consulting engineer for the Compañia Minera Sierra de Mines, in the Province of Rioja, which position I have held up to the present year.
Present position: Commissioned by the Argentine Government to investigate and report on the petroleum industry in the United States, Mexico, Roumania, Galicia, and Russia.

Karl Eloff Olsen, East Chicago, Ind.
Proposed by G. P. Hulst, Fred P. Clark, Charles Lindmueller.
Born 1883, Sweden. Lindsborg High School; Bethany Academy. 1901–05, Bethany College, A. B., Chemistry, Geology, Mineralogy. 1905–07, Univ. of Kansas. 1907–08, Scholarship, Yale Univ.; Special studies at Armour Institute of Technology. 1912, Univ. of Chicago. 1908–09, United Zinc & Chemical Co., Kansas City, Kans. 1910, Lackawanna Steel Works, Buffalo, N. Y. 1911, Mallinckrodt Chemical Works, St. Louis, Mo. 1912, Swift & Co., Chicago, Ill. 1913, Goldschmidt Detinning Co., East Chicago, Ind.
Present position: Chemist, Goldschmidt Detinning Co.

Michael Frederick Olwage, West Allis, Wis.
Proposed by F. W. Traphagen, W. G. Haldane, Harry J. Wolf.
Born 1889, Kimberley, So. Africa. 1909, Matriculation graduate. Three and one-half years at Durban College, Natal, So. Africa. 1910–11, Completed a course of Manual Training at Speo Bona Training Centre, Johannesburg. 1913–16, Completed a course in Steam Engineering, Milwaukee School of Engineering, evenings. 1910–11, worked in the Reduction Dept., viz., Sampler of Cyanide Solutions, Dressing Plates and Collecting Amalgam, etc., Geldenhuis Deep Gold Min. Co. 1911–12. Stope Sampler and Computing Values, Panning, Underground Mining, etc., J.

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Simmer Deep Gold Min. Co. 1912, Assisting J. D. Olwage, Cyanide. 1912, Contractor, etc., N. B. Geldenhuis & Simmer Deep Mines, Johannesburg, So. Africa.
Present position: Allis-Chalmers, erecting crushers, hoists, pump engines, etc.

Morey Austin Park, Parkesburg, Pa.
Proposed by H. A. Beale, Jr., Joseph T. Hilles, A. A. Stevenson.
Born 1886, Rummerfield, Pa. 1902, Wilmington High School, Wilmington, Del. Since leaving high school, have completed two correspondence courses; one in chemistry and one in metallography. 1902–04, Chemical Laboratory, Diamond State Steel Co., Wilmington, Del. 1904–07, Chemical Laboratory, Phoenix Iron Co., Phoenixville, Pa. 1907–08, Yocum Eachus Laboratories, Newark, N. J. 1908–10, Traveling in the West for health and study, covering chiefly the Rocky Mountain region from Mexico to Montana.
Present position: 1910 to date; Chemist, Parkesburg Iron Co.

Edward Franklin Pelton, Tyrone, New Mexico.
Proposed by Arthur Notman, Y. S. Bonillas, G. F. G. Sherman.
Born 1877, Flushing, New York. 1898, Princeton Univ., B. S. 1902, Columbia Univ., E. M. 1898–1900, Installation of Electrical Machinery with D. H. Darrin Co., New York. 1900–02, Columbia Univ. 1902–03, Miner, Anaconda Co. & Boston and Montana Co., Butte. 1903–06, Geol., Anaconda Copper Co. 1906–12, Geol., Detroit Copper Co., Morenci, Ariz. 1912–13, Examining Engr., Phelps, Dodge & Co.
Present position: 1913 to date; Mine Supt., Burro Mountain Co.

Charles Edgerton Pickett, McGill, Nev.
Proposed by C. N. Lakenan, G. C. Riser, A. G. Marsh.
Born 1890, St. George, Utah. 1904, Public School, St. George, Utah. 1909, High School, St. George and Provo, Utah. 1911–13, Univ. of Arizona. 1913–15, Univ. of Utah. 1910–11, Operator, Utah Copper Co., Garfield, Utah. 1911, Chainman and Classifier Foreman, Ray Cons. Copper Co., Hayden, Ariz. 1912, Mill Operator, Miami Copper Co., Miami, Ariz. 1913, Assayer, Copper Queen Cons. Copper Co., Bisbee, Ariz. 1915–16, Assayer and Foreman, Cadmium Plant, U. S. Smelt. Co., Midvale, Utah.
Present position: Sampler, Nevada Cons. Copper Co.

William R. Price, Marysville, Mont.
Proposed by C. W. Goodale, P. M. McHugh, George T. McGee.
Born 1875, Cardiff, Wales. 1880–92, Common School, Cañon City, Colo. 1894–96, Miner and Foreman, Western Fuel Co., Chandler, Colo. 1896–98, Colorado Fuel & Iron Co., Chandler, Colo. 1898–99, Miner, Barrick Gold Min. Co., Quinda, Colo. 1901–10, Mgr., Empire Gold Mines Co., Laplata, Colo. 1912–14, with Duncan MacVichie, Salt Lake City. 1914–15, Supt., Shannon Min. Co., Marysville, Mont.
Present position: Supt., Shannon Mine of the Barnes King Dev. Co.

Dan Reichel, Miami, Ariz.
Proposed by B. B. Gottsberger, Frederick W. Solomon, R. B. Yerxa.
Born 1888, Sturgeon Bay, Wis. 1912–16, Univ. of Cal., B. S. in Mining. 1907–08, Assayer, C. E. Bogardus, Seattle, Wash. 1910–11, Assayer, Alaska Treasure, Juneau, Alaska. 1911–12, Assayer, Alaska Perserverance Min. Co., Juneau, Alaska. 1913–16, summers, Harvard Mine, Jamestown, Cal. Goldfield Cons., Goldfield, Nev. Aurora Cons., Aurora, Nev.
Present position: Asst. Assayer, Miami Copper Co.

Charles Robinson, Friesland, Queensland, Aust.
Proposed by Stephen Harris, E. E. Booth, P. L. Goddard.
Born 1875, Westbury, Tasmania. State School education, Westbury, Tasmania. Christian Brothers' Secondary School, Westbury, Tasmania, nine months. 1886, 14 years' training at Smelt. Works, Mt. Lyell, Tasmania, from its inception North Mount Lyell smelting works. 1900, Several years on Western Australian Goldfields in various capacities such as mining mill work. 1902, Fremantle Smelt. Works, Western Aust. 1906, United Reefs Gympie, Queensland.
Present position: 1910 to date; Smelter Foreman, Hampden Cloncurry Copper Mines, Ltd.

George Clark Rogers, Mascot, Tenn.
Proposed by J. N. Houser, H. A. Coy, Charles B. Strachan.
Born 1893, Lexington, Ky. 1901–08, Graded Schools, Lexington, Ky. 1908–11, Morton High School, Lexington, Ky. 1911–15, Kentucky State Univ., College of

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Mines and Metallurgy, Lexington, Ky., B. E. M. 1912, Glen Mary Coal & Coke Co., Glen Mary, Tenn. 1913, Transitman, Mineral Fuel Co., Fleming, Ky. 1914, underground work, Elkhorn Min. Corpn., Fleming, Ky. 1915–16, Engr., American Zinc Co. of Tenn., Mascot, Tenn. Formerly a member of Student Branch, at College of Mines and Metallurgy, Lexington, Ky.

Present position: Min. Engr., American Zinc Co. of Tenn.

Otto Ruhl, Joplin, Mo.

Proposed by H. A. Wheeler, H. A. Buehler, C. R. Forbes.

Born 1881, Republic, Mo. 1904, Drury College, Springfield, Mo., B. Sc. 1906, Drury College, Master's Degree in Science. 1904–15, Field Correspondent, *Lead and Zinc News*, St. Louis, Mo. Field Asst. Missouri Geological Survey. 1905–06, Editor, *Lead and Zinc News*, Joplin, Mo. 1906–08, Doing some professional mining. Asst. City Engineer of Webb City, Mo. 1909–11, Private work, Consulting Min. Engr. 1912, Expert Witness and Min. Engr. for Zinc Mining Association of the United States, Washington, D. C. Tariff Revision. 1913–14, Private Office, Consulting Min. Engr. 1915, Represented State of Missouri. Designing and installing Mines Exhibit at Panama Pacific International Exposition.

Present position: Private Office, Consulting Min. Engr.

J. Franklin Sinclair, Bisbee, Ariz.

Proposed by G. F. G. Sherman, Arthur Notman, Charles A. Mitke.

Born 1870, Nova Scotia, Canada. Early education at Nova Scotia. 1891–94, Miner in Leadville, Colo. 1894–95, Miner in Iron Mines of Nova Scotia Steel Co. 1895–96, Leadville, Colo. 1896–98, Connected with Midnight Min. Co., Midnight, N. Mex.

Present position: 1898 to date; Underground Supt. of North end of Copper Queen Cons. Min. Co.

Henry Byron Slater.

Proposed by Seeley W. Mudd, Robert E. McConnell, Philip Wiseman.

Born 1850, Birmingham, England. 1876–79, Drury College, Springfield, Mo. 1879–81, Brown Univ., Providence, R. I. 1882–85, Cooper Institute, New York, N. Y. All study and training along chemical and metallurgical lines. 1889, Drury College, M. S. 1885–90, in professional work in electro-metallurgy. 1890–1900, Mgr., Canyon City Electric Light & Power Co., Canyon City, Colo. 1900–02, engaged in exploration for oil at various points in Colo. 1902–05, in electrical work in Cal. 1905–10, in mining and exploration in Mexico notably in lower Cal. and also in the state of Cal.

Present position: 1910 to date; engaged in research work and the development of my own inventions—notably, the Slater Process for the extraction of copper, etc., from low-grade ores.

Raymond Weir Smyth, Youngstown, O.

Proposed by F. C. Langenberg, Albert Sauveur, H. M. Boylston.

Born 1888, Baltimore, Md. 1905–09, Harvard College, A. B. 1909–11, Grad. School of Arts and Science. 1913–14, Grad. School of Applied Science. 1914–15, Grad. Student, Carnegie Institute of Technology, Pittsburgh. 1915, Steel Inspector for British Government. 1914–16, Physical Laboratory, American Steel & Wire Co., Worcester.

Present position: Met. Dept., Youngstown Sheet & Tube Co.

Frederick W. Snow, Superior, Ariz.

Proposed by W. H. Aldridge, W. C. Browning, Fred B. Ely.

Born 1883, Manti, Utah. 1901–03, High School, Brigham Young Univ., Provo, Utah. 1903–05, Preparatory School, Univ. of Utah, Salt Lake City, Utah. 1905–09, College, Univ. of Utah, B. S. in Min. Engrg. 1903–04, Milling, Utah Copper Co. 1907–12, Leasing. 1912–14, Sampling, Shift Boss, Inspiration Cons. Copper Co. 1914–16, Miner, Shift Boss, Magma Copper Co.

Present position: Shift Boss, Magma Copper Co.

Clarence O. Stee, Cerro de Pasco, Peru, So. Amer.

Proposed by C. Mason Farnham, Paul S. Couldrey, Myron R. Walker.

Born 1886, Dazey, N. Dak. 1893–1903, Common school grades, Dazey, N. Dak. 1904–07, High School and Preparatory work, at the University, N. Dak. 1907–11, Grad., Univ. of N. Dak., College of Min. Engrg., E. M. 1912, Foreman of Plant,

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Bithulitic Paving and Contracting Co., Winnipeg, Man., Can. 1912, Cerro de Pasco Min. Co., Cerro de Pasco, Peru, S. Amer. 1913–14, Assayer and Assistant in the Engineering Office, Cerro de Pasco Min. Co. 1914–15, Shift boss, Esperanza Mine, Cerro de Pasco Min. Co. 1915, Foreman, Excelsior Mine, Cerro de Pasco Min. Co. 1915–16, Six months vacation to the States. 1916, Shift boss, Esperanza Mine, Cerro de Pasco Min. Co.

Present position: Foreman, Esperanza Mines, Cerro de Pasco Mining Co.

George Richards Stevens, Mina, Nev.

Proposed by Tasker L. Oddie, Fred J. Siebert, W. R. Hamilton.

Born 1882, Benicia, Cal. 1901–03, Leland Stanford Jr. Univ., B. A., Geology and Mining. 1905–06, Asst. Supt., Wei-Hai-Wei Min. Co., Wei Hai Wei, China. 1906–07, mostly operating for myself short surveys and examinations for a few clients. 1907–09, Assayer and Chemist, Glasgow and Western Exploration Co., Golconda, Nev. 1909, Engr., National Mines Co., National, Nev. 1910–13, Tunnel, street, sewer, etc., contracting. Periodic employment as engineer for R. C. Storrie & Co. (contractors), San Francisco. Also interested in several mining ventures in Cal. 1914, Shift boss, Jim Butler Mine, Tonopah, Nev.

Present position: Supt., Olympic Mines Co.

Alfred Clark Stoddard, Miami, Ariz.

Proposed by C. E. Arnold, George R. Lehman, W. F. Palmer.

Born 1879, Denver, Colo. 1902, Michigan College of Mines, Houghton, Mich., B. S. and M. E. 1908–10, Engr. and Mill Foreman, New York and Honduras Rosario Min. Co. 1910–12, Engr. and Shift Boss, Old Dominion Copper Min. & Smelt. Co.

Present position: 1912 to date; Engr., Inspiration Cons. Copper Co.

Wilber Stout, Columbus, O.

Proposed by F. A. Ray, J. H. Frantz, R. H. Sweetser.

Born 1876, Chester, O. Ohio State Univ., B. of Engrg. Six years Chemist, Columbus Iron and Steel Co., Columbus, O. Four years with Geological Survey of Ohio.

Present position: Asst. Geol., Geological Survey of Ohio.

Fritz Oscar Stromborg, New York, N. Y.

Proposed by Edward L. Dufourcq, J. W. H. Hamilton, N. O. Bagge.

Born 1877, Sweden. 1896, Grad., Filipstad Schools of Mines, Filipstad, Sweden. 1896–97, Draughtsman, Avesta Steel Works, Sweden. 1897–98, Melter's helper, O. H. Dept., Avesta Steel Works. 1898, Arrived in U. S. A. and worked six months as chemist at Washburn and Noen, Worcester, Mass. 1898–1907, Draughtsman, Carnegie Steel Co.; Jones and Laughlin; Garrett Cromwell Engrg. Co.; Illinois Steel Co. and C. F. & T. Co.; Squad boss from 1902. 1907–08, Chief Draughtsman, Youngstown Sheet & Tube Co. 1908–11, Cons. Engr., Seattle, Wash. 1911–13, Charge of construction, Cons. Min. & Smelt. Co., Trail, B. C.

Present position: Consulting Engineer.

Aguiles Concha Stuardo, Santiago de Chile, So. Amer.

Proposed by Fred Hellman, E. E. Barker, D. L. H. Forbes.

Born 1885, Santiago. 1894–1901, General instruction at the Lyceum "Miguel Luis Amunategui" and that of "de Aplicacion" in Santiago. 1902–05. Four years at the School of Arts and Crafts of Santiago where I obtained the title of electrician, of the first class. The Government sent me to Europe for six years. 1906–07, Duvignan School in Paris. 1907–09, National High School for Mining, of Paris, as "Foreign Functionary." 1909–11, Special School of Public Works of Paris. 1912, I returned to Chile after having visited many mines in France and Germany. 1912, in the month of June I entered the General Directorate of Public Works of the Ministry of Industries in Santiago as "Engineer of Seccion." 1913, I obtained the position of Mineralogist in the Inspectorate of Mines. 1914, I was promoted to Geologist of the Inspectorate of Mines of the General Directorate of Public Works, which position I still occupy at present.

Present position: Geol., General Directorate of Public Works.

Quin W. Stuart, Ensley, Ala.

Proposed by Robert Hamilton, H. S. Geismer, R. J. Holden.

Born 1887, Roanoke, Va. 1911, Virginia Polytechnic Institute, Blacksburg, Va., B. Sc. in Min. Engrg. 1911, Timberman and drill runner, Asst. Engr., Tennessee

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Copper Co., Ducktown, Tenn. 1911–12, Levelman, Transitman on Min. Division, Tennessee Coal, Iron and R. R. Co., Birmingham, Ala. 1912–13, Construction Engr., Tennessee Coal, Iron and R. R. Co., Blocton, Ala. 1913–14, Construction Engr., Tennessee Coal, Iron and R. R. Co., Bayview, Ala. 1914–15, Division Engr., Tennessee Coal, Iron and R. R. Co., Edgewater, Ala. 1915–16, Genl. Supt. of Capitol City Construction Co. (Contractors) of Nashville, Tenn.

Present position: Construction Engr., Tennessee Coal, Iron and R. R. Co. of Birmingham, Ala., at Edgewater, Ala.

Donald McDonald Sutor, El Paso, Tex.

Proposed by William D. Gordon, R. F. Manahan, W. J. Deavett.

Born 1874, Wis. 1879–92, Public School, Grade and High School. 1893–94, Univ. of Wisconsin, Civ. Engrg., no degree. 1894–99, Construction, survey and inspection, Mississippi River Improvement, under U. S. Engr. Corps., U. S. Army. 1900–01, Coal Miners, Whitebreast Fuel Co. 1902 to present, in Sales Dept., Sullivan Machinery Co., Chicago; at Denver, The Rand, So. Africa, and Southwest U. S. and Mexico.

Present position: Sales Mgr., Southwest U. S. and Mexico, Sullivan Machinery Co.

Herbert Ellsworth Treichler, Chuquicamata, Chile, So. Amer.

Proposed by Henry Hay, E. E. Barker, Fred Hellmann.

Born, 1888, Niagara, N. D. 1908, Univ. of North Dakota, Mech. Engr. 1909, Univ. of North Dakota, Min. Engr. 1909, in charge of State Mine Experimental Station, Hebron, N. D. 1909–13, Min. Engr., Chino Copper Co., Santa Rita, N. M. 1913–14, Min. Engr., Detroit Copper Min. Co., Morenci, Ariz. 1914 to date, Mine Engr. and Asst. Mine Supt., Chile Exploration Co., Chuquicamata, Chile.

Present position: Asst. Mine Supt., Chile Exploration Co.

Herbert Winfred Walter, Bayonne, N. J.

Proposed by Robert B. Stanton, James T. Kemp, William Campbell.

Born 1880, Odessa, Minn. 1905, Washington State College, B. S. in Mining. 1909, Columbia Univ., Met. Engr. 1906–07, Supt., Clifton Water and Improvement Co., Clifton, Ariz. 1909–16, International Nickel Co. 1909–10, Foreman, Monel Metal Refinery. 1910–11, Asst. to Supt., Monel Metal Sheet Mill Dept. 1911–12, Supt., International Nickel Co., Camden Works, Camden, N. J. 1913–14, Supt., Electrolytic Nickel Dept., and Platinum Refinery. 1915, Supt., Nickel and Copper Cupolas.

Present position: Asst. Genl. Mgr., International Nickel Co., Port Calborne, Ont., Canada.

Brinsley M. Walton, So. Porcupine, Ont., Canada.

Proposed by Henry W. Nichols, Robert A. Bryce, Arthur A. Cole.

Born 1866, Hersperpoint, England. Common school education. 1872–80, Arnpror, Pembroke, Ont. Studied nights. Have not been to any technical school. Have good general knowledge of history, latin, mathematics. 1880–83, Drug business with James Findly, Pembroke, Ont. 1884–91, Engrg. Dept., Grand Trunk Ry. 1891, Mining as common Miner, Beaver Mine, Port Arthur, Ont., Foreman Ajak Mine, Sandow, B. C. 1894, Foreman Mount Royal Mine, Elk Lake. 1909, Mgr., Guelph Mine, Monroe Township, New Ont. 1910–11, Foreman, Ross Group, So. Porcupine.

Present position: Mgr., Porcupine Premier Mine.

Keith Prout Webb, Friesland, Queensland, Aust.

Proposed by Stephen Harris, W. H. Corbould, P. L. Goddard.

Born 1888, Brighton, Vict., Aust. 1907–09, Workingmen's College, School of Mines, Melbourne, Metallurgy Diploma. 1910, Junior Assayer, Broken Hill Prop. Co., Ltd., Port Pirie, So. Aust. 1910–14, 1st Asst. Assayer, Hampden Cloncurry Copper Mines, Ltd., Friezland, North Queensland.

Present position: 1914 to date; Chief Assayer and Ore Buyer, Hampden Cloncurry Copper Mines, Ltd.

Joseph R. Wilkinson, Anaconda, Mont.

Proposed by Frederick Laist, Charles D. Demond, Louis V. Bender.

Born 1892, Berrien Springs, Mich. 1905–09, High School, Lansing, Mich. 1909–11, Mech. Engrg., Michigan Agricultural College, East Lansing, Mich. 1912–14,

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Min. Engrg., Michigan College of Mines, Houghton, Mich., E. M. 1911–12, Efficiency work. Installation of Taylor System of Seager-Engine Works, Lansing, Mich. 1914–15, Mine Engr., Union Basin Min. Co., Golconda, Ariz. 1915, Draughtsman, American Blower Co., Detroit, Mich.

Present position: 1915 to date; Testing Dept., Anaconda Copper Min. Co.

Howard E. Williams, Calumet, Mich.

Proposed by C. H. Benedict, Claude H. Cooper, H. D. Conant.

Born 1873, Boston, Mass. 1891–95, Cornell Univ., M. E. 1895, Machinist and draftsman, New York Central R. R. 1896–97, E. Rutsler, Heating Contractor, in estimating dept. 1898–99, Draftsman in Mechanical Engineer's office, Lehigh Valley R. R. 1899, part, Mech. Engr., Erie R. R. 1899–1901, Draftsman, Calumet & Hecla Min. Co. 1901 to date, Chief Draftsman, Calumet & Hecla Min. Co., Principal work designed; Kimberly Skips in Red Jacket Shift. Rebuilding Hecla and Calumet Mills. Designing No. 1 and No. 2 regrinding plants, with dredge, shore plant and classifying plant. Sand Searching Plant at Cobb Mills, Rebuilding Isle Royale, Osceola, Isle No. 1 and No. 2 Stamp Mills. Enlarging Abmeels Stamp Mill. Surface Plant at Abmeels No. 3 and No. 4 Shaft, including Roch house, boiler and engine house, and dry house. Roch Houses at Alloney No. 2, Abmeels No. 2, North Kearsary No. 1, Isle Royale No. 5 and No. 7 Shafts. Surface plant at White Pine Mine including Mill, Crushing Plant, Shaft Horses, Engine and Boiler Houses, etc.

Present position: Chief Draftsman, Calumet & Hecla Min. Co.

Maurice Eugene Williams, Timmins, Ont., Canada.

Proposed by C. E. Rodgers, A. R. Globe, L. B. Eames.

Born 1888, Blockpack, Kans. 1894–1906, Common School education, Colorado Springs, Col., Pasadena, Cal. 1906–11, Genl. Mill. and Ref. work, Pittsburg Silver Peak Gold Min. Co., Blair, Nev. 1911–12, Ref. and Mill work, Goldfield Cons. Mines, Goldfield, Nev. 1912–13, Shift Boss, Mother Lode Sheep Creek Min. Co., Sheep Creek, B. C. 1914–15, Mill Supt., Mother Lode Sheep Creek Min. Co., Sheep Creek, B. C.

Present position: 1915 to date; Asst. Mill Supt., Hollinger Cons. Gold Mines.

Ridgeway Robinson Wilson, Fernie, B. C., Can.

Proposed by W. F. McNeill, Lewis Stockett, S. F. Kirkpatrick.

Born 1890, Carnegie, Pa. 1897–99, Public School, Carnegie, Pa. 1900–01, Public School, Fernie, B. C. 1901–02, Alfreton Road Board School, Nottingham, England. 1902–05, Grey College, Bloemfontein, So. Africa. 1905–06, Matriculated, Pickering College, Ont., Can. 1909–13, School of Mining, Kingston, Ont., B. Sc. in Min. Engrg. Member, Canadian Mining Institute. 1913, Certificate as Assayer, Province of British Columbia. 1906, Chainman and rodman Railway location, East Kootenay, B. C. 1907, Prospecting, Cobalt, Timagami and Gowganda, Ont. 1908, Asst. to W. R. Wilson, Cons. Engr., Toronto, Can. Examination of properties, Cobalt, Timagami, Gowganda, East Kootenay and Nova Scotia. 1909, Diamond drill, Lake Superior Iron and Steel Co., Missanabie, Ont. 1910, Charge exploratory work (Davis Calyn core drill) Twin City Coal Co., Edmonton, Alta. Asst. to W. R. Wilson, examination of properties, Brazeau and Jasper Park Coal Fields, Alta. 1911, Sampler and Assayer, Crows Nest Pass Coal Co., Fernie, B. C. 1912, Mine Surveyor and draughtsman, Crows Nest Pass Coal Co., Fernie, B. C. 1913–14, Engr., Crows Nest Pass Coal Co., Fernie, B. C. Mine Surveys, Topographical, Railroad Location and Construction M. F. & M. Ry. Hydro. Survey Elk River Falls and Canyon for Crows Nest Pass Electric Light & Power Co. Survey and cruise of Coal Co. timber lands. General office work.

Present position: 1915 to date, Outside Supt. and Supt., Coke Dept., Crows Nest Pass Coal Co., Michel, B. C.

Jay Pendleton Wood, Ouray, Colo.

Proposed by G. H. Barnhart, C. R. Wilfley, Fred Carroll.

Born 1889, New York, N. Y. 1908, DeWitt Clinton High School, New York City. 1912, Columbia Univ., E. M. 1912–15, The Wanakah Min. Co., Ouray, Colo. 1915, Member of the firm of Ingersoll, Wilfley & Wood. Mining on own account. 1916, The Mountain Top Min. Co., Ouray, Colo.

Present position: Supt., The Mountain Top Mine, Sneffels, Colo.

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Associate Members

Frederick W. Copeland, Salt Lake City, Utah.
Proposed by Frederick K. Copeland, H. T. Walsh, E. J. Rossbach.
Born 1892, Winnetka, Ill. 1913, Harvard, A. B. 1913–16, Sales Engr., Sullivan, Mach. Co.
Present position: Sales Engr., Sullivan Machinery Co.

A. D. Johnson, Jr., Kansas City, Mo.
Proposed by M. K. Shaler, Edgar Rickard, W. L. Honnold.
Born 1892, Kansas City, Mo. 1898–1905, Garfield School, Kansas City, Mo. 1905–09, Central High School, Kansas City, Mo. 1909–13, Univ. of Kansas, B. S. 1913–14, School of Mines, Freiberg, Saxony. 1914, Exploration—mining and geological in Greenland, as assistant to Mr. S. H. Ball, New York. 1914–15, Delegate in Belgium, Commission for Relief in Belgium, London, England. 1915, Geol., Wichita Natural Gas Co., Bartlesville, Okla. 1916, Examination in Spain for Mr. Ball and Anaconda Copper Min. Co.
Present position: Testing Dept., Anaconda Copper Min. Co., Anaconda, Mont.

William W. Miller, Schenectady, N. Y.
Proposed by Sanford B. Belden, W. R. Whitney, H. M. Warren.
Born 1880, Rochester, N. Y. Grammar and High School. Private tutoring for Electrical and Mechanical Engineering. 1899–1902, Student's course, General Electric Co. 1902–08, Commercial Engineering, Power and Min. Dept., General Electric Co. 1908–10, Electrical Engr., Bausch and Lomb Optical Co., Rochester, N. Y. 1910 and at present, Power and Min. Dept., General Electric Co.
Present position: Sales Engr., Mining Section, Power and Mining Dept., General Electric Co.

Junior Members

Joe Barton, Rolla, Mo.
Proposed by Charles Y. Clayton, H. A. Buehler, H. T. Mann.
Born 1894, Mineola, Mo. 1909–13, High School, Montgomery City, Mo. 1914, summer, Utah Copper Co., Bingham, Utah; Daily Judge Min. Co., Park City, Utah. 1915, summer, Carson Min. Co., Joplin, Mo. 1916, summer, American Zinc Co., Mascot, Tenn.
Present position: 1913 to date; Student, Missouri School of Mines.

Ralph Dale, Rolla, Mo.
Proposed by Charles Y. Clayton, H. A. Buehler, H. T. Mann.
Born 1895, Herrin, Ill. 1908–12, Herrin High School. 1912–13, Company Weighman and Clerk, St. Louis Carterville Coal Co. 1916, Laborer, Copper Queen Cons. Min. Co., Bisbee, Aris.
Present position: 1913 to date; Student, Missouri School of Mines.

Clemence W. Hippard, Rolla, Mo.
Proposed by Charles Y. Clayton, H. A. Buehler, H. T. Mann.
Born 1892, Belleville, Ill. 1906–09, High School, Belleville, Ill. 1909–12, Weighman, clerk, top foreman, Vulcan Coal & Min. Co., Belleville, Ill. 1912–13, coal inspector, St. Louis Coulterville Coal Co., Coulterville, Ill. 1913, summer, in charge of mine, it being shut down, St. Louis Coulterville Coal Co. 1915, summer, mucker, Portland Gold Min. Co., Victor, Colo. 1916, summer, mucker, Copper Queen Cons. Min. Co., Bisbee, Aris.
Present position: 1913 to date; Student, Missouri School of Mines.

Frederick L. Schmidt, So. Bethlehem, Pa.
Proposed by Howard Eckfeldt, Joseph W. Richards, Benjamin L. Miller.
Born 1894, New York, N. Y. 1901–03, Private School, Brooklyn. 1903–09, Private and Public Grammar Schools, Brooklyn and New York. 1909–13, Manual Training High School, Brooklyn. 1908–09, Elementary Architecture in Pratt Institute Night School. 1915, summer, Foreman, Bethlehem Steel Co. 1916, summer, Asst. Millman, Bethlehem Steel Co.
Present position: 1913 to date; Student, Lehigh Univ.

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CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Sept. 10, 1916 to Oct. 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 118, February to October, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of Oct. 10, 1916.

ADAMS, ARTHUR K., Min. Geol., Geol., Andes Copper Co., Chanaral, Chile, So. Amer,
AMBLER, J. B.....703 Kearns Bldg., Salt Lake City, Utah.
ANDERSON, AXEL E..DuPont & Co., 901 Central Savings Bank Bldg., Denver, Colo,
ANDERSON, HECTOR GEORGE S., Asst. to Genl. Mgr., Chino Copper Co., Hurley, New Mexico.
AUSTIN, EDWIN A., Min. Engr.....Jarbidge, Nevada.
AYER, FRANK A.....Inspiration Cons. Copper Co., Miami, Ariz.
BAGG, RUFUS M.....7 Brokaw Place, Appleton, Wis.
BAILEY, LEWIS NEWTON.....Barstow, San Bernadino Co., Cal.
BARREN, HENRY A.....12337 Cedar Road, Cleveland Heights, Ohio.
BARRETT, THEODORE H., Min. Engrg. Dept., Case School of Applied Science, Cleveland, O.
BECK, ERICH A.....Goldschmidt Thermit Co., 120 Broadway, New York, N. Y.
BEESON, JOSEPH JOSIAH.....Alta, Utah.
BELLINGER, HERMAN C., Asst. Cons. Engr., Chile Exploration Co., Chuquicamata, via Antofagasta, Chile, So. Amer.
BERGÉN, RAYMOND C.....58 South Arlington Ave., East Orange, N. J.
BERNARD, CLINTON P., Min. Engr., Société Internationale Forestiere et Miniere du Congo, Forminiere Mission Kasai, Tshikapa, Congo Belge, via Boma and Luebo.
BISSELL, ROBERT WILSON.....Greensburgh, Westmorland Co., Pa.
BLOW, JOHN J.....404 High Holborn, London, W. C., England.
BOYLE, HARRY, Civ. and Min. Engr.....42 Story-Smith Bldg., Seattle, Wash.
BRADT, M. L.....Arizona Asbestos Assn., Globe, Ariz.
BREWSTER, BURT B., Sullivan Machinery Co., Kearns Bldg., Salt Lake City, Utah.
BROWN, THOMAS E.....Room 1402, 18 E. 41st St., New York, N. Y.
BROWNE, SPENCER C.....43 Exchange Place, New York, N. Y.
BRUNS, C. L., JR., Supt., Mina Constancia, San Vicente, Vinales, Pinar del Rio, Cuba.
BURNS, JAY JOSEPH.....Instructed to hold everything.
CAMPBELL, ARTHUR R.....Rockhill Manor, Kansas City, Mo.
CAMPBELL, W. C.....Box 145, Tooele, Utah.
CARSON, ELLARD W., Alpine Quicksilver Min. Co., Hernandez, San Benito Co., Cal.
CARSON, JOSEPH A.....Midvale Steel Co., Widener Bldg., Philadelphia, Pa.
CARTER, RUSSELL S.....52 Vanderbilt Ave., New York, N. Y.
CHARLES, W. W.....346 So. Ardmore Ave., Los Angeles, Cal.
CHEN, FAN.....Hartley Hall, Columbia Univ., New York, N. Y.
CLARK, HENRY, Messrs. Head, Wrightson and Co., Ltd., Stockton Forge, Stockton-on-Tees, England.
COHEN, SAMUEL W.....601 Dominion Express Bldg., Montreal, Canada.
COONER, JOHN D.....9 Platt Place, Scranton, Pa.
CORBET, EDWARD B.....2650 Scott St., San Francisco, Cal.
COURTIS, WILLIAM M.....133½ Ferris Ave., Highland Park P. O., Detroit, Mich.
CRAIG, JOHN J.....Northwestern Improvement Co., Brainerd, Minn.
CREGAN, JOHN F.....1010 Broad St., Newark, N. J.
CREMER, FELIX.....Andes Copper Min. Co., Chanaral, Chile, South America.
CUNNINGHAM, GEORGE HAMILTON, Chief Engr., Electrolytic Zinc Co. of Australia, Hobart, Tasmania.
DALEY, S. HOWARD, JR.....Cananea Cons. Copper Co., Cananea, Mexico.
DAMAN, ARTHUR CHESTER.....2678 Endora St., Denver, Colo.
DICKINSON, ALBERT W.....Supt., Chicago & Carterville Coal Co., Herrin, Ill.
DIXON, ABNER FAISON.....86 South Oxford St., Brooklyn, N. Y.
DUFOURCQ, EDWARD L., Cons. Min. Engr., Rooms 808, 809 and 810, 18 Broadway, New York, N.Y.
DUNN, THEODORE S.....P. O. Box 431, Du Bois, Pa.
EYOUB, DJEVAD.....16 Beekman Place, New York, N. Y.
FAIRBAIRN, ALAN J.....St. John Mining Co., Montezuma, Colo.

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FARISH, JOHN B., Min. Engr.....P. O. Box 182, San Mateo, Cal.
FAUST, GUY C.....19 West Jackson St., Wilkes-Barre, Pa.
FAWCETT, JAMES H.....Athenaeum Club, Sydney, N. S. W., Australia.
FELL, HAROLD B., Supt. and Engr., Wyoming Valley Water Supply Co., Markle Bldg., Hazleton, Pa.
FRASER, DONALD D.....406 W. Granite St., Butte, Mont.
FRASER, LEE, South American Engineering Corpn., 52 William St., New York, N. Y.
FULLER, MYRON L.....185 Spring St., Brockton, Mass.
GANTHIER, CHARLES B.....Box 102, Golden, Colo.
GARRISON, MURRAY.....940 West Granite St., Butte, Mont.
GATCH, ELIAS S.....Care N. N. Gatch, 5572 Waterman Ave., St. Louis, Mo.
GEPP, HERBERT WILLIAM, Genl. Mgr., Amalgamated Zinc (deBavay's) Ltd., 2914 Equitable Bldg., New York, N. Y.
GILL, PHILIP L.....165 Broadway, New York, N. Y.
GLEASON, FRANK A.....Taylor, Pa.
GREENOUGH, WARREN E., Min. Engr.....Old National Bank Bldg., Spokane, Wash.
GRIFFITHS, ANDRÉ P., Cons. Min. Engr., Horton Kirby, King's Road, Kingston-on-Thames, England.
GUERNSEY, JOHN B.....Westmoreland Club, Richmond, Va.
HALL, MORTIMER LOUIS.....555 S. Los Robles, Pasadena, Cal.
HANAHAN, MARION L.....Hartley Hall, Columbia Univ., New York, N. Y.
HANLEY HERBERT R., Supt., Electrolytic Zinc Plant, Mammoth Copper Co., Kennett, Shasta Co., Cal.
HARDY, THOMAS WOODLAWN, JR., Nova Scotia Steel & Coal Co., New Glasgow, N. S., Canada.
HASELTINE, RICHARD S.....149 So. Kingsley Drive, Los Angeles, Cal.
HAYDEN, WALLACE H.....Batavia, N. Y.
HAYWARD, CARLE R., Instructor in Min. and Met., Mass. Inst. of Technology, Cambridge, Mass.
HEATHCOTE, C. F., The Bank of Australasia, Ltd., Threadneedle St., London, England.
HEDLEY, ROBERT R.....30 Fairfield Bldg., Vancouver, B. C., Canada.
HERR, IRVING.....Supt., Crucible Flake Graphite Co., Ashland, Ala.
HOGOBOOM, W. C.....Box 173, Tar River, Okla.
HONNOLD, W. L.....Hotel Gotham, 5th Avenue and 55th St., New York, N. Y.
HOUSHOLDER, E. ROSS.....P. O. Box 302, Rolla, Mo.
HOWARD, L. OGILVIE.....Supt., International Smelt. Co., Miami, Ariz.
HUNTER, CHARLES.....149 W. 74th St., New York, N. Y.
HURD, RUKARD, Cons. Min. Engr., Director Mines, Minnesota Tax Commission, State Capitol, St. Paul, Minn.
JEWETT, FRANK G.....1816 Fremont Ave. South, Minneapolis, Minn.
JOHNSON, CLIFTON BEACH.....Co. E, 7th Infantry, McAllen, Tex.
JOHNSTONE, JAMES O.....910–144th St., E. Chicago, Ind.
JONES, WILLIAM S.....Silver City, New Mexico.
KEE, HARRY A.....Kerr Lake Min. Co., Ltd., Cobalt, Ont., Canada.
KEMP, L. W., Chile Exploration Co., Chuquicamata, via Antofagasta, Chile, South America.
KENT, WILLIAM ST. G.....808 West End Ave., New York, N. Y.
KERR, DUNCAN M.....Winthrop, Cal.
KIMMEL, HARRY ROBERT.....18725 Sloane Ave., Lakewood, O.
KISHMAN, MAURICE WALTER.....Inspiration Cons. Copper Co., Miami, Ariz.
LANCASTER, THOMAS, Civ. Engr.....2 Rector St., New York, N. Y.
LAU, KAAN.....Kung Yuen Co., 223 Wing Lok St., Hong Kong, China.
LAWR, CLARENCE W., Tigre Min. Co., Ysabal, Esqueda, Son., Mexico, via Douglas, Ariz.
LEAHY, RICHARD A.....605 Mercantile Bldg., Denver, Colo.
LEWIS, CLANCEY M.....205 White Bldg., Seattle, Wash.
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Date of Election.	Name.	Date of Decease.
1893	**Coxe, Eckley B.	Sept. 20, 1916.
1905	**Crawford, Walter Howard	July 1, 1916.
1906	*Heck, Elmer Cooper	—— —, 1916.
1886	*Koch, Walter E.	—— —, 1916.
1889	*Stanton, F. McM.	Sept. 12, 1916.
1914	*Young, Ralph William	—— —, 1916.

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History of the Flotation Process at Inspiration

Discussion of the paper of RUDOLF GAHL, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1627 to 1681.

RUDOLF GAHL, Miami, Ariz.—Since I wrote the paper on flotation which is in your hands, important developments have taken place, and, for this reason, I will try in a few words to bring it nearer up to date as far as the Inspiration plant is concerned.

You may have noticed that in the Inspiration concentrator, flotation is not solely relied upon for the recovery of the coarser material, but that flotation tailings are split into a sand and a slime product on drag classifiers. The sand product is treated on tables which thus supplement the work of the flotation machines.

Extensive tests have shown us that, if we wanted to, we could leave out these tables and substitute additional flotation machines, as they will make fully as good, if not a better recovery than tables on the deslimed feed, but the treatment would be more expensive, especially in view of the fact that oils are required which cost more than those which we are now using in our main flotation plant. We have, however, decided to apply flotation treatment to our table middlings.

I would also like to add a few words regarding the treatment of oxidized copper ores. Although our experience shows that the addition of hydrogen sulphide and other soluble sulphides effects a very good recovery of copper carbonates with certain ores, we have not been able to prove that we could use it advantageously for the ore mixture which we are treating in our concentrator, and for this reason, have looked toward leaching for extracting the carbonate and silicate copper that we are losing now. Experiments in this direction are going on and are giving very encouraging results. It may interest you to hear that we intend to use limestone for the precipitation of the copper which goes into solution, as electrolytic precipitation seems to be out of the question on account of the diluteness of the resulting solutions, and precipitation by iron was rightly objected to on account of the unavoidable contamination of the water supply. We feel very hopeful about the success of our limestone precipitation which, if it holds what it seems to promise, will develop into a novel feature of copper metallurgy.

FREDERICK LAIST, Anaconda, Mont. (written discussion).—Dr. Gahl is assuredly to be complimented on the preparation of so well-written and complete a paper as well as upon the excellent work done under his supervision. One cannot but be impressed by the care and thoroughness with which all possible combinations were investigated,

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and the paper leaves one with the impression that the equipment finally chosen is without a doubt the best obtainable for the conditions prevailing.

I was specially interested in Dr. Gahl's description of the development of the Inspiration flotation machine, and its final perfection is certainly a credit to his keenness and powers of observation. The machine is simplicity itself and strikes me as the logical development of the pneumatic type.

The relative merits of the impeller and pneumatic types of flotation machines have been the subject of much discussion and the selection of type is doubtless dependent largely on local conditions and on the characteristics of the ore undergoing treatment. We have always been of the opinion at Anaconda that whenever a neutral or alkaline treatment was used and the oils could be added to the pulp going to the ball mills, the pneumatic type had an advantage both as regards power consumption and installation cost. When, however, the treatment requires the use of acid as is the case on some copper ores and most zinc ores, the pneumatic machine loses much of its advantage. Obviously, the acid cannot be introduced into the ball mill and it is generally necessary to add it ahead of, or at the same time as, the oil.

Thus it becomes necessary to insert an agitator between the ball mill and the flotation machine as the pneumatic treatment alone is not sufficiently vigorous. The early pneumatic machine installations contained Pachuca tanks for this purpose. These, however, did not prove effective, for the reason that an emulsification of the oil is required, and not merely agitation. It therefore becomes necessary to use an impeller or some form of mechanical emulsifier, and the power required to operate this machine must be added to the power consumed by the flotation machines proper.

In Montana we find that our power consumption for emulsifying and flotation is about 0.24 hp. per ton as compared with 0.15 hp. for flotation alone at Inspiration. The capacity of an impeller-type machine is materially greater when the emulsification of the oil in the pulp is done in the ball mill. In this connection an interesting suggestion was recently made by Dr. Cottrell. He suggests making an emulsion of oil and water in a special emulsifier, such as made by the De Laval people, consisting of two disks running almost in contact. The oil and water is fed in at the center and is thrown out at the circumference. Thus the work of spreading the oil through the pulp might be done more efficiently than is now the case.

For some time it seemed to us that the main point to be considered in choosing between the two types of machines was power, and that this depended largely upon whether acid or neutral or alkaline treatment were decided on. Of late, however, it has seemed to us that the impeller type of machine has another advantage, which, I recall quite distinctly,

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was cited as a disadvantage when the first Callow machines were brought out. I refer to the toughness of the froth. Most of you doubtless recollect that one of the advantages of the pneumatic machines was supposed to be that the froth breaks down quite readily, thus rendering the mineral content of the froth easier of collection.

It is becoming more evident to us, however, that this apparent advantage is actually the reverse for the reason that the coarser mineral grains tend to fall back before they can be skimmed off and are thus lost or must be recovered by tabling. We are beginning to believe that the tougher froth is a distinct advantage of the impeller machine, and our belief has been considerably strengthened of late by tests made on a disseminated copper ore from South America. It was impossible to make as lean a tailing on the pneumatic machine as on the impeller machines, owing to falling back of the coarser mineral grains.

FRANCIS S. SCHIMERKA, Clifton, Ariz.—Regarding the proposed scheme of Dr. Gahl, to precipitate the copper from a sulphate solution by means of ground limestone, I had a discussion with him a few days ago. I called his attention to the fact that the result of this operation would be a low-grade, very slimy and voluminous precipitate consisting of the rather insoluble gypsum and basic copper carbonate both in highly hydrated form, which is difficult to handle and can be worked up only by matte smelting in the blast furnace.

Dr. Gahl has proposed limestone as a precipitant for the copper in the leaching solutions to avoid contamination of the water supply to the mill which would result from returning the exhausted liquors into the mill system. I think the difficulty could be overcome by applying the acid to a de-watered thickened pulp; this procedure would assist the leaching operation and the exhausted liquors could be run to waste.

RUDOLF GAHL.—I would like to say that I agree completely with everything that Mr. Laist says, except, of course, his compliments. It occurred to me through Mr. Laist's remarks regarding the relative character of the froth produced by the different types of flotation machines, it might be worth while trying to modify the froth of the pneumatic machines by reducing the air supply, and perhaps also by reducing the frothing agent in the flotation oil mixture. I feel sure that the character of the froth can be modified to some extent, although I doubt very much if without retreatment it ever would approximate the froth of the standard-type M. S. machine. Regarding Mr. Schimerka's remarks, I might say that I know very little about smelting, and have not considered the smelting problem very carefully. All I know is that Mr. Wallace, who used to be smelter superintendent of the International Smelting Co. here assured me that he could smelt that stuff, and could pay for the lime also. Regarding the other point Mr. Schimerka brought out, about add-

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ing the acid to a thickened pulp, my impression is that still a large part of the water supply would be contaminated because we have to figure on leaching mainly very fine slime as it contains most of the oxidized copper; and it is the experience of every mill man that it is impossible to settle such slime to a consistency exceeding 3 to 1—that is, 3 parts of water to 1 part of solid matter. That would mean, if we added the acid to the thickened pulp, we would spoil 3 tons of water for each ton of leached ore.

C. E. MILLS, Miami, Ariz.—It seems to me that one of the members might tell the experiences at Chino in leaching oxidized ore.

H. W. MORSE, Los Angeles, Cal.—We have not gone far enough to say anything about what we are going to do, but I think at the next meeting of the Institute we will be able to give some very interesting results on this class of work. Dr. Ricketts has always turned down my proposals because they involved the handling of a thin pulp, settling and washing, and a great many other miseries which would certainly be involved in this class of work. But, when he saw the proposed scheme at Chino, he expressed himself, fortunately, as being satisfied that we had, provided we could make it go, decided on a scheme that met his approval, which involved no thinning of material or washing, but simply the ordinary passage of mill pulp through an agitator and flotation machine. We have not any results worthy of consideration as yet, but I feel encouraged by the laboratory work and very confident of the future of this scheme. If it will go, it will be an important process because it will enable us to treat mixed ores, provided that they contain a sufficient amount of soluble ores to precipitate on iron, and provided that they contain sufficient sulphides to pay for flotation operation.

DAVID COLE, El Paso, Texas.—I understand that Dr. Ricketts "turned down" Dr. Morse when advocating leaching processes for low-grade tailings by methods which involved first getting the copper into solution and then separating the solution from the pulp in a clarified condition, which is difficult and involves much washing with clear water, etc., resulting in thin solutions, the latter entailing difficulties or "miseries" in effecting precipitation and recovery of the metal.

On the other hand, Dr. Ricketts approves the present process which Dr. Morse is using at Chino, in which the oxidized copper is first taken into solution by the use of about 3 lb. of sulphuric acid per pound of copper digested; this copper is then precipitated upon iron in an agitated mass, which results in metallizing the copper at the expense of about 2 lb. of iron per pound of copper precipitated. The metallized copper is then in a very fine state of division in the pulp, and the remaining copper sulphides which were not attacked by the acid are also in the pulp in a very fine state of division, and the whole is subjected to flotation treatment for the removal of both the sulphide and metallized contents, thus doing

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away with the necessity of removing the solution in a clarified condition from the pulp under treatment, avoiding the washing of the pulp for complete removal of solutions pregnant with copper, and avoiding most of the "miseries" previously unavoidable.

The exhibit of this new scheme of treatment operating on a small tonnage basis, which was so kindly thrown open to examination by our party while at Chino, was very interesting.

Miami has been experimenting for some time along the same line and has achieved success in a laboratory scale. Flotation has been successfully used in the separation of ultra-fine native copper in Michigan, and there seems to be much promise in this new scheme of treatment for the recovery of copper in low-grade ores when in a mixed condition of oxide and sulphide form. As Dr. Morse has suggested, the development promises to be important, and we may be on the threshold of another important step in copper metallurgy. I congratulate him.

(Here Mr. Cole read an abstract of his discussion of Dr. Gahl's paper which was printed in the October, 1916, *Bulletin*, pp. 1870 to 1875.

RUDOLF GAHL.—Mr. Cole's question, why hydraulic classifiers were installed in the Inspiration mill, is very pertinent, and his doubt about their usefulness for the classification of the deslimed flotation machine tailings seems well justified. In defense of the installation, I will say this:

1. The expense of operating these classifiers is very low, the principal cost item being labor for their attendance.

2. It requires a very small settling capacity to settle the table tailings. At the Inspiration mill it is accomplished by three 60-ft. Dorr settling tanks, with an additional tank installed for the purpose of keeping the coarsest sand out of the Dorr tanks. This small installation handles more than 7,000 tons per day. As the reclamation is accomplished right at the foot of the mill, the reclaimed water does not have to be lifted very high, and the reclaimed cost for this water is, therefore, low.

In other words, the benefit derived from the installation of hydraulic classifiers does not need to be very great to make them pay.

It is true that when we operated our test mill, we succeeded in obtaining quite satisfactory recoveries in spite of the fact that we put the whole of the flotation machine tailings on the tables without preliminary classification. We realized, however, that the table tailings (at least the coarser sizes) were not as low in their copper contents as we hoped to have them some day. We knew, furthermore, that by grinding finer we could reduce the coarser screen sizes in copper. As grinding finer than to a certain point is an economical impossibility, we had it in view to make different sizes and to send them to separate tables. This separation we intended to accomplish in hydraulic classifiers. The coarser tailings, or

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perhaps only middlings from the tables, could then be reground and reconcentrated.

To Mr. Cole's question as to what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation, I have to reply:

That we have not made the test which he suggests, but that I am inclined to agree with him that the tables might do as well on an unclassified as on a classified feed. The hydraulic classifiers were put in as I have indicated before, in anticipation of further development of our flowsheet.

We are doing a great deal of experimental work on this special point, but have not gone far enough to predict just what the details of the additional sand treatment will be.

R. S. HANDY, Kellogg, Idaho.—I would like to ask if anyone has determined the relative economic efficiency of flotation as compared with gravity treatment on granular material.

C. E. MILLS.—I don't know whether I could answer that, but would say that through some experiments which Dr. Gahl has mentioned on the coarse feed to the tables, we have come to the conclusion that we do slightly better by flotation than by tabling of that material.

E. P. MATHEWSON, Anaconda, Mont.—I would like to say, in regard to Mr. Handy's question, that it is the practice in Montana to take out everything that is possible by means of tables or other water concentration machines; and at the Anaconda plant we take out concentrates 1½ in. size, and keep on taking out finer and finer material by water concentration until we get to the tables. What is left from the tables and not saved is then ground up and put through the flotation machines. We find by our experiments that it is better to keep the slime separated from the sand.

R. S. HANDY.—In experimenting on lead ore, I found it advisable to separate the granular material, tabling the sand material; and I wondered if that same experience had been gone through in copper ore.

C. E. MILLS.—That is similar to what we are doing at Inspiration in our callow sections. The pulp is first subjected to a primary flotation, then sent to drag belts which separate slime from sand. The sand is tabled while the slime is given another flotation treatment.

DAVID COLE.—I consider it unnecessary to first remove the slime, for the reason that after the slime has been subjected to flotation treatment there is nothing in it that a table treatment can save—no sulphides in a sufficiently fine state of division to be transported by the slime, because the previous frothing operation has removed it all, and when slime is

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"denatured" in this manner it is no longer harmful upon the table, and does not interfere with the working of the sands upon the table. The table will do exactly the same work upon the sands that it would do if the slimes were not going across and off in the rear of the sands with the excess water. If no previous division or washing out of slime is practiced we have gained to the extent of the trouble and cost that would be entailed in making the division. That is the way I view the matter.

RUDOLF GAHL.—I would like to express a doubt as to Mr. Cole's statement. Our experiences distinctly show that we get a better sand recovery by flotation if we remove the slime beforehand.*

E. P. MATHEWSON.—I beg to confirm Dr. Gahl's statement as to the practice at Anaconda.

R. S. HANDY.—I would like to say that I have separated a minus 200-mesh lead ore into granular and flocculent matter, treating the former on tables and the latter by flotation. By this treatment I have recovered 92 per cent. of the total lead, in a concentrate assaying 63 per cent. lead. By flotation of the total ore I could recover only 85 per cent. of the lead when the concentrates assayed 63 per cent. lead and in order to get 92 per cent. extraction, the best grade of concentrates I could get was 50 per cent. lead.

C. E. MILLS.—I wish to say, in response to Mr. Cole's question, that I think Dr. Gahl overlooked something that we have at the Inspiration mill—a section that is running tables on an unclassified feed. The results from that section are just as good as in the others. I think that may be due to the fact that we do not do very good classification.

F. G. COTTRELL, San Francisco, Cal.—There is one point in connection with what Mr. Handy was saying. I think what he has in mind is the distinction between sands under 200-mesh. It is more of a physical distinction than a mill distinction. I mention it because it might be something for the practical mill man in drawing a distinction between clay material and the very finely ground crystalline material. It is customary to lump under slimes everything, irrespective of its chemical properties.

C. E. MILLS.—I think Mr. Cole has done a lot of work along that line. He might give us some information about it.

DAVID COLE.—In my paper "The Advent of Flotation in the Clifton-Morenci District," I referred to that. In devising the early flow sheets for

*NOTE by Dr. Gahl: I judge from Mr. Cole's remarks in the form in which they appear on the stenographer's transcript of the discussion after being corrected by Mr. Cole that very likely he did not refer, as I understood him to do in the discussion, to the flotation treatment of a mixture of slime and sand. If this is so, my answer does not fit the case.

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Morenci we thought it would be advantageous to make the separation because we did not at the time consider flotation as at all applicable to the problem. In my paper I have alluded to the methods devised and reasons for making the divisions.

We knew, of course, that the separation could not be made upon screens. Hydraulic or washing separation I considered to be impracticable after trial of some of Mr. Overstrom's methods, and spitzkasten or pointed box or Callow tank separation is only partially successful. A better result would be accomplished by shortening the period of settling. To go into an explanation of why this is so would take too much time, but those who have been dealing with slime and know its peculiarities in settling and coagulation—its power to arrest the falling of successively larger particles of the ultra-fine sands which Mr. Handy has in mind—will realize what I mean by hastening the removal of the sands, and we developed a machine which was probably misnamed a "colloid separator," the office of which was to remove the millman's type of "colloid"—the very much diluted clay—before it should have time to settle into a mass that would seize and hold the granular material which we knew we *could* treat successfully on tables or vanners, and at the same time allow the dilute clay, which it did no good to treat by table or vanner, to escape to the tails. In the light of our present knowledge of flotation treatment, we realize that this overflow, the dilute clay material referred to, is the part best prepared for flotation treatment, and outside of the oxidized copper content would yield the lowest tails and the most complete recovery, and that it is not necessary to remove the ultra-fine sands for flotation treatment, for these are best removed by flotation also.

The colloid separator that was devised for making the separation consisted of wide thin belts dragging upon the bottom of a shallow basin (2 in. deep), across which the pulp was fed in a wide, thin, gently moving stream. The belts covered nearly the whole of the bottom of the basin (2 in. apart) and therefore practically the whole floor of the basin was (figuratively speaking) moving toward the discharge, where it rose gently above the surface of the water, received a gentle washing from clear water sprays like the sprays used upon vanners, and discharged its load upon vanners for gravity treatment, where a good grade of concentrate and a low tail were made. I think the illustrations in my paper will make the matter clear.

I have noted Mr. Mathewson's remarks concerning the present Anaconda practice, and that he has the impression that they really are desliming the feed before treatment on tables. The Anaconda flow sheet shows such a separation and the Anaconda type of conical deslimer is installed with that end in view, but like Mr. Mills' admission of a few minutes ago as to Inspiration's poor classification practice, Anaconda

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doesn't do good classification. The feed to the tables is not deslimed as it was intended it should be, the reason being that there are not enough of the Anaconda classifiers to do the work put upon them, and since the only office of these classifiers is to prepare feed for the Butchart riffle treatment, and since these tables have no office but to impoverish the ore treated by them, the complete separation of the slime from the feed is of little consequence, for all reject from the tables is taken at once to the regrinding mills where the cleaning-up work is most thoroughly accomplished by the flotation process. If the tables were making a reject to tailing, the Anaconda classifiers would have to do their full duty in desliming the feed to them, because the slimes going across the Butchart tables would result in serious losses, but since it is immaterial whether the primarily made slimes reach flotation treatment over the top of the classifiers or through the spigot, the classifiers' inefficiency and the results as to the reject from the tables are tolerable and there does not seem to be any reason to change. Obviously, it does not matter at Anaconda where the copper is taken out so long as a minimum amount of it is allowed to get away with the final tailing, and it is also obvious that with Anaconda's present practice wherein the ore is reduced by their splendid treatment scheme from a 60-lb. copper content to less than a 3-lb. copper content in the final tail on a ratio of practically 3 into 1, the chance for improvement in mill practice through modification or more perfect slime classification is extremely remote.

GUY H. RUGGLES, Miami, Ariz.—To get back to the subject of classification at the Inspiration: In the early part of the year tests were made showing that the section which treats the pulp from the flotation machines directly upon the tables, that is, without classification, did as good work as the other sections in which the pulp was classified. Since that time we have improved the settling in our drag classifiers The average of the general tailings on a number of sections for the month of July and August shows that the section which treats the flotation pulp directly upon the tables is, if anything, not quite as good as the sections in which we have classification. There are some sections which beat this section by 0.02 per cent. copper and others which beat it by only 0.01 per cent. copper. Every 0.02 per cent. which we reduce the general tailings means 180 lb. of copper per 24 hr. per section, and when this is multiplied by 18 or 20 it means quite a lot. While our classification is not very good, it has helped out quite a little and if improved we might get more out of it.

R. C. CANBY, Wallingford, Conn. (communication to the Secretary*). —At the time I was working upon flotation with the late Robert S. Towne, he told me of his interesting experience in the use of iron balls in a sort of ball mill, with iron balls used simply as an oiling device in connection with the Murex process.

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Now in the Murex process there is no question of modification of surface tension, etc., simply the oiling by preferential affinity of the sulphide surfaces that the pulverized magnetic oxide may be made to adhere to the sulphides and not to the gangue.

Mr. Towne believed that the iron balls readily became coated with the oil and that *then* the sulphide surfaces very readily became coated as the particles—through the ball-mill action—came into contact with the surfaces of the oil-coated iron balls.

May it be that metallic iron has a somewhat similar assisting function in the cases referred to by Dr. Gahl? If so, it would suggest that an oil film upon the sulphide surfaces may be an assistance, if not altogether a necessity, in the frothing flotation process.

I had not intended to give Dr. Gahl the impression, as expressed in the footnote on page 1637 of the September *Bulletin*, that the experiments which I had conducted had given to Mr. Towne the idea of the porous medium for disseminating air bubbles. It was furthest from my thought to make such a claim.

What I suggested was the use of a Frenier pump spiral for incorporating the air bubbles, having in mind the converse of the Elmore vacuum, in which I proposed to produce the bubbles by pressure instead of by a vacuum. Upon investigation, however, I found that Norris had first suggested using air bubbles in solution.

While the manner of releasing the discharge from the Frenier into a separating compartment gave possibly an effect somewhat that of the "bubble column," still it was not actually the "bubble column" in the sense the word is now understood, and I did not intend to claim it as such.

I wish I felt it proper to take the space here to more fully attest the remarkably analytical manner in which Mr. Towne studied into all of the problems which he encountered, taking as he always did, the lead in the constructive thought of his staff, with whom he worked in close touch.

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* Received Sept. 14, 1916.

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Flotation Concentration at Anaconda, Mont.

Discussion of the paper of FREDERICK LAIST and ALBERT E. WIGGIN, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 111, March, 1916, pp. 549 to 581.

O. C. RALSTON, Salt Lake City, Utah.—I have merely glanced over this paper, consequently, I am hardly in a position to discuss it intelligently. There is one thing, however, that is of interest, that is, the claim that the oil mixture which they have chosen is the best adapted to the work at hand. I would like to ask Mr. Mathewson if they still feel that that is the case—if that really does give the best grade of concentration and the best extraction for the same money, compared to any particular oil mixture which they could get.

E. P. MATHEWSON, Anaconda, Mont.—We are still using that mixture, and we consider it the most economical mixture used.

DAVID COLE, El Paso, Texas.—I would like to ask Mr. Ralston if there are some other mixtures that would be better?

E. P. MATHEWSON.—We are open to suggestions.

O. C. RALSTON.—My own paper on oils* calls attention to the fact that certain hardwood oils, especially the tars, are now being burned or wasted. A great deal of hardwood tar is available as low as 4 or 5 c. per gallon, that being its value as fuel under the boilers. As a matter of fact, a great deal of testing has been done with only the commercial products that are as a usual rule sold on the market. Very little hardwood tar is sold on the market. In asking a great many people who have tested flotation oils, I have not heard hardwood tar mentioned very often. I think I would add to that that the hardwood tars can probably be bought in quantity comparable with the sludge acid at Anaconda, and should contain a greater proportion of the frothing constituent, and at a price fairly commensurate with what they pay for the sludge acid.

E. P. MATHEWSON.—I would like to make a correction in one statement I made that there has been no change made in the mixture. This change has been made: We found that the amount of wood creosote in treating the sand from tables was extremely small, and we tried some experiments on a large scale, dropping it out and using simply the sludge acid and sulphuric acid. We found that this gave practically as good results as wood creosote. We use wood creosote in treating the slimes. We find it necessary in that operation.

DAVID COLE.—I would like to ask, what is sludge acid?

* Some Miscellaneous Wood Oils for Flotation, *Bulletin* No. 116, August, 1916.

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E. P. MATHEWSON.—Sludge acid is refuse from the refining of oil. It contains sulphuric acid and some greasy material from petroleum.

DAVID COLE.—I believe that you are making sulphuric acid very cheaply at Anaconda. Since sludge acid consists of coal oil and sulphuric acid, I have wondered if you could not compound it at Anaconda more cheaply than you can buy it.

E. P. MATHEWSON.—We have made sludge acid at Anaconda, but it is more expensive than that on the market. We got good results with the acid we manufactured.

RUDOLF GAHL, Miami, Ariz.—Some of the wood oils have the drawback that the concentrates produced with them have the tendency of retaining some of the gangue very firmly and it is difficult to settle such concentrates. We have found very few we could use. We have found some pine oils of the more refined character, but we cannot do anything at all with pine tar. Hardwood tar may have different qualities, and I would like to ask Mr. Ralston and Mr. Allen if they have given some attention to the physical characteristics of the froth made by the wood oils which they have tested.

O. C. RALSTON.—Of course, the tests reported in our paper on wood oils are laboratory flotation tests, and give the flotative value of the oil and do not discuss later mechanical treatment of the concentrates. We cannot tell what will happen later in that part of the mill. That is another point. Sometimes you cannot tell even when you get to the mill what you can do with such a froth. I think Mr. Cole's patent for breaking froth is a rather interesting sidelight on that subject. To answer Dr. Gahl directly, our suggestions are only based on laboratory experiments on the flotation problem alone.

RUDOLF GAHL.—Mr. Ralson has done so much work on the theoretical end of flotation, he might be able to answer a question which I have often put to myself. Why is it that at Anaconda they can float chalcocite ore (Is it not chalcocite ore?) with acid and when we add acid here we spoil our flotation altogether?

O. C. RALSTON.—Some day when I understand what flotation is—I will answer that.

NORMAN CARMICHAEL, Clifton, Ariz.—One point which has not been touched on in this discussion today. In going through the Old Dominion concentrator, I noticed that they were using what appeared to be caustic soda. I think if there is anyone here who can give us any explanation in regard to the use of the caustic soda, it would be interesting.

W. B. CRAMER, Globe, Ariz.—The point which Mr. Carmichael has just spoken of, I shall try to answer. I was interested in what Mr.

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Mathewson said about the use of acids at Anaconda. I believe the analyses of the Anaconda and Old Dominion slimes would be much the same. We find, as Dr. Gahl finds, that the moment we use acid in flotation, our flotation suffers considerably. In fact, it almost ceases. If we use a pulp slightly alkaline, the flotation work is satisfactory. Caustic soda is expensive, and we are using about 1 lb. to a ton. A distinct advantage that caustic soda has at Old Dominion, at present we are running on high flotation feed, averaging 3 per cent., is in the matter of settling. We are installing a new Dorr thickener, and our capacity so far as thickening concentrates are concerned is very limited. Soda has the effect of flattening out the froth considerably, allowing us to cut down the water in the launders at machine less than half, which allows us to handle the concentrates in the Dorr thickener very well. Without the use of caustic soda, we may lose 4 or 5 tons of concentrates a day. It goes over and is recovered later in the secondary tanks. Why the soda should have this effect at the Old Dominion and the opposite at other places is hard to explain. We cannot use acid. We get better results with the caustic soda. In other places the opposite is found. At Nacozari a few years ago we were not able to get a satisfactory froth without acid. If we put acid in small amounts, the pulp still being non-acid, the froth was light in character; but increasing the amount of sulphuric acid until pulp was acid to methyl orange, a splendid froth was obtained.

C. W. MERRILL, San Francisco, Cal.—But the use of caustic soda is solely for the purpose of decreasing the amount of water from the overflow?

W. B. CRAMER.—Caustic soda does increase our extraction. So, in a matter of dollars and cents, it pays us to use caustic soda, but it brings the cost per ton for treatment up about 5 c. per ton. It does increase our extraction, increases the copper content of the froth and lowers the insoluble content.

C. W. MERRILL.—I understand that the purpose is threefold; to cut down the water, increase the extraction, and clean the concentrates.

R. S. HANDY, Kellogg, Idaho.—I would like to suggest that in the treatment of refining petroleum, acid is used, which before the treatment is finished is neutralized by caustic soda, so that the residue is sludge acid. If I am not mistaken, it contains a considerable quantity of caustic soda. That has been my experience in California. I would like to know if that is true.

O. C. RALSTON.—The acid sludge from petroleum refining is usually a product obtained by the addition of strong sulphuric acid to the partially refined petroleum products in order to remove certain impurities,

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and is usually removed before sodium hydrate is added to the remaining oil to neutralize the small amounts of sulphuric acid held by the oil. Hence it consists of strong sulphuric acid combined with some of the dark tarry constituents of the oil. Good flotation acid sludge comes from the California oil fields and titrates about 50 per cent. H_2SO_4. The other constituents are apparently the sulphonated asphaltic compounds contained in the oil. On that account sludge acid does not contain sodium hydroxide. Turning to the question of the use of sodium hydroxide in the flotation pulp and the reason for the good results obtained, I think that an explanation is easy. This substance, as well as other substances of alkaline reaction, like sodium silicate and sodium cyanide, is used in a number of mills which have a great amount of clay and other finely divided material in the ore pulp treated. Such an ore pulp is usually partially flocculated and sodium hydrate is known to be a powerful reagent for deflocculating certain gangue slimes. That is, its hydroxyl ions are adsorbed into the film of water in contact with the surfaces of individual particles, in greater proportion than are the sodium ions. This amounts to giving these particles a stronger electric charge of the negative sign and they fly apart (are deflocculated). In case any sulphide particles had been entrained in the flocs of gangue, this deflocculation liberates these sulphide particles for flotation. This explains the statements by Mr. Cramer that he gets better extraction and higher grade of froth by the use of caustic alkali in the pulp. I am informed by other operators that it also reduces their consumption of flotation oil as much as 50 per cent. in some cases. This seems reasonable, as the sulphide particles are all liberated and ready to be oiled so that oil will not have to penetrate to the center of a floc to oil the entrained particles of sulphides. Further, the great number of oils used are more easily emulsified in alkaline solutions. In the cases where the use of sodium cyanide has proven beneficial I am not sure that the alkaline reaction of the reagent is the sole contribution of this reagent to improvement of the flotation, but the case is interesting because Charles Butters and associates claim that it is an impossibility to float in the presence of cyanide. In at least two instances coming under my observation, the flotation of the sulphide minerals was improved by the addition of cyanide. It is just another instance of being told that certain things cannot be done in flotation, only to discover later that with proper care and right conditions they can be done. Don't believe any one who tells you that you can't float under some particular conditions. He may be partially right but the statement of negative results should always receive the qualifying phrase, "as far as I know, and to the best of my experience." For instance, Dr. Gahl claims that acid spoils flotation at Inspiration. I feel confident that the right conditions for an acid flotation of that ore could be found.

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DAVID COLE.—When we were at Chino a few days ago, we saw flotation machines handling vanner concentrates in which they were using alkali and rosin as the flotation agent. Mr. Ralston looked at this, and I would like to have him tell us what we saw.

O. C. RALSTON.—That solution is a solution of sodium resinate, the idea of the alkali being to get the rosin into solution. The rosin has a very good effect on the froth, giving a particularly stable froth. They were working under conditions where the oil froth wanted to die, and the addition of the rosin was the proper thing to bring up its strength and allow it to rise until it got over the discharge board. I think there was no intended significance in the additional alkalinity. In that case the alkali was added for a particular purpose, simply as a solvent for the rosin.

THE CHAIRMAN.—I understand that Mr. Ralston wishes to get some questions answered by the members of the Institute, and I will ask him to kindly present those questions.

O. C. RALSTON.—A great many of the questions I am interested in have been asked and answered this afternoon, so that you will all welcome the fact that the list can be cut down. This list was presented by D. A. Lyon, who sent them out in a circular letter all over the country. So far I have not heard of anyone who has prepared a list of answers for publication. I don't see why people need to be afraid to answer them. The other day somebody said, "it is safe to talk about theories of flotation, because none of us know anything about them." I will only pick out a few questions. The first one is this: What is the effect of dilution of pulp with water on the flotation of the minerals contained? The reason for asking that question is, of course, obvious. There are enough large-scale operators who must have tested this question out. So, it will be interesting to get the consensus of opinion. I think that Dr. Gahl might help us.

RUDOLF GAHL.—Well, I would say offhand that the effect of dilution is to make a cleaner concentrate, and to make it more difficult to produce concentrate. I think it takes more oil to produce the same amount of concentrate from a dilute pulp than it takes to produce it from a thicker pulp.

O. C. RALSTON.—Might I supplement by asking you what determines the amount of oil necessary for flotation? Is it the amount of water you are using, making a certain strength of solution of frother in water, or is it the amount of mineral in the ore? If you had an ore consisting of 50 per cent. mineral, would you use more oil than if it contained 5 per cent.? What amount of oil must be used in flotation?

RUDOLF GAHL.—What amount of oil must be used in flotation? I do not feel qualified to answer Mr. Ralston.

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O. C. RALSTON.—The Butte and Superior ore, containing 30 per cent. of mineral, as compared with the Inspiration ore with 5 per cent. or 10 per cent. of mineral, nevertheless uses less oil, if anything. So, the proposition would be that the amount of mineral in the ore is not determinative of the amount of oil necessary. It must be the amount of water in the pulp which is determinative of the amount of oil necessary.

DAVID COLE.—I think the kind of oil has a lot to do with the amount used.

O. C. RALSTON.—That is a question not of theory but of experience. For instance, the amount of pine oils being used in the Coeur d'Alene district is often less than ⅕ lb. oil to the ton of ore, while at Anaconda they use several pounds of oil per ton. At Anaconda they use a considerably less expensive oil.

A. P. WATT, St. Francois, Mo.—Mr. Ralston has asked the question whether the amount of oil used in flotation is proportional to the water or to the solids in the feed to a flotation machine. My experience with the flotation of lead ores may be of interest. In one particular case a flotation machine was receiving a feed with a ratio of solids to liquid of 1 to 7. This ratio was later changed to 1 to 3.5 and the amount of oil used was decreased approximately 50 per cent. This particular case would seem to indicate that the amount of oil used in flotation is proportional to the water rather than to the solids in the feed. I also found that the extraction was increased by the use of a thicker pulp. This result would be expected as the volume of pulp passing through the machine was decreased, thus permitting it to be acted upon for a longer time than would be the case with a thinner feed.

DAVID COLE.—Speaking of the small amounts of oil—we carried out some experiments the other day at El Paso, and on a density of 7 to 1, approximately, on an ore that carried 6 per cent. lead, 9 per cent. zinc, and 1.3 per cent. copper, with the use of $\frac{3}{10}$ lb. of cresylic acid per ton, we were able to take out a large part of the lead. Then, by the addition of $\frac{3}{10}$ lb. of No. 350 pine oil per ton, we were able to take out nearly all of the zinc. The products made assayed as per the accompanying table.

	Per Cent. of Weight	Ounces Au	Ounces Ag	Per Cent. Pb	Per Cent. Cu	Per Cent. Fe	Per Cent. Zn
Lead concts.....	12.86	0.110	12.85	35.7	7.32	7.7	13.0
Zinc concts.....	25.08	0.025	2.67	4.4	1.30	3.9	23.6
Tailing.....	59.68	0.005	0.45	0.3	tr.	3.0	1.9

NOTE.—This was done with a single operation without the use of cleaner for the concentrates.

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There was very little oil used—a total of ½ lb. per ton of ore. If we had used a larger amount of oil, we would not have been able to get that separation. We put in just enough cresylic acid primarily to get that result.

C. E. MILLS.—I think Mr. Gottsberger might contribute something. The fact is that the Miami Co. and the Inspiration Co. are treating similar ores, except we have not as much copper as they have. We use quite a little more oil in flotation than Mr. Gottsberger does; and I think, perhaps, they have got more water in the pulp. Is that about the condition, Mr. Gottsberger?

B. B. GOTTSBERGER, Miami, Ariz.—I am sorry to say I could not give the density of the pulp.

C. E. MILLS.—We are using 1:3.

B. B. GOTTSBERGER.—Our oil mixture is much thinner because we are using less coal tar. I think we have found that with a thick mixture composed very largely of coal tar it is really necessary to add the oil in the grinding mill. We do find, however, that it is not essential to add these oils in the grinding mill in order to get good flotation work. At present the necessary mixture is obtained by adding the oil in a bucket elevator.

C. E. MILLS.—Can you give us another question, Mr. Ralston?

O. C. RALSTON.—I might ask this question, which would supplement Mr. Handy's question: Are ores which contain very much fine colloidal material harder to treat successfully than granular ores? As far as I know, they usually are. Dr. Gahl has mentioned something about the treatment of them. I would like to pick on Dr. Gahl again to answer that.

RUDOLF GAHL.—I would say that colloidal ores are harder to treat than granular ores.

O. C. RALSTON.—There is a question in my mind. Is it possible to treat by any method of flotation colloidal material? Would it be possible to deflocculate that material and separate the granular material? In our experimental laboratory we did work on material sent us by Mr. Watt which approximated that condition. It was from Missouri, in the disseminated lead section, where the leady limestone had weathered away, leaving practically nothing but clay. The galena oxidized to lead carbonate, and Mr. Allen's attempts to sulphidize and float that ore were completely unsuccessful. It was practically nothing but a clay—about 100 per cent. colloidal material. So, a question arose: How much colloidal material is allowable in an ore?

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C. W. MERRILL.—Do you mean only colloidal gangue or the whole ore to be of colloidal nature?

O. C. RALSTON.—This question is very well put. No one seems to have studied the question as to whether only the gangue in an ore is colloidal. "Colloids" are blamed for a lot of trouble and it occurred to me that whether the flotative minerals are reduced to colloidal size and condition, or not, colloidal gangue might cause trouble.

W. B. CRAMER.—Our experience at the Old Dominion on primary slimes differs from that at other plants. Our mill feed contains about 15 per cent. minus 200-mesh material which may properly be called "primary slime." This slime was separated in the first compartment of our fine jigs, thickened in a Dorr thickener and treated separately from the secondary mill slimes by flotation. An extraction of 84 per cent. was obtained in the flotation treatment of this purely primary slime. Hence it may be stated that the primary slime at the concentrator of the Old Dominion Copper Mining and Smelting Co. is as readily adapted to flotation as the secondary slimes.

C. W. MERRILL.— I can conceive of colloidal ore where the mineral values were extremely finely divided that could not be treated.

A. P. WATT.—Referring to the ore from Missouri of which Mr. Ralston has just spoken. I believe that Mr. Ralston and Mr. Allen were unable to make any extraction by flotation on the sample of slime I sent them. When, however, we took a more granular sample we were able to make some separation. We found that a good extraction could be made on the granular portion if the colloidal portion were first eliminated. The colloidal portion of the ore, however, appeared to prevent flotation.

The sample of slime submitted to Mr. Ralston was extremely fine. Although it contained 2 per cent. lead no signs of cerusite could be detected when the slime was treated on a vanning plaque. Whether or not the cerusite was in a "colloidal condition" I do not know.

F. S. SCHIMERKA, Clifton, Ariz.—In connection with the question brought up a short while ago as to the different quantities needed of oil to successfully float, I wish to ask one question. Has it ever been noticed by the flotation experts present today whether it makes a difference as to what density the pulp is in when the oil is added in the grinder or mixing machine—whether a pound of oil goes farther at a high density?

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A New Flotation Oil and A New Source of Flotative Agents

Discussion of the papers of MAXWELL ADAMS AND G. H. CLEVENGER presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1683 to 1684 and 1685 to 1692.

OLIVER C. RALSTON, Salt Lake City, Utah.—We are indebted to these two gentlemen for the work which they have done with the oils derived from sage brush. Their work should not be confused, however, as the work of Mr. Adams deals with steam-distilled oils and that of Mr. Clevenger with destructively distilled oil, largely. In other words, Mr. Adams' oil is probably the original essential oil that existed in the brush before distillation while Mr. Clevenger's oil not only contained this essential oil but also some products of destructive distillation of the wood constituents.

I had noticed the work of Mr. Adams on attempting to separate the various constituents in the oils of some of the Western conifers and had sent to him to ask if he could send me small samples of some of these pure products. This he very kindly did, and with them included some samples of oil from the common sage brush, whose characteristics he has here described, as well as a sample of oil from the common "rabbit brush." Only the sage brush oil proved of much value for flotation, but it was very powerful, and I so informed Mr. Adams. Even in amounts as small or smaller than $\frac{1}{10}$ lb. per ton of ore, it gave very quick and positive flotation of many ores, with high-grade concentrates and good extractions. It proved to be especially desirable for the flotation of carbonates of lead and of copper which had received an artificial coating of sulphide by treatment with a solution of a soluble sulphide.

I also received a sample of crude pyroligneous acid from sage brush prepared by Dr. W. C. Ebaugh, of Salt Lake City. It did not do very good work in flotation, but it is possible that by concentration of this product to obtain the residual tar a good flotation oil might result. This has not been done as yet and I am sorry to see that Mr. Clevenger has not done this. It might be that the dissolved tar, added to the settled tar, would make a total yield of flotation oil higher than the 4 per cent. which Mr. Clevenger feels is a safe estimate. The importance of this point will be seen in the following discussion of the possible costs of sage brush oils for flotation.

I have been fortunate in obtaining the approximate costs of collecting sage brush, as well as the costs of operation in the ordinary wood-distillation plants. I was given the former by E. H. Snyder, who with a number of associates cleared some land of sage brush and made potash from the

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ashes of the burned brush. The latter were given me by R. C. Palmer, formerly of the Forest Products Laboratory of the Department of Agriculture.

The clearing of land of its sage brush is ordinarily accomplished by means of a tractor pulling a special frame made up of railroad rails. This breaks off the bushes so that they can be collected in hay ricks and hauled to a central point. For merely uprooting the plants and burning them on the spot the cost is about $4 per acre, but to recover them and take them to a central point ready for burning or distillation, costs about $6 per acre.

Mr. Snyder further informs me that with the average 4-ft. stand of brush in southeastern Nevada the yield per acre is about 7 tons of brush. I am informed that there are considerable stretches of country around Bear Lake in Idaho and Utah where the brush is nearly 10 ft. high. Hence considerably higher yields might be possible. However, we can take as a safe figure $1 per ton for cutting and collecting the brush to a central point.

The cost of destructive distillation of hard wood is in the neighborhood of $8 per cord (4,000 lb.) of wood, or about $4 per ton—possibly $5 per ton of wood. While the sage brush might be more bulky than the wood, and hence cause higher costs, this factor can not be estimated at the present time and it is probably best to count on a distillation cost of $5 per ton.

That would make the total cost of treatment of each ton of sage brush about $6, and the yield, according to Clevenger, is 4 per cent. or 80 lb. of tar. This would mean about 10 gal. produced at a cost of $6, or $0.60 per gallon, or 7.5 c. per pound. This is rather high but is comparable with the present price of pine oil, of which sage brush oil seems to be the full equivalent, if not the superior. With most ores, less than 0.5 lb. of sage brush oil should be needed per ton of ore.

If Clevenger's figure of 4 per cent. yield of oil could be raised to 6 per cent. by addition of the dissolved tar from the pyroligneous acid, the cost of the oil would be about 43 c. per gallon. Further, if greater yields of brush from each acre were obtainable, the costs would be cut slightly, although this point is not of so much importance, as the principal costs are due to the distillation. It is also possible that in some cases owners of the land would be glad to pay $1 to $2 to have their land cleared of brush, where it would usually cost them about $4. Only skillful management and the choice of the proper regions for operations would probably bring in this latter source of income.

There is one other source of income from the products of the sage plants, namely, the potash. Mr. Synder informs me that he found the potash in carefully burned brush ashes to amount to 15 to 20 per cent. of the total weight. Most of this was soluble in water but the remainder

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was "acid soluble K_2O." The ash amounts to 7 to 10 per cent. of the weight of the plants. Hence the K_2O content of the brush can be estimated as from 1 to 2 per cent., or 20 to 40 lb. per ton of brush. In normal times potash is worth about 3 c. per pound. This would make the value of the potash in the brush about $0.60 to $1.20 for every ton. The cost of leaching and crystallizing the potash from the ashes is unknown, but might possibly be $3 on every ton of ash or 20 to 30 c. referred to every ton of brush. It is not impossible that the burning of the sage brush charcoal under the boilers of the distillation plant, followed by leaching of the ashes would yield potash of sufficient value to allow a profit, and hence allow of production of sage brush oils for flotation at a lower net cost than the above rather pessimistic figure. One difficulty at the potash end of the work is that it tends to be carried away in the combustion gases and a Cottrell precipitator might have to be used in order to insure recovery of all the potash.

The above is merely a suggestion to show what might be expected if one were to attempt the commercial production of sage brush oil. Rather high costs were assumed, but it is certain that other sources of expense than those enumerated will make such a pessimistic figure desirable, for the sake of safety.

About 10,000 gal. of steam-distilled pine oil are being used every month in the United States for flotation purposes and more would be used if the price were lower. The last quotation that I heard for pine oil was 67 c. per gallon, f.o.b. New York. If sage brush oil could be produced for 40 c. per gallon it is probable that the market would jump to at least 25,000 gal., or roughly 1,000 gal. per day. With this would be produced 3,000 to 4,000 lb. of potash, an amount which is much less than 1 per cent. of the total consumption in the United States. Hence the sale of potash from such a source should have no bad effect on the market. On the other hand, if the oil could be produced for 25 c. per gallon, there is no reason why its use should not amount to 10 times the consumption of the oil at a 40-c. rate.

It is to be hoped that these figures are attractive enough to cause experimentation on a reasonably large scale, to determine whether operations can be profitable. Only by such a method can it be estimated whether we have a new source of flotative agents and a new source of potash.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Conservation of Phosphate Rock in the United States*

BY W. C. PHALEN,† PH. D., WASHINGTON, D. C.

(New York Meeting, February, 1917)

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INTRODUCTION

NOBODY will dispute the fact that the conservation in every legitimate manner of our valuable high-grade phosphate-rock deposits is a present-day problem of importance.

The table and curve, given herewith, show that during the past 10 years the exportation of high-grade rock has averaged very close to half the total output of the entire country. The bulk of this exportation is from Florida, for obvious reasons. It is plain that the deposits in this State, more particularly, are being wastefully depleted under a system of selecting the cream of the product for exportation to Europe, leaving the comparatively low-grade rock, running from 65 to 70 per cent., and under, in bone phosphate of lime, for our own fertilizer manufacturers to work up when all the best rock is gone.

In this paper the writer has endeavored to make a point of describing in some detail the important methods of production and conservation in Tennessee. These ought to prove of educational value to our American agriculturists and fertilizer manufacturers, and should result in a demand for the highest-grade rock, both for direct application to the soil and for use in making acid phosphate. It is certainly evident that the European

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manufacturer is alive to the situation and is demanding the highest-grade rock from the Pacific Islands and Florida. The cost of transporting in this country low-grade rock and the acid phosphate resulting from it, is another factor which should appeal to the self-interest, if to no higher motive, of the American producer, and user as well. This factor, as well as the important one of keeping our high-grade rock at home, is the fundamental reason for a change in our policy with reference to our high-grade phosphate rock.

PRODUCTION AND EXPORTATION OF PHOSPHATE ROCK

Since the beginning of phosphate-rock mining in the United States, there has been a total output of 48,457,906 tons, more than half of which has been produced in the past 10 years. During this 10-year period there has been an exportation of nearly 11,000,000 tons, or about 43 per cent. of the marketed production in the same period.[1] The exported material does not represent average grades, but the highest-grade rock, running 77 per cent. and more in phosphate of lime (bone phosphate) and 3 per cent. and less in iron and alumina.

Production and Exports of Phosphate Rock with Ratio between Them, 1905–1914

Year	Production, Long Tons	Exports, Long Tons	Percentage
1905	1,947,190	934,940	48.0
1906	2,080,957	904,214	43.4
1907	2,265,343	1,018,212	45.0
1908	2,386,138	1,188,411	49.8
1909	2,338,264	1,020,556	43.6
1910	2,654,988	1,083,037	40.8
1911	3,053,279	1,246,577	40.8
1912	2,973,332	1,206,520	40.6
1913	3,111,221	1,366,508	44.0
1914	2,734,043	964,114	35.0
Total.....	25,544,755	10,933,089	43.0

METHODS OF CONSERVATION

Introductory Note

There appear to be many differences of opinion among soil chemists and agriculturists as to the form in which phosphorus shall be applied

[1] Figures for 1915 are not included in these computations, for the reason that the phosphate-rock industry was in an abnormal condition during that year and also during the latter part of 1914. The production of phosphate rock during 1915 was 1,835,667 long tons, and the exports 253,549 tons, or 13.8 per cent. of the production.

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to the soil. One group argues for the use of superphosphate, especially where quick returns are desired, and another for the use of ground rock or

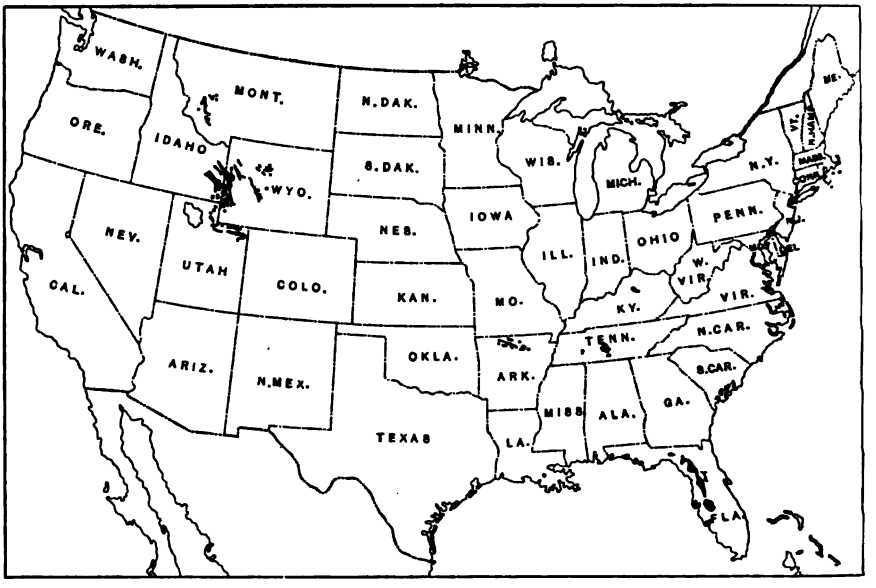

FIG. 1.—MAP SHOWING THE LOCATION OF THE PHOSPHATE-ROCK DEPOSITS OF THE UNITED STATES.

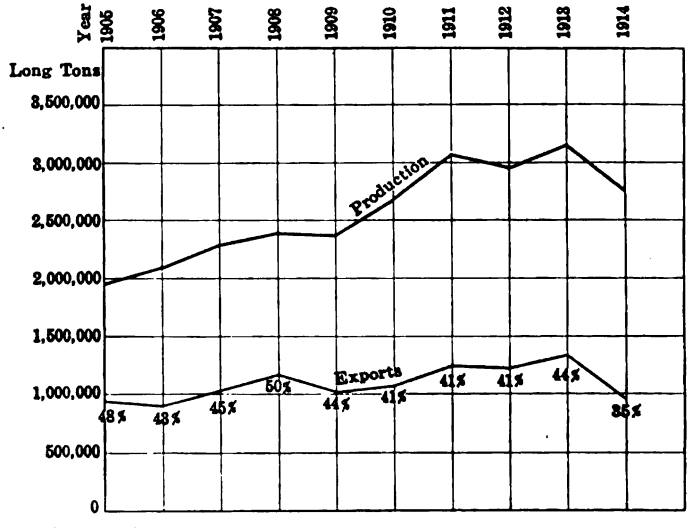

FIG. 2.—CURVE SHOWING PRODUCTION AND EXPORTS OF PHOSPHATE ROCK, 1905–1914. FIGURES ON EXPORT CURVE INDICATE PERCENTAGE OF PRODUCTION LEAVING THE COUNTRY.

"floats," especially where permanent results are the object. Without considering the merits of either side of this question, it is certain that the conservation of our phosphate resources must be of interest to both.

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The conservation of phosphate rock will be considered in the following pages, under the heads of (1) mechanical, and (2) chemical methods of conservation, and (3) the use of substitutes, either natural or manufactured, as a source of phosphate of lime or phosphorus.

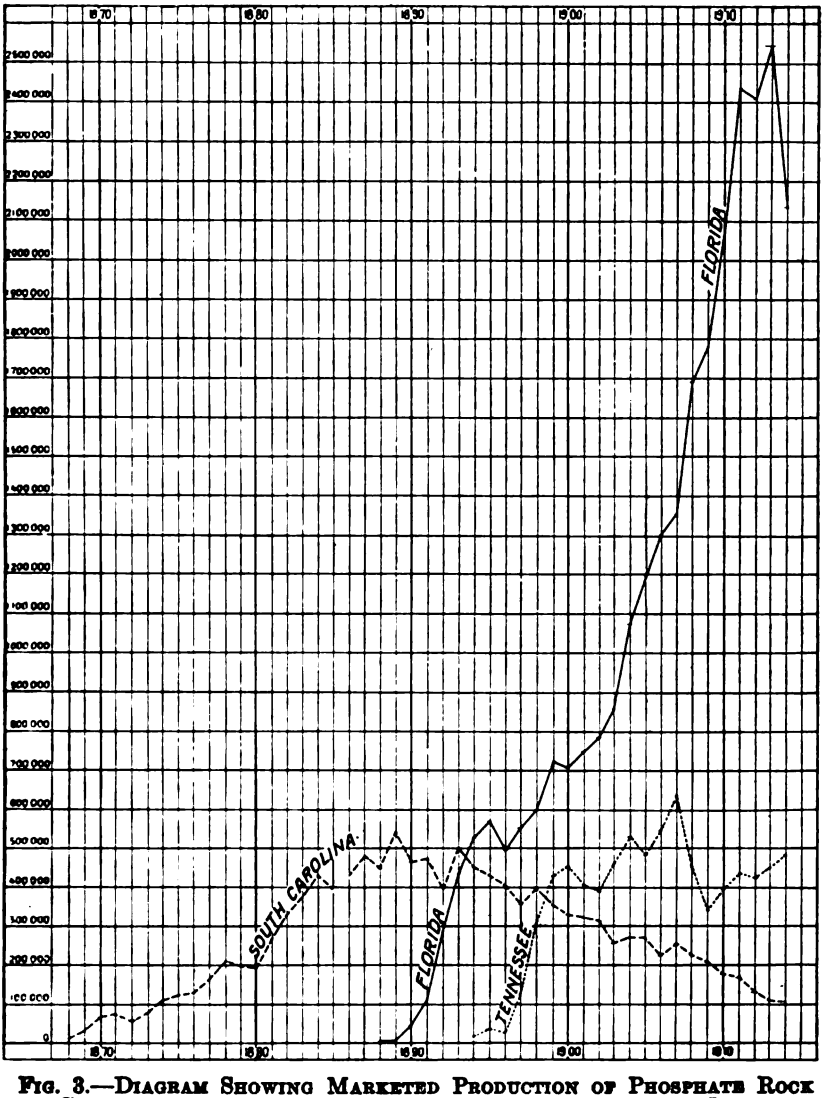

FIG. 3.—DIAGRAM SHOWING MARKETED PRODUCTION OF PHOSPHATE ROCK IN SOUTH CAROLINA, FLORIDA AND TENNESSEE IN LONG TONS FOR YEARS INDICATED AT TOP AND BOTTOM.

MECHANICAL METHODS OF CONSERVATION

It is well known that in the early days of phosphate mining in the more important phosphate fields of the United States, there was a large

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waste of good material. In many places this waste is still going on. In certain places the material once thrown aside is in such condition that it may be reworked and is actually being reworked. In other places, it is lost beyond all hope of recovery in so far as can be told at the present time. The devising of methods to prevent or reduce such losses and yet maintain the grades set by commercial standards is one of the problems that has faced and is now facing the phosphate-rock miner; and in the following pages will be given descriptions of the methods by which the problem is being met in Tennessee, which may serve as a type of what is going on in the other important phosphate-producing States.

In speaking of the early wasteful methods employed in Tennessee, W. H. Waggaman of the Bureau of Soils[2] says: "For years the richest of brown rock in the Mount Pleasant region was worked by hand, and only when these deposits were considered to be nearly exhausted did the operators seem to realize the crudity, wastefulness, and inefficiency of the methods they were using. Even now a few small firms and farmers are employing the pioneer method of shaking out all rock not held by the tines of a potato fork and drying the larger pieces in the sun or on ricks of wood."

The same writer, in speaking of the waste connected with the mining of hard rock in the Florida phosphate field,[3] says: "In order to meet the present demand for a high-grade product, a vast amount of phosphatic material is thrown aside. The marketed product is probably not more than 15 per cent. of the total material mined. The remainder, consisting of sand, clay, and the phosphates of lime, iron, and aluminum, is washed out upon a waste pile. This discarded material varies in its content of phosphoric acid, but seldom if ever contains less than 10 per cent. The actual amount of phosphoric acid discarded, therefore, is almost twice as great as the quantity saved. This enormous amount of low-grade phosphate will no doubt eventually be used."

In speaking of the waste connected with the mining of pebble phosphate in Florida, the same writer makes the following observations:[4]

"The percentage of phosphoric acid washed away in preparing pebble phosphate for the market is fully as great as that wasted in mining hard-rock phosphate. In the vicinity of the large phosphate washers many acres are covered to a considerable depth with this detritus. The author made no determination of the exact composition of this finely divided material, but it resembles the wash from the hard-rock mines. . . . An analysis, however, was made of the material having a diameter greater than 2½ in. discarded before the material containing the phosphate goes through the washing process. The composition of a sample thrown

[2] *U. S. Bureau of Soils, Bulletin* No. 81, p. 9 (1912).
[3] *U. S. Bureau of Soils, Bulletin* No. 76, p. 14 (1911).
[4] *Loc. cit.*, pp. 21–22.

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from the picking table at one of the plants was also determined." These two analyses are given in Table 1.

TABLE 1.—*Chemical Analyses of Discarded Material Connected with Preparation of Florida Pebble Phosphate*

Description	Analysis				
	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Fe_2O_3, Per Cent.	P_2O_5, Per Cent.	$Ca_3(PO_4)_2$, Per Cent.
Clay balls, 2½ in. in diameter, containing phosphate pebbles from discard.....	44.10	5.24	1.78	17.60	38.54
Material from picking table..	50.58	5.18	12.96	7.20	17.68

In the Mount Pleasant phosphate field at the present time, and probably in other parts of the Tennessee brown-rock phosphate areas, changes are being made that will result in leaving little or no wasted phosphate rock in the ground. Some phosphate is going into the waste ponds, but, without doubt, the time will come when all this material will be reworked, and even now some companies are working or are planning to work these old tailings. The modern mining and milling methods of the last decade are revolutionizing the industry and incidentally conserving this valuable fertilizer material. They are in striking contrast with the crude and wasteful methods formerly employed in the brown-rock field. Though the large operators are using up-to-date methods, even now some of the small operators are employing the old-fashioned hand methods which in the past resulted in the loss of much valuable rock.

The object, of course, in preparing phosphate rock for market is to remove as much of the clay, chert, and limestone as possible from it. Theoretically, it is possible to remove all these impurities, but this is not practicable, especially in the case of the clay. There is no sharp division between the finest phosphate sand and the clay, and it would obviously be wasteful to carry the process of obtaining the fine sand up to or beyond the point where the cost would offset the value of phosphate obtained. This is one of the practical considerations connected with the modern conservation of phosphate rock which perhaps has not always been given just and deserved consideration.

The point beyond which it is not practicable to carry the preparatory treatment is not fixed, and standards vary from time to time, and probably at a given time, among individuals and corporations. Thus, in the phosphate-mining industry as practiced in Tennessee in the early 90's, rock was discarded which has a high value today, and the former apparent lapses from the highest standards have in the course of time proven to be not lapses at all, but simply conditions imposed by the trade and the

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times. In other words, the phosphate once discarded is now being utilized. The open-cut or surface method of mining brown rock, as practiced in the Tennessee field, with which the writer is more especially familiar, is peculiar in this respect, and the generalizations made do not cover any other classes of mining, and certainly will not apply to underground mining in general.

Grades of Brown Phosphate Rock

Most of the rock from the Mount Pleasant field is shipped in three grades, namely, those containing 72, 75, and 78 per cent. of calcium phosphate. Five per cent. of iron oxide and alumina is the maximum allowed, and this is usually referred to as "I and A" in the trade and in commercial analyses. At one time only 78 per cent. rock was shipped from the Mount Pleasant field, and rock of this grade is still known as export rock. The guaranteed content in phosphate of lime, "bone phosphate," or BPL, as it is commonly referred to in the trade, next fell to 75 per cent., and at the present time many of the companies are finding it difficult to ship this grade exclusively, and the life of the 75 per cent. rock is limited. Every per cent. of iron oxide and alumina less than the 5 per cent. limit is regarded as equivalent to an additional 2 per cent. of calcium phosphate, for it is considered that in the subsequent treatment of the phosphate in the manufacture of fertilizers the harmful effect of 1 per cent. of iron oxide and alumina offsets the good effect of 2 per cent. of calcium phosphate. If there is more than 5 per cent. of iron oxide and alumina, the superphosphate becomes gummy and farmers find it difficult to drill it into the land.

Preparation of Phosphate Rock for Market, with Phases of Conservation Involved

There are many stages to be considered under the heading of preparation of phosphate rock for market, but they may all be subdivided into three major operations as follows: (1) Removal of overburden, (2) mining, and (3) milling, in which is included drying. The present methods of utilizing and thus protecting from loss, or conserving, the brown phosphate-rock supplies, especially in the Mount Pleasant field, naturally are included under the above headings, and, therefore, will be described in connection with them so far as this can be done.

Removal of Overburden

The overburden of the brown rock in the Mount Pleasant field varies from 1 ft. up. Usually it is less than 20 ft., but a thickness of 30 ft. is known, although that is excessive in places where mining is now in prog-

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ress. The methods of removal of overburden are diverse. Under exceptional conditions, the old-time crude and expensive hand methods have to be resorted to, but in most places, and especially where virgin ground is being opened, operations are conducted in the most up-to-date fashion. Where the overburden is not very thick or hard, it may simply be plowed up and removed with scrapers, or it may be loosened with dynamite and then removed with scrapers. A favorite method of getting rid of the overburden, used especially in ground that is being reworked, is to first "hog" or undercut it, pry it off with bars, and then scrape or carry it away.

The drag-line excavator and the steam shovel are types of up-to-date machinery used in removing overburden in this field. The hydraulic method is also used. Both the overburden and the rock itself are removed by this last-named method, which is simplicity itself in action and which requires a minimum of labor in operation, usually one man to handle the hydraulic gun and two to keep the sluices clear. As practiced at the plant of the Blue Grass Phosphate Co., in the southern part of the Mount Pleasant field, the rock is mined out in small areas and the overburden from one area is washed into the mined-out cavity next to it. The resultant topography is level. The land is thus conserved for farming purposes for future generations and, indeed, greatly improved, for the phosphate sand and rock, formerly below the subsoil, is thoroughly incorporated in the soil and the fertilizing value of the phosphate thus rendered available in time.

Methods of Mining

Much of the mining in the Mount Pleasant field has to be done by hand on account of the method of occurrence of the brown rock. The steam shovel has not proved successful because it cannot discriminate as to grade of rock mined, with the result that much clay and flint get into the product, and have to be subsequently removed. The cantilever adjunct to mining, which is employed at the plant of the Hoover and Mason Phosphate Co., is unique, there being only one in this field. The hydraulic method of mining is used at two plants and has many advantages, as pointed out under the preceding topic. These mechanical methods of mining and removing overburden, which have cheapened operating costs, have played the major part in conserving Tennessee brown rock.

Reworking Deposits.—The brown phosphate rock occurs in blanket form. From the base of this blanket, so-called deep "cutters" project like a network of roots, into the underlying limestone, the irregular projections of the latter, upward and into the phosphate rock, being known as "horses." In the early days of mining, all the phosphate rock occurring between the limestone "horses" was left, owing to the difficulty in mining it. Moreover, nearly everything that went through the tines of

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a phosphate fork or a 2-in. screen was discarded. The latter material, for this reason, has come to be known as "screenings" or "throwbacks," and at certain plants this is now being worked. The screenings or throwbacks can be easily distinguished from the normally occurring plate or lump rock by its mixed-up or heterogeneous appearance, the lump rock being scattered irregularly throughout the mass. From the fact that the throwbacks represent material that has been worked over and from which the larger lump rock has been removed, it follows that the proportion of muck, sand, and clay is larger than in unworked territory.

On the Ruhm Phosphate Co.'s property, in the northeastern part of the Mount Pleasant field, the worked-over phosphate deposits are located on one of the terraces that surround the town.[5] This location at the top of a hill is such that the ore can be handled easily by gravity, and the hydraulic method of mining is therefore employed. The rock appears practically at the surface in quantity, and in addition to the muck left from early operations it contains much small lump rock, discarded in the days of early mining. The limestone horses often get in the way and have to be blasted out, but this is not difficult, owing to the loose or platy character of the phosphatic limestone associated with the brown-rock deposits. In addition to the hydraulic method, which can be employed only in certain favorable locations, hand mining is also employed. Where hand mining is practiced, the ore is usually screened on the spot where the miner is at work. The fine material passes through the screen and is saved and washed; and the coarse rock is hauled away and dried by burning on ricks of wood in the open, thus saving rehandling in the mill. The lump rock, as mined, usually contains from 20 to 21 per cent. of moisture, and drying it in this way reduces the moisture to 1 per cent. or less.

At the Century plant of the Federal Chemical Co., on the Williamsport Pike, northwest of Columbia, screenings are now being worked on an extensive scale.

The accompanying analyses show the content in phosphoric acid and phosphate of lime, of material left in earlier mining operations but now being worked at the plants mentioned above. It is certain that these analyses do not exactly represent the standard in phosphate content of a great deal of the material which is being reworked in this field. It is doubtful whether any hand sample can adequately represent the normal content in calcium phosphate of this material, owing to the difficulty of properly apportioning the lump rock and the muck. It is more than likely, however, that very low-grade material can and will be profitably worked, provided some of the cheaper methods of operating can be brought to bear upon it. There is also given in Table 2, an analysis of a sample of phosphate muck. This material is found in larger proportion

[5] The field observations were made in the Fall of 1914.

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in the throwbacks than in the normally occurring rock, for the reason that in the original mining operations only the lump rock was saved. The writer has been told that the recovery of the phosphate in the muck handled is about 50 per cent. on the wet basis and about 40 per cent. on the dry, assuming that there is from 20 to 21 per cent. of moisture present. Unless the muck runs well up in calcium phosphate, it hardly pays to handle it.

TABLE 2.—*Analyses of Brown and Muck Phosphate from Reworked Deposits, Maury Co., Tennessee*

(W. C. Wheeler, Analyst)

	P_2O_5, Per Cent.	$Ca_3(PO_4)_2$, Per Cent.
No. 2	21.16	46.24
No. 27	31.88	69.66
No. 37	20.13	43.98
No. 38	27.31	59.69

No. 2: Phosphate muck from 2-ft. layer; Hoover & Mason Phosphate Co.

No. 27: Throwbacks or material reworked 5 ft. thick; Federal Chemical Co., Century plant.

No. 37: Material reworked. Average thickness, 4 ft. where sample was collected; Ruhm Phosphate Mining Co.

No. 38: Material reworked. Average thickness, 4 ft. where sample was collected; Ruhm Phosphate Mining Co.

Working "Cutters."—The phosphate rock in the "cutters," or deep places between the limestone horses, was left unmined in the early days of mining, as has been pointed out. The development of the cutters, which probably took place along original joint planes, varies greatly within the restricted Mount Pleasant field. In some places, notably south of Mount Pleasant, on the Hoover and Mason property, they are of large size. Some were observed 30 to 35 ft. wide and as much as 20 to 25 ft. deep, averaging probably 18 to 20 ft. They vary greatly in length. In these abnormally wide and deep cutters, it is not uncommon to have small limestone horses. A short distance away from the Hoover and Mason Co.'s property, some of the cutters are so narrow that the phosphate rock in them can be removed only with difficulty.

Hand methods of mining have to be employed almost exclusively to remove the phosphate rock from these cutters, owing to the peculiar method of its occurrence (Fig. 4). Hydraulic methods are also employed, as is the case in the Hickman County field where the depth of the cutters is also great. Owing to the depth of the cutters the work has to be done in benches of convenient height for the miners. The ore is picked out and shoveled from bench to bench, and finally into wagons in which it is hauled

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to the mills. Mining the deep cutters is usually carried on in fair weather, or when the roads are good. In working over virgin ground at the present

FIG. 4.—METHOD OF REMOVING PHOSPHATE ROCK FROM A DEEP "CUTTER." HOOVER AND MASON PHOSPHATE CO., MT. PLEASANT, TENN.

FIG. 5.—OLD PHOSPHATE WORKINGS SHOWING PHOSPHATIC LIMESTONE, AT THE PLANT OF THE BLUE GRASS PHOSPHATE CO., SOUTH OF MT. PLEASANT, TENN.

time, the rock in the cutters is readily and cheaply obtained by the cantilever method in use at the Hoover and Mason plant.

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Territory formerly worked over is again being worked at the Arrow mine of the Charleston, S. C., Mining and Manufacturing Co. Most of this work is in rather shallow cutters. The material is picked out and screened either on the tines of a phosphate fork or on a small movable 1-in. mesh screen. The coarse rock is dried or burned on ricks of wood; the muck is washed at the company's mill. The old cutters containing phosphate rock are located by hand prospecting with a long, sharp, steel rod.

FIG. 6.—OUTCROPPING LIMESTONE HORSES AND ASSOCIATED PHOSPHATE ROCK, TENNESSEE-ARKANSAS PHOSPHATE CO., SCOTTS MILL, MAURY COUNTY, TENN.

Washing and Drying

The washing processes whereby the mined rock is freed from clay, chert, and limestone are elaborate, and the mills in which the work is done are, for the most part, large and modern. These modern washing plants, which have done so much to make the mining of low-grade rock in this field profitable, and which, therefore, are playing such an important rôle in the conservation of phosphate rock in Tennessee, have practically all been installed during the last decade. The principles of the washing processes are the same in the different plants, but the details of manipulation differ. The phosphate rock, as mined, is brought to the washer either in wagons or by tram. Where hydraulic mining is practiced, it goes to the plant through a flume. The material mixed with water is delivered into a hopper at the top of the mill and the subsequent opera-

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tions, for the most part, are conducted by gravity. From the hopper the rock passes through a toothed revolving crusher and then into log washers. From the washers, it passes to a cylindrical or conical screen with circular perforations. The coarse, or lump rock, which fails to pass through the screen, passes on to a picking belt where limestone and chert fragments and clay balls are removed. The material then goes to the wet-storage sheds or piles to be dried later. The fine material may go through a settler or clarifier provided with riffles, or through several settling tanks in succession, in which the sand settles out. The clay and sand not caught in the process go to the waste ponds. The above descriptions briefly outline the fundamentals of the washing process as carried on at most of the plants, but, of course, as has been mentioned, details are widely divergent.

The clay and the phosphate sand which pass to the waste ponds are of great interest in the problem of conservation. When the material reaches the waste pond, the coarse sand settles out first, and, naturally, nearest the end of the waste pipe or flume. This material is the highest in calcium phosphate. It is planned to work material of this character at one of the plants near Mount Pleasant, and already at another the old tailing dumps are being worked. At the latter plant much attention has been paid to the process of separating the clay and phosphate sand. There is a washer at this particular plant which differs from any other in the field, and is most thorough in its action. The clay resulting from the action of this washer was observed in the waste pond. It had been in suspension for a long period of time, and material taken and rubbed between the fingers appeared almost of impalpable fineness. Some of the phosphate sand from this washing process is so fine in texture that it sifts through the meshes of the sacks in which it is shipped.

The analyses of material from ponds abandoned years ago, but which are either being worked or which it is planned to work, are given in Table 3. It has been suggested that the material in these waste ponds might be used in its present form on Tennessee farms, but this has been found impracticable, as it will not bear the cost of transportation. The high phosphate content in certain of the samples collected is noteworthy.

Drying is accomplished in two very different ways, which are representative of the old and new methods employed in the Tennessee brown-phosphate field. At nearly all the large plants, modern rotating cylindrical driers, similar to rotary cement kilns, are in use, but the rock is fed both at the hot and cold ends. It would seem that the latter method would be the more efficient. There is generally some special cause when the old-fashioned method of drying on wood ricks is employed, and where it is in use it usually saves extra handling or haulage. Drying generally reduces the moisture present from 20 to 21 per cent. to 1 or 2 per cent.

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TABLE 3.—*Analyses of Waste Material from Phosphate Washers in the Mount Pleasant, Tenn., Phosphate Field**

(W. C. Wheeler, Analyst)

	P_2O_5, Per Cent.	$Ca_3(PO_4)_2$, Per Cent.
No. 13	31.79	69.47
No. 20	27.20	59.44
No. 25	29.91	65.34
No. 32	20.38	44.53
No. 36	14.36	31.37
No. 42	30.63	66.92
No. 47	20.71	45.26
No. 50	28.59	62.47
No. 54	26.50	57.90

No. 13: Sample of material from waste pond; International Agricultural Corporation, Frierson plant.

No. 20: Sample of material formerly discarded but now being reworked; Federal Chemical Co., Tennessee plant.

No. 25: Sample of material from waste pond; Tenn. Ark. Mining Co..

No. 32: Sample of material from waste pond, near outlet of waste pipe. It is planned to rework this; Federal Chemical Co., Century plant.

No. 36: Sample of material from waste pond; Ruhm Phosphate Mining Co.

No. 42: Sample of material from waste pond; International Agricultural Corporation, Jackson plant.

No. 47: Sample of material from waste pond; Blue Grass Phosphate Co.

No. 50: Sample of material from waste pond; Hoover and Mason Phosphate Co.

No. 54: Sample of material from waste pond; Charleston, S. C., Mining and Manufacturing Co.

Conservation of Fines

In drying phosphate rock, much material in finely divided form has been lost by being carried out through the flue, owing to the powerful drafts employed, especially in the modern types of driers. At many of the plants, steps have been taken to save this material. This is accomplished by means of bends in the flue or by hoods or baffles. The following is an analysis of fine material caught in the chimney at the Century plant of the Federal Chemical Co. (W. C. Wheeler, Analyst):

No. 35.....P_2O_5, 30.29 per cent.; $Ca_3(PO_4)_2$, 66.19 per cent.

CHEMICAL METHODS OF CONSERVING PHOSPHATE ROCK[6]

Their Application in the West

There is associated with all large important phosphate-rock deposits, considerable rock which is not up to the present commercial requirements

* Some of this material is now being worked and some of it will soon be worked.

[6] J. A. Barr: Tennessee Phosphate Practice, *Trans.*, vol. 50, pp. 917–933 (1915).

W. H. Waggaman: *U. S. Bureau of Soils, Bulletin* No. 76, pp. 15 and 16 (1911); also *U. S. Department of Agriculture, Bulletin* No. 144, pp. 25 and 26 (1914).

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in content of calcium phosphate. There is also being produced, in connection with the preparation of commercial phosphate rock for market, a great deal of low-grade material. To bring these classes of material up to commercial grade, that is, to a grade containing 70 per cent. or more calcium phosphate, various chemical methods have been used. The time will undoubtedly come when these chemical methods will find much more extended application than at present, and when this time arrives it will result in conserving a great deal of phosphate rock now consigned to the waste ponds and dumps. Such methods are of more than ordinary interest in connection with the Western field, owing to the long distance that phosphate rock now has to be transported before reaching a market. The immense quantities of sulphuric acid potentially available in the immediate vicinity of the Western phosphate field, and which should become available in increasing quantity as time goes on, is another important element in the situation. Indeed, the chemical method of concentrating phosphate, and thus enabling it to be transported long distances, may well be worked out in connection with the high-grade rock that this field is able to produce, and it may also be the means of conserving the enormous amount of low-grade phosphate rock in the Florida, Tennessee, South Carolina, and other Eastern fields.

Chemistry of Process

Phosphate rock is marketed now as such, and in the form of acid phosphate, including in the latter term ordinary super and double-acid phosphate, the latter containing two to three times as much soluble phosphoric acid as ordinary superphosphate.

Before the discovery of the extensive high-grade deposits of phosphate rock in this country and abroad, the manufacture of the concentrated grades of soluble phosphate was in fairly common practice. The large supplies of high-grade phosphate rock have rendered this unnecessary, though in France, Germany, and possibly other European countries, and in South Carolina, this practice is reported to be still in use.

The basic reaction involved in the preparation of soluble acid phosphate takes place when ordinary rock phosphate $Ca_3(PO_4)_2$ is treated with sulphuric acid. In simple form, the reaction that takes place may be represented thus:

(1) $Ca_3(PO_4)_2 + 3H_2SO_4 = 2H_3PO_4 + 3CaSO_4$.
(2) $4H_3PO_4 + Ca_3(PO_4)_2 = 3CaH_4(PO_4)_2$

Reduced to one equation, this is as follows:

$$Ca_3(PO_4)_2 + 2H_2SO_4 = CaH_4(PO_4)_2 + 2CaSO_4.$$

In the presence of water, which has been omitted from the above equations in order to simplify them, the calcium sulphate would be changed

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into gypsum by abstracting water from the mass. The last reaction is the one desired by the manufacturers.

To utilize low-grade rock and tailings, and to make concentrated phosphatic fertilizers, the phosphoric acid produced by the first reaction is evaporated in pans until it contains about 45 per cent. phosphoric anhydride. It is then treated with a fresh supply of phosphate rock, when the following reaction ensues:

$$Ca_3(PO_4)_2 + 4H_3PO_4 = 3CaH_4(PO_4)_2.$$

It will be observed, therefore, that ordinary superphosphate is largely a mixture of soluble calcium phosphate and gypsum, while the double-acid phosphate contains little or no calcium sulphate, or dehydrater, and thus has to be artificially dried. Either the phosphoric acid itself, the double phosphate, or such compounds as ammonium phosphate, might be shipped from our Western field, since they are highly concentrated products.

SUBSTITUTES FOR PHOSPHATE ROCK

The use of substitutes for ordinary phosphate rock has been in the past of great importance, but, since the discovery of large deposits of high-grade phosphate rock, the price of phosphatic fertilizers has greatly decreased, as a result of which the more highly priced guanos and similar materials have been driven out of the market. The use of substitutes, however, still continues. There is still much low- and medium-grade material which could, if necessary, actually take the place of phosphate rock as a source of phosphorus and which may possibly be used at some future time as commercial conditions change.

In the following pages, the different substitutes for phosphate rock that have suggested themselves are named and briefly described. They may be classified in two groups: (1) The natural and (2) the artificial substitutes for phosphate rock. Under the natural substitutes come (*a*) phosphatic limestone; (*b*) other phosphate-bearing minerals, such as apatite, nelsonite, wavellite, and others; (*c*) guano; (*d*) marl; (*e*) excrement, both human and animal; (*f*) bones. Within the class of artificial substitutes may be included (*a*) the basic slags and (*b*) manufactured compounds, like ammonium phosphate; the double-acid phosphate, phosphoric acid, and ordinary superphosphate from the low-grade phosphate rock discarded at phosphate mines.

The Natural Substitutes for Phosphate Rock

Phosphatic Limestone

Directly below the phosphate-rock horizon in the Mount Pleasant and other parts of the Tennessee field, occurs the phosphatic limestone from which the brown rock itself has been derived. There must be an

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enormous tonnage of this phosphatic limestone scattered throughout the phosphate-rock areas of middle Tennessee. A long period of time must elapse before any attention will be given to this comparatively low-grade material as a source of phosphate, but it would be hazardous to say that this will never be done. Of course, in the mined-over area, much of the richer limestone has been covered so deeply that it will be difficult and expensive to get at. Analyses of this limestone, some of which was in a leached and some in a partially leached condition, occurring in horses between cutters, show calcium phosphate ranging from 4 to more than 42 per cent. The carbonate and the phosphate of lime mixture in this material has considerable value as fertilizer when applied directly to the land in finely pulverized form, and, although it is difficult to predict how and when this material will be utilized, it seems fairly certain that it will prove of value at some future time.

Under present conditions of mining, the phosphate is dug from around the limestone horses and boulders, and the pits are then either abandoned or filled with material from the overburden. The time has not arrived for the utilization of such material, but, when it does, it will be removed, broken up, crushed, and spread on the land, either with or without treatment with acid, or, as suggested by Waggaman,[7] the phosphatic limestone may be burned in a kiln and then slaked with steam or hot water and the rock thus disintegrated. In experiments made by heating the phosphatic limestones, after their phosphate content had been determined, it was found that the percentage of the latter was increased in quantity, the increases ranging from 1.40 to 8.62 per cent.[8]

Apatite, Nelsonite, and Other Minerals Containing Phosphate

The mineral apatite is among the most definite in composition, if not the most definite, of the crystalline phosphate-bearing minerals. It is widely distributed and occurs in rocks of various kinds, but most commonly in those of the metamorphic and crystalline types, such as crystalline limestone, dolomite, gneiss, the mica schists, etc. The two common varieties are fluor-apatite, $3Ca_3(PO_4)_2.CaF_2$, and chlor-apatite, $3Ca_3(PO_4)_2.CaCl_2$, with intermediate compounds containing both chlorine and fluorine. The normal varieties contain in the case of fluorine 42.3 per cent. phosphoric acid (P_2O_5), and the chlor-apatite 41 per cent. of phosphoric acid (P_2O_5). The fluor-apatite is much the more common variety and here belongs the apatite found in St. Lawrence County, N. Y., Canada,[9] Spain, and the Alps. The Norway apatite is of the chlor-

[7] *U. S. Bureau of Soils, Bulletin* No. 81, p. 13 (1912).

[8] *Loc. cit.*

[9] R. W. Ells: *Geological Survey of Canada, Mineral Resources of Canada, Bulletin on Apatite* (1904).

For a complete list of minerals containing at least 1 per cent. of phosphoric acid, see W. B. Phillips: *Trans.*, vol. 21, pp. 188–196 (1892–93).

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ine-bearing type. The apatite associated with the magnetite at Mineville, northern New York, on the property of Witherbee, Sherman & Co., is worthy of special mention. Attempts have been made, according to report, to separate the apatite on a commercial scale, but they have not proved successful.

Apatite in Virginia.—Rock of igneous origin, rich in titanium and phosphorus, occurs in the eastern foothills of the Blue Ridge in Virginia, near Roseland, about 7 miles northwest of Arrington on the Southern Railroad, and 24 miles northeast of Lynchburg. The titanium occurs in the form of ilmenite, and the phosphorus in the form of apatite, these being the dominant minerals. The rock occurs in dikelike forms of varying size and irregular shape, and to it the name nelsonite has been applied.[10]

Many varieties of rock have been included under this name, for example, ilmenite nelsonite, magnetite nelsonite, biotite nelsonite, hornblende nelsonite, and rutile nelsonite, from the dominant mineral that may be present, and gabbro nelsonite, from the rock-type gabbro, which some of the facies of nelsonite resemble. Ilmenite nelsonite, to which the name was first applied, is the normal and most abundantly occurring variety. The apatite present is the fluor-apatite, chlorine being present only in traces. The content of the nelsonite in apatite ranges from 0.3 to 30 per cent.

Experiments having as their object the commercial utilization of nelsonite have been carried on at the Bureau of Soils. Both the minerals ilmenite and apatite contain elements which seriously affect each other so far as commercial applicability is concerned. The problem, then, first, is to separate them. Two mechanical methods have been tried, depending (1) on differences in specific gravity, (2) on the magnetic properties of the ilmenite. Neither of these methods was found entirely satisfactory. The chemical method was then resorted to. In the experiments performed by W. H. Fry, of the Bureau of Soils, it has been found that ilmenite may be almost completely freed from apatite by means of sulphuric acid with a minimum of waste and without involving great expense. Moreover, all the products obtained can be utilized commercially.

Preliminary experiments showed that ilmenite is entirely unattacked by dilute sulphuric acid, while apatite is acted on quite energetically by this reagent. Subsequently, it was shown that the apatite remaining in ilmenite after mechanical separation can nearly all be extracted by means of dilute sulphuric acid without appreciably affecting the ilmenite. The

[10] T. L. Watson: *Mineral Resources of Virginia*, p. 300 (1907); Geology of the Titanium and Apatite Deposits of Virginia, *Virginia Geological Survey, Bulletin 3A*, pp. 100 *et seq.* (1913); T. L. Watson: and Stephen Taber: Virginia Rutile Deposits, *U. S. Geological Survey, Bulletin* 430, p. 206 (1910).

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details connected with the experiments are outlined in an article by Waggaman.[11]

Thus the possibility is shown of obtaining phosphate fertilizer material from this occurrence.

The great objection to apatite as a source of phosphate is the expense of mining and preparing the rock for treatment with acid. In the case of the apatite containing fluorine, hydrofluoric acid gas and possibly other poisonous gases are given off when the mineral is acted upon by sulphuric acid. Thus, the treatment of this mineral with acid may be attended with danger, unless care is exercised in the manipulation, or unless there is enough of silica or silicates present to react with the liberated hydrofluoric acid. The cheap and accessible sources of phosphate have caused a practical stoppage of the mining of apatite for fertilizer purposes, though at one time a considerable amount of the mineral was so used.[12]

Guano

Guano consists chiefly of the excrement of birds and bats, and has been found in considerable quantity in some places. It has been extensively used in the manufacture of acid phosphate. There are two types: (1) The unleached deposits found in caves and sheltered places, which contain phosphoric acid in readily available form, and also nitrogen, a fertilizer of great importance and high price. (2) The leached deposits, which contain no nitrogen and in which the phosphoric acid content, though high, is generally insoluble. Accessible and valuable deposits of guano are now rather scarce, and only those having the best transportation facilities are being worked.

Greensand (Marls)

Phosphate is found to a small extent in certain greensands of the eastern and southeastern States, notably in New Jersey, Kentucky, Tennessee, South Carolina, Florida, Alabama, and without doubt in many other coastal-plain States of the East.

New Jersey.—About 30 years ago, the greensands of certain portions of New Jersey were in great demand. On the first geologic map of that State, the location of the beds containing them was shown, and in some of the earlier reports the deposits were described and numerous analyses given, as well as instructions for their use. In recent years, however, greensand has been supplanted to a large extent by the more highly concentrated artificial fertilizers and is no longer dug extensively.

[11] W. H. Waggaman: A Possible Commercial Utilization of Nelsonite, *Journal of Industrial and Engineering Chemistry*, vol. 5, No. 9, pp. 730–732 (1913).

[12] Ells: *Loc. cit.*

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The analyses in Tables 4 and 5 show the composition of the different grades of greensand as dug and applied to the soil. The glauconite in them is of nearly uniform composition, but mixed with it are carbonate, sulphate, and phosphate of lime, quartz sand, sulphide and phosphate of iron, shells, etc. The differences in the kind and quantity of these substances cause wide differences in the appearance of the greensand containing them, as well as in its composition and properties.

Table 4 gives the phosphoric-acid content, in percentages, of typical specimens of New Jersey greensand:

TABLE 4.—*Phosphoric Acid in Typical Greensand (Marl) of New Jersey*

	1	2	3	4	5	6	7	8	9	10
Phosphoric acid.....	1.14	1.33	1.02	2.24	2.69	2.56	3.58	3.87	2.58	2.30

New Jersey greensand[13] has been of incalculable value to the region in which it is found. It has raised this region from the lowest stage of agricultural exhaustion to a high state of improvement. Found in places where no capital and but little labor were needed to get it, the poorest people have been able to avail themselves of its benefits. Lands which in the old style of cultivation had to lie fallow, by the use of marl produce heavy crops of clover and grow rich while resting. Land which had been worn out, and left in commons, is now, by the use of this fertilizer, yielding large crops of the finest quality. Everywhere in the marl district, may be seen farms which in former years would not support a family but which are now making their owners rich through their productiveness. Bare sands, by the application of marl, are made to grow clover and then crops of corn, potatoes, and wheat. "Pine barrens," by the use of marl, are made into fruitful land. The price of land in the greensand-marl belt of New Jersey was considerably below that in the northern part of the State 40 years ago; now the price is higher.

Kentucky.—The greensand (marls) of the Leitchfield, Kentucky, region have been described by N. S. Shaler[14] and have been analyzed by Robert Peter.[15] A sample from near Leitchfield, Grayson County, was sent to the United States Geological Survey by M. H. Crump, of Bowling Green, Ky. It was analyzed in the Survey laboratory by Chase Palmer and found to contain 3.73 per cent. potash (K_2O) and 0.13 per cent. phosphoric acid (P_2O_5).

Greensands containing approximately these quantities of potash and phosphoric acid are found in large quantities over a considerable area in west central Kentucky, and in the reports cited above their use was sug-

[13] *Annual Report of the State Geologist of New Jersey*, 1886, p. 154.

[14] *Kentucky Geological, Survey Report, new ser.*, vol. 3, pp. 46–47 (1877).

[15] *Kentucky Geological Survey, Chemical Analyses*, vol. *A*, pp. 250–254 (1884).

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gested to rejuvenate the lands of the State, worn out as a result of the excessive cultivation of tobacco and other crops.

Tennessee.—The greensands of Tennessee were mentioned by G. Troost as early as 1835. They are found in Hardin, McNairy, and Henderson Counties. The analyses by Troost shown in Table 5 are of the greensands of McNairy County:

TABLE 5.—*Analyses of Greensand, McNairy County, Tenn.**

	1	2	3
Silica (SiO_2)	48.00	45.30	51.70
Alumina (Al_2O_3)	7.00	6.20	6.50
Ferrous oxide (FeO)	20.70	18.00	21.20
Potassa (K_2O)	10.10	10.40	11.30
Carbonate of line ($CaCO_3$)	5.70	10.80	2.00
Water (H_2O)	8.00	8.50	7.30
Loss	0.50	0.80	0.00
Total	100.00	100.00	100.00

Phosphoric acid, which is doubtless present in the marls, does not appear to have been separated.

Alabama.—Materials of several kinds, containing phosphoric acid, have been found in Alabama, in both the Cretaceous and Tertiary formations. In several places in the lower part of Marengo County, notably near Dayton and Nixonville, there occur tolerably compact beds of shell-casts, containing from 20 to 25 per cent. phosphoric acid. According to E. A. Smith,[16] these beds are the most promising sources of phosphate in the State. Such high-grade phosphate beds should have future value as fertilizers.

Excrement

A large part of the phosphorus sent from the country in the form of cattle and grains for consumption in the cities, finds its way ultimately, via the modern sewage system, to the ocean. There it is lost so far as its fertilizer value to the land is concerned. Whitson[17] estimates that the loss in the cities due to human excreta alone is equivalent to 2 or 3 lb. of phosphoric oxide per acre for the entire cropped region of the United States. Supposing this loss to be 2 lb., $\frac{1}{1,000}$ ton, this amounts, for

* J. M. Safford: *Tennessee Geological Survey*, p. 515. Nashville, 1869.

[16] E. A. Smith: The Phosphates and Marls of Alabama, *Trans.*, vol. 25, p. 811 (1895).

[17] A. R. Whitson and C. W. Stoddart: The Conservation of Phosphates on Wisconsin Farms, *University of Wisconsin Agricultural Experiment Station Bulletin*, No. 174, pp. 1–8 (April, 1909).

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400,000,000 acres of cropped land, to 400,000 tons of phosphoric oxide—equivalent to 1,200,000 tons of phosphate rock.[18] Surely here is a problem for the municipal and chemical engineer of the future.

Bones

Before phosphate rock was found in quantity in the United States and in other parts of the world, bones were the main source of phosphoric acid for fertilizing purposes. There is still a domestic production of this form of phosphate fertilizer. The bones were steamed, charred or burned, and applied to the land after such treatment, or they were made into superphosphate in the way ordinary bone phosphate is now converted, by treatment with sulphuric acid. Ground bone, or bone meal, is a valuable fertilizer without any treatment whatever, due to its nitrogen and phosphorus. The phosphoric acid contained in bones has considerable value as a source of phosphorus for chemical purposes other than fertilizer.

Artificial Substitutes for Phosphate Rock

Basic Slag[19]

Basic slag is not soluble to any extent in water, but it has been shown that the phosphate contained in it is readily assimilated by growing crops and splendid results have followed from its use as a fertilizer.

It is produced in the manufacture of steel by the basic Bessemer and basic open-hearth processes. In these the phosphorus is separated from the iron by the addition of lime or limestone, which, having a strong chemical affinity for the phosphorus, forms with it basic phosphoric compounds which enter the slag. In slags produced by these processes, the phosphoric acid may run from 10 to more than 25 per cent.

The iron ores used in this country, with the exception of some mined in Alabama, are low in phosphorus and consequently the basic Bessemer process is not applicable to them. The slags produced from such low-phosphorus ores are consequently low in this fertilizing element, but abroad, and especially in Germany, where the iron ores are high in phosphorus, the basic Bessemer process is widely used and as a result the slag, rich in phosphate, is widely applied as a fertilizer. There are indications that the importance of these high-phosphatic slags is coming to be realized in this country.

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[18] C. R. Van Hise: *Conservation of Natural Resources in the United States*, p. 325. The MacMillan Co., New York (1910).

[19] E. C. Eckel: Utilization of Iron and Steel Slags, *U. S. Geological Survey, Bulletin No.* 213, pp. 225–227 (1903).

W. H. Waggaman: Utilization of Acid and Basic Slags in the Manufacture of Fertilizers, *U. S. Bureau of Soils, Bulletin No.* 95 (1913).

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In rock phosphate the phosphoric acid is combined with lime as tri-calcium phosphate with the symbol $Ca_3(PO_4)_2$. In slags, however, the combination is that of tetra-calcium phosphate ($4CaO.P_2O_5$ or $Ca_4P_2O_9$), according to certain investigators. Other experimenters have found that different compounds have formed with variations in the composition of the molten bath and that toward the end of the operation, when much silica was present, a compound having the formula $P_2O_5.SiO_2.5CaO$ was formed. It has also been shown that in slags having the same phosphate content, those containing silica were the most soluble and hence the most valuable for fertilizer purposes.

PHOSPHATE ROCK RESERVES

FLORIDA[20]

Hard Rock

Several maps of the Florida phosphate fields have been published, but they are of a general nature and attempt to show only the approximate location of boundaries. Even this is difficult to do with any degree of accuracy without careful prospecting. In estimating the available hard rock, some idea of the total area in the State within which such deposits may occur is essential. The estimates made agree fairly closely, and indicate that the workable beds of hard phosphate rock in Florida occur throughout an area of several hundred square miles. The area actually underlain by workable deposits of hard rock is, however, but a small fraction of that within which the deposits have been mapped. There are, moreover, no existing data from which one may calculate the area underlain by workable deposits with a degree of accuracy that would have any practical value whatever.

Sections of square miles could be taken within which the deposits have been most completely mined out and tonnage estimates made from them, but figures thus obtained could not be used in other areas as a standard, since over many square miles there are no deposits at all. The deposits in the sections which have been most completely mined out, moreover, are usually the most accessible. With due allowances, it is conservatively estimated that there is as much hard rock available in Florida as has already been removed, that is, approximately 10,000,000 tons, and with an annual output of 500,000 tons the hard-rock phosphate deposits may be expected to last at least 20 years longer.

Land Pebble

The land-pebble beds are more regular in their occurrence than the hard-rock deposits, but close estimates cannot be made except by actual

[20] The writer is greatly indebted to Dr. E. H. Sellards, State Geologist of Florida, for valuable suggestions in preparing this note on the Florida phosphate-rock reserves.

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prospecting; and this will be done only gradually by those who are interested in or are engaged in mining. The land-pebble phosphate belt is approximately 30 miles long by 5 to 10 miles wide. On the basis of a conservative estimate of acreage and of tonnage per acre, the writer has calculated a total of 190,000,000 tons of land pebble.

The output of land pebble per year in Florida is, in round numbers, 2,000,000 tons. The estimate of available land pebble, which is considered extremely conservative, leads to the conclusion that this type of phosphate rock in Florida will last several generations, and for present purposes it may be considered practically inexhaustible. The refinements in methods of mining land pebble are gradually reducing the quantity of small pebbles that go to the waste dump, and this factor will tend to prolong the life of these deposits beyond that calculated from the figures given above.

In making up the estimates for Florida, river pebble has not been included, owing to the difficulties connected with estimating its quantity. This factor also adds to the conservatism of the figures given for this State.

Waste Material

The phosphoric acid in the Florida deposits in the form of soft phosphate, so-called, together with large quantities of aluminum and iron phosphate, go to the dumps in the preparation of the hard and pebble rock for market. The loss calculated in terms of phosphate of lime is considerable and it may possibly equal the actual quantity saved and marketed. It has been calculated by W. H. Waggaman[21] that the marketed material is probably not more than 15 per cent. of the total material mined, and that in the discarded material is an average of at least 10 per cent. phosphoric acid. The total quantity of Florida phosphate rock marketed up to and including 1914 is approximately 27,500,000 long tons. On these bases, the low-grade material in the waste heaps is approximately equivalent to 27,000,000 to 30,000,000 tons of high-grade material.

TENNESSEE

Introductory Note

There are three varieties of phosphate rock in Tennessee, the white, the blue, and the brown. The very irregular character of the white phosphate-rock deposits, and the fact that they have been but meagerly prospected, make it impossible to estimate the available supplies even approximately. The omission of estimates of white rock may be considered to contribute to the conservatism of the total figure for the State.

[21] W. H. Waggaman: *U. S. Bureau of Soils, Bulletin No.* 76, p. 14 (1911).

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Blue Rock

The blue bedded phosphate-rock deposits are found chiefly in Perry, Hickman, Lewis, and Maury Counties. In estimating them, only rock 24 in. and over in thickness will be considered. The area underlain by rock of this thickness is large, but the area in which the rock is thinner is much greater. This estimate includes the rock under heavy cover, but this is not a bar to its availability, since blue rock is worked wholly underground.

As stated in another part of this paper, blue phosphate rock, even in those localities where thickest, is extremely variable and may locally be absent altogether. As an offset to this factor, however, a very large area has been omitted in the calculations in which the rock has been considered to be less than 24 in. in thickness, but which future prospecting may show to be thicker. It may reasonably be expected that outlying areas in which the rock is locally workable will be found in the future. It must also be considered that rock only a foot thick, but of high-grade, may be considered of future workability. It is also possible that all of the blue rock estimated may not be up to present commercial requirements. As time goes on, however, the standard in this respect will tend to fall, and ore may then be treated chemically to raise the grade for shipment to distant points.

Taking all the above factors into consideration, it is estimated that there are approximately 84,000,000 tons of available blue rock phosphate in Tennessee.

Brown Rock

The brown phosphate rock included in the estimate given below occurs chiefly at the Bigby horizon. It is found chiefly in Maury, Giles, Hickman, Lewis, and Sumner Counties, Tennessee. The Mount Pleasant, Maury County, district contains the most extensive deposits, and the major operations on brown rock phosphate in the State are located there. The quantity of brown phosphate rock in the Mount Pleasant district, which may be taken as a type, ranges from 600 to 1,000 tons per acre-foot; and 850 tons is considered a fair average. A considerable portion of the areas where are located important deposits of brown rock has been worked over in Tennessee, especially in the Mount Pleasant field. This fact, and the additional fact that the entire acreage can not be underlain by phosphate rock, would tend to reduce the estimate; but, on the other hand, it is very reasonable to suppose that as detailed prospecting takes place new and important areas, though small, will be found. Two such areas have recently been brought to the writer's attention. Taking these factors into consideration, it is estimated that an available tonnage of approximately 4,000,000 tons of brown rock

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remains in middle Tennessee, or approximately about as much as has already been marketed.[22]

In addition to the virgin rock remaining unmined in Tennessee, a great deal of phosphate rock will be recovered from waste ponds and from the dumps from old workings. Such work is in progress, as described earlier in this paper. The recovery of this material will tend to greatly prolong the life of the brown phosphate-rock field beyond the period indicated by a theoretical calculation based on the estimate of reserves made above and the yearly marketed output of brown rock in Tennessee.

South Carolina

Several estimates have been made of the quantity of phosphate rock available in South Carolina for future use. In the very early days of the industry, the data for such estimates were lacking, but increased knowledge gained from working the deposits has lessened this difficulty to some extent. Shepard, in 1880, estimated the total available supply to be less than 5,000,000 tons, but from 1881 to 1914, inclusive, the quantity of rock removed was 11,600,000 tons out of a total of 13,000,000 tons for the State since the industry began in 1867.

P. E. Chazal[23] in 1904 estimated the quantity of rock remaining in the land deposits at between 9,000,000 and 11,000,000 tons. Since 1904, there have been removed from the South Carolina deposits more than 2,000,000 tons, leaving still available between 7,000,000 and 9,000,000 tons of phosphate rock. These figures do not include the river rock. The higher figure, namely, 9,000,000 tons, may therefore be taken as a conservative estimate of the available supply in South Carolina.

The annual production of South Carolina phosphate rock is approximately 100,000 tons. At the present rate of production, the phosphate rock supply of South Carolina would last approximately 90 years. In view of the fact that the production of this State has been steadily decreasing for many years, and the possibility that other deposits may be found, it seems safe to conclude that the South Carolina deposits can produce rock of somewhat low grade for many years to come and should become an important source of supply when the chemical concentrating methods come into general practice.

It is interesting in this connection to note that W. H. Waggaman[24]

[22] James A. Barr: (*Bulletin* No. 110, p. 244, Feb., 1916) states: "In the event of low grades of phosphate becoming commercial products, 20,000,000 tons would become available within a 50-mile radius of Mount Pleasant alone, and perhaps 100,000,000 tons of rock, especially if phosphatic limestone is taken into account."

[23] P. E. Chazal: *A Sketch of the South Carolina Phosphate Industry*, pp. 17 and 18 (1904).

[24] W. H. Waggaman: *Journal of Industrial and Engineering Chemistry*, vol. 6, No. 6, p. 464 (June, 1914).

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has estimated a somewhat higher available supply of phosphate rock of high grade in South Carolina, namely, 10,000,000 tons. F. B. Van Horn's[25] estimate made in 1909 places the available South Carolina phosphate rock at 3,000,000 tons, but that writer qualifies the estimate by observing that careful and deep prospecting may increase these figures.

KENTUCKY

The rock occurring in this field is of the brown-rock type, similar to the brown phosphate-rock deposits occurring in middle Tennessee. The areas underlain by high-grade deposits are scattered, but the most important areas occur to the south and northwest of Midway, Woodford County. The writer has explored this region with the drill, though not closely enough to give close figures. A very conservative estimate of the phosphate rock available in this region is 1,000,000 tons.

ARKANSAS

Though there is only a small production of phosphate rock in Arkansas at the present time, it must not be inferred that the deposits in this State will not prove of future value. In many of the analyses published by Branner and Newsom,[26] the content of the iron oxide and alumina runs high, and although this factor tends to unfit the Arkansas rock for the manufacture of superphosphates under present conditions, the time without doubt will come when these deposits will be extensively exploited for direct application to the soil. Chemical treatment should also render a very large part of the Arkansas phosphate available for future use. The tonnages per acre calculated for small tracts in which conditions are well known, show a large quantity of phosphate rock.[27] Calculations of tonnages indicate as much as 11,600 tons in one locality where the thickness of the rock was approximately 4½ ft. In another place, the acre tonnage was 23,200 where the average thickness was considered to be 8 ft. The latter figure is reckoned on the basis of 2,900 tons per acre, which, in turn, is based on a weight of 150 lb. per cubic foot of rock. Waggaman[28] estimates 20,000,000 tons of high-grade rock in Arkansas.

THE WESTERN STATES

The available tonnage estimates for the Western field, which are only partial, are chiefly for Idaho, but include figures for small parts of Utah,

[25] F. B. Van Horn: Phosphate Deposits of the United States, *U. S. Geological Survey, Bulletin No.* 394, p. 164 (1909).

[26] *Arkansas Agricultural Experiment Station Bulletin, No.* 74, pp. 116–119 (1902).

[27] *Loc. cit.*, pp. 71–80.

[28] W. H. Waggaman: *Journal of Industrial and Engineering Chemistry*, vol. 6, No. 6, p. 464 (June, 1914).

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Wyoming, and Montana. The estimates prepared, which are considered conservative, are all for high-grade rock, 65 per cent. or more tricalcium phosphate, and refer chiefly to the main bed which ordinarily lies near the base of the phosphate shales and in the Idaho field is usually 5 or 6 ft. thick. Thus they do not include the thinner beds of high-grade rock nor the great body of low-grade material. The main bed itself is included only for those parts of the field in which, according to the present practice of the Geological Survey, the rock lies at depths considered workable. The estimates are based on the assumption that the phosphate rock at depth, and remote from the outcrop, maintains the generally uniform qualities displayed at the surface in so many parts of the field. Some tendency toward enrichment by weathering has indeed been noted, but observations thus far obtainable suggest no marked decrease in richness within the body of the material.

Under the above conditions the areas examined in the field work of 1909 to 1913, inclusive, in Utah, Wyoming, and Idaho, are estimated to contain at least 5,290,296,900 long tons of high-grade phosphate rock. In addition, the Elliston field in Montana is estimated to contain 86,000,000 short tons,[29] or approximately 76,785,700 long tons, a total available estimate of 5,367,082,600 long tons. Since 1913, additional areas in some of these regions have been examined in detail and the presence of still more of the high-grade rock has been determined. There yet remains a considerable area of withdrawn land that has not been examined, in which it is probable that high-grade phosphate will be found. When the results of these examinations and the results of more detailed work in regions now known only by reconnaissance are added to the figures given, it is probable that the estimates of high-grade rock in the Western field will be considerably increased.

Total Tonnage Available

In Table 6 is given the estimated tonnage of phosphate rock available in the United States at the present time.

Table 6.—*Phosphate Rock Available in the United States*

Eastern States	Long Tons	Western States	Long Tons
Florida	227,000,000	Montana, Idaho, Utah, and Wyoming	5,367,082,600
Tennessee	88,000,000		
South Carolina	9,000,000		
Kentucky	1,000,000		
Arkansas	20,000,000		
	345,000,000	Total	5,712,082,600

[29] R. W. Stone and C. A. Bonine: The Elliston Phosphate Field, Montana: *U. S. Geological Survey, Bulletin No.* 580, pp. 382–383 (1915).

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The United States is now producing for domestic use and export about 3,000,000 tons annually.[30] More than 99 per cent. of this comes from the Eastern States, and in 1914 nearly 80 per cent. came from Florida. On this basis, Eastern phosphates should last fully 100 years, taking into account material of good grade.

FOREIGN PHOSPHATE DEPOSITS

Important deposits of phosphate rock are located outside the United States. Perhaps the best known of these foreign deposits are those in the South Sea Islands, Christmas Island in the Indian Ocean, the African deposits, and those on Curaçao in the Dutch West Indies. In Africa, the principal deposits are located in Egypt, Tunis, and Algeria. In the South Pacific, Angaur of the Pellew group of islands, Nawoda (Pleasant), Panapa (Ocean), and Makatea (Aurora) islands, all contain important deposits. A deposit has recently been reported in northern Chile.

African Deposits

Egypt[31]

The development of the extensive deposits of phosphate near the Red Sea has, during the past 2 years, assumed important proportions. The mines are worked by a British concern and are connected by rail with the Red Sea, where the rock is loaded on steamers for export. The rock contains 65 per cent. or more of tricalcic phosphate.

A company managed by Italians and founded in 1912, has obtained extensive concessions about 12 miles inland from Kosseir and also at Sebaia, on the eastern bank of the Nile, between Keneh and Assouan. The former concession is being connected with the port of Kosseir by a light railway, which should shortly be completed. The rock from the latter mines will be transported down the Nile by a ropeway and thence to Alexandria by boat for shipment.

The total output of phosphate in Egypt for the years 1908 and 1912 was as follows: 1908, 700 tons; 1909, 1,000 tons; 1910, 2,397 tons; 1911, 11,925 tons; and 1912, 69,985 tons. According to the *Financial Adviser's* report, the output during 1913 exceeded that for 1912 by about 33,000 tons.

Although other beds of phosphate are found in various districts in Egypt on both sides of the Nile Valley, the Red Sea area is responsible for almost the whole output. The rapid development of the business of the Egyptian Phosphate Co., the British concern, and the impending com-

[30] Normal conditions are referred to.

[31] U. S. Consul-General Olney Arnold: *U. S. Daily Consular and Trade Reports*, p. 991 (Aug. 20, 1914).

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mencement of active operations by the other company, will lead to a considerable increase in production.

Practically all the raw phosphate produced is shipped from Egypt, principally to Japan. That country in 1912 took about 49,000 tons out of a total export of 52,000 tons, and in 1913, 59,000 tons out of a total of 64,000. According to the customs' returns, the average value of the raw phosphate shipped during 1913 at Port Safaga was about £1 ($4.87) per ton.

A foreign correspondent of the Survey gives the following interesting information with reference to the extent and continuity of the North African deposits:

They correspond geologically to the deposits of phosphate rock in Tunis and Algeria, and there can be little doubt that the phosphate-bearing formation extends from Algeria to the Red Sea, a distance of about 2,000 miles. The intervening territory, comprised in Tripoli and the Lybian Desert, has not been explored. There is also evidence that the same phosphate-bearing formation extends beyond the Red Sea and through the Arabian Desert into Persia, a farther distance of at least 1,000 miles, and there is a possibility that it extends much farther.

Algeria and Tunis[32]

The deposits of phosphate rock in Algeria are continuations of the deposits in Tunis. The two important mining districts in Algeria are located near the towns of Setif and Tebessa, in the eastern part of the State. They have been developed during the last 15 years, the production having increased from 1,057 short tons in 1899 to 550,000 in 1912. The percentage of lime phosphate in the rock exported from the Setif district ranges from 58 to 63 per cent., and that in the rock from the Tebessa district, from 58 to 68 per cent.

Tebessa District.—The deposits of Kouif, the most important exploited in Algeria, are located near Tebessa, close to the Tunisian frontier. Of the five beds, three are workable, the thickness ranging from 3 to 9 ft., 3 to 4½ ft., and 1⅘ to 3 ft., in the different beds. The average percentage of phosphate in the rock varies, being, respectively, 58.64 per cent., 68.50 cent., and 48 per cent. in the three beds. Where the overburden does not exceed 24 ft. in depth, open-cut mining is practiced; where the overburden is deeper, the rock is mined by tunneling. Because of the basin, or saucer shape of the deposits, the tunnels are inclined.

Important deposits of phosphate rock have been found at Dy Nord and Djebel-Onck, about 62 miles south of Tebessa, which are believed to contain 300,000,000 to 400,000,000 tons of rock. These deposits have not been thoroughly explored, and estimates are therefore not entirely dependable.

[32] *U. S. Daily Consular and Trade Reports*, pp. 1240–1242 (Aug. 30, 1913).

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Setif District.—In 1906, La Compagnie des Phosphates de Paris leased for 20 years the deposits in the commune of Bordj-Rhir. Its shipments are made by the way of a 12-mile cable to El Anasser, on the railway from Setif to Algeria; thence they go to the part of Bougie, from which the exports amounted to 64,986 short tons in 1910, 64,824 tons in 1911, and 62,702 tons in 1912.

La Compagnie Algérienne des Phosphates is exploiting two deposits in the commune of Tocqueville. The beds are from 1 to 6 ft. thick. The phosphate is transported by a 90-mile narrow-gage branch line to the railway station of Texter-Tocqueville. In 1900, 17,807 short tons, and in 1912, 230,864 short tons of rock were mined. At present 350 workmen are employed.

Another French company, with a capital of $1,000,000, is exploiting the Mzaita mine in the communes of Maadia. A broad-gage railroad has been built from the mine to the station of Ain-Tassers, and an electric plant of 150 hp. has been installed for the stamp mill. Only 7,560 tons of rock were extracted in 1912, but it is expected that eventually 300,000 tons of rock will be mined annually. The estimated supply of rock is 16,500,000 tons.

Southern Pacific Ocean (South Sea) Islands

The South Sea islands contain the richest, if not the most extensive, phosphate deposits in the world. Japan, Australia, Hawaii, and our own Pacific coast under normal conditions, take very nearly half the production of the British, French, and German South Sea companies engaged in mining the phosphate rock. The deposits in the South Seas are notably high in calcium phosphate, and contain from 85 to 90 per cent. of this ingredient. They are also notable in their very low content of iron oxide and alumina, which ranges usually below 1 per cent.

Angaur (Pellew Group)

It is understood that the Japanese Government is now in control of the Pellew group of islands in the South Pacific, the islands having passed from German to Japanese control since the outbreak of hostilities in Europe. The principal deposits of phosphate rock are located on Angaur Island, and these are reported to be now worked to supply the Japanese fertilizer manufacturers. The deposits on the island are reported to contain in round numbers from 2,000,000 to 3,000,000 tons of high-grade phosphate rock.

Ocean Island

Ocean Island is estimated to contain not less than 50,000,000 tons of the highest grade of phosphate rock. Some shipments from this island

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are reported to run from 87 to 89 per cent. calcium phosphate and less than 1 per cent. of iron oxide and alumina.

Makatea (near Tahiti)

On Makatea, near Tahiti, it is estimated that about 10,000,000 tons of phosphate rock of high grade are available.

Other Foreign Deposits

Christmas Island

Christmas Island, near the west end of Java in the Indian Ocean, produces as high-grade phosphate rock as is found in the South Sea islands. Great secrecy is maintained as to the quantity of phosphate existing on the island. It has been assumed that there is an available tonnage of high-grade rock amounting to 8,000,000 tons, but this estimate is not considered authoritative.

Curaçao (Dutch West Indies)

After a suspension lasting approximately 20 years, the reorganized Curaçao Phosphate Mining Co. began operations in June, 1913, at its Santa Barbara mines, shipped its first cargo in October, and is pushing business now. The phosphate goes to Germany and England. About 200 men are employed, and in one way or another a considerable amount of money is left in the island because of the industry.[33]

Curaçao produces the same high grade of phosphate rock as do the South Sea Islands. It is reported that there came from this island in 1914 about 100,000 tons of phosphate rock averaging 85 to 90 per cent. calcium phosphate. The mining of such high-grade deposits, which will tend to increase as time goes on, owing to the opening of the Panama Canal, may to a certain extent influence the export trade of Florida, but should at the same time tend to conserve these deposits for future use.

Chile[34]

A large, rich deposit of phosphate has been discovered in the valley of the Huasco River, about 300 miles north of Valparaiso. Government engineers are preparing a report thereon, and it is considered of much importance, since the use of phosphate on the farms of Chile is increasing rapidly, with good results. In 1905, only 3,726 metric tons were con-

[33] Consul Elias H. Cheney: *U. S. Daily Consular and Trade Reports*, p. 359 (April 20, 1914).

[34] *U. S. Daily Consular and Trade Reports*, p. 1560 (June 15, 1914).

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sumed in Chile, against 20,000 metric tons for 1912. The Government railways give a reduction of 30 per cent. on transportation charges for fertilizers.

Available Foreign Reserves

From the recent available data, it is evident that the foreign reserves of phosphate are very large, but apparently they are not so large as those within the United States. It must be remembered, however, that the north African deposits, which are thought to extend eastward across Arabia, into Persia, have not been explored sufficiently to know even approximately what their real magnitude is. The Algerian deposits, those considered in some detail in this report and in a former report by the writer,[35] are apparently low-grade, but the apparent tonnage runs up into the hundreds of millions. The high-grade rock of the South Sea Islands is estimated approximately at 70,000,000 tons.

[35] W. C. Phalen: Production of Phosphate Rock in 1913, *U. S. Geological Survey. Mineral Resources of the United States* 1913, Pt. II, pp. 273–289 (1914).

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Geology and Ore Deposits of Mohave County, Arizona*

BY FRANK C. SCHRADER,† WASHINGTON, D. C.

(New York Meeting, February, 1917)

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* Published with the permission of the Director of the U. S. Geological Survey. Originally presented at a meeting of the Southern California Local Section, October, 1915.

† Geologist, U. S. Geological Survey.

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INTRODUCTION

THE present sketch is submitted by request in the hope that it may serve as a basis for geologic discussion of the mining camps in Mohave County, which is experiencing a marked revival of activities.

The region, commonly known as the Mohave district and Kingman district, lies in western Arizona in the southern part of Mohave County, bordering California and Nevada on the west (Fig. 1). Kingman, the principal town, is situated near the center of the area on the Atchison, Topeka & Sante Fe Transcontinental Railway.

This region is composed of naked desert ranges of mountains and broad detritus-filled valleys, the southern extension of the characteristic topography of the Great Basin. In altitude it varies from 500 ft. in the southwest to 8,300 ft. on Hualpai Peak southeast of Kingman.

The mountains trend north-northwest. They rise about 3,000 ft. above the valleys, are generally rugged and were formed mainly by erosion. They are composed in the main of a complex of pre-Cambrian granitoid rocks which underlies the area as a whole. Like the valleys, they average about 10 miles in width. Beginning on the east, they are the Grand Wash Cliffs, the Cerbat Range, the Black Mountains or River Range, and the Eldorado Range.

The upper or dominantly cliff half of the Grand Wash Cliffs, marking the edge of the Colorado Plateau, is composed of nearly horizontal sedimentary Paleozoic strata of the Grand Canyon section, and the lower half of the underlying pre-Cambrian complex.

The Cerbat Mountains situated in the central part of the area, and the Black Mountains situated between Detrital-Sacramento Valley on the east and Mohave Valley, the great trough of the Colorado River, on the west, are locally flanked or overlain by Tertiary volcanics (Fig. 2). The latter consist of five groups of mountains of which the most important is the Black Mesa group on the south.

The Eldorado Range, rising from the great trough of the Colorado on the west and containing Searchlight, Eldorado Canyon, and other camps, is topographically and geologically similar to the Black Mountains.

GEOLOGY OF THE DISTRICT[1]

The rock groups beginning with the oldest are the pre-Cambrian complex, Paleozoic sediments, pre-Tertiary intrusives, Tertiary volcanics, and Tertiary (?) and Quaternary sediments (Fig. 1). The first and third of the divisions named are the most important.

The pre-Cambrian complex consists of gray gneissoid granites, coarse,

[1] A fuller description of the rocks appears in *Bulletin* No. 397, *U. S. Geological Survey* (1909).

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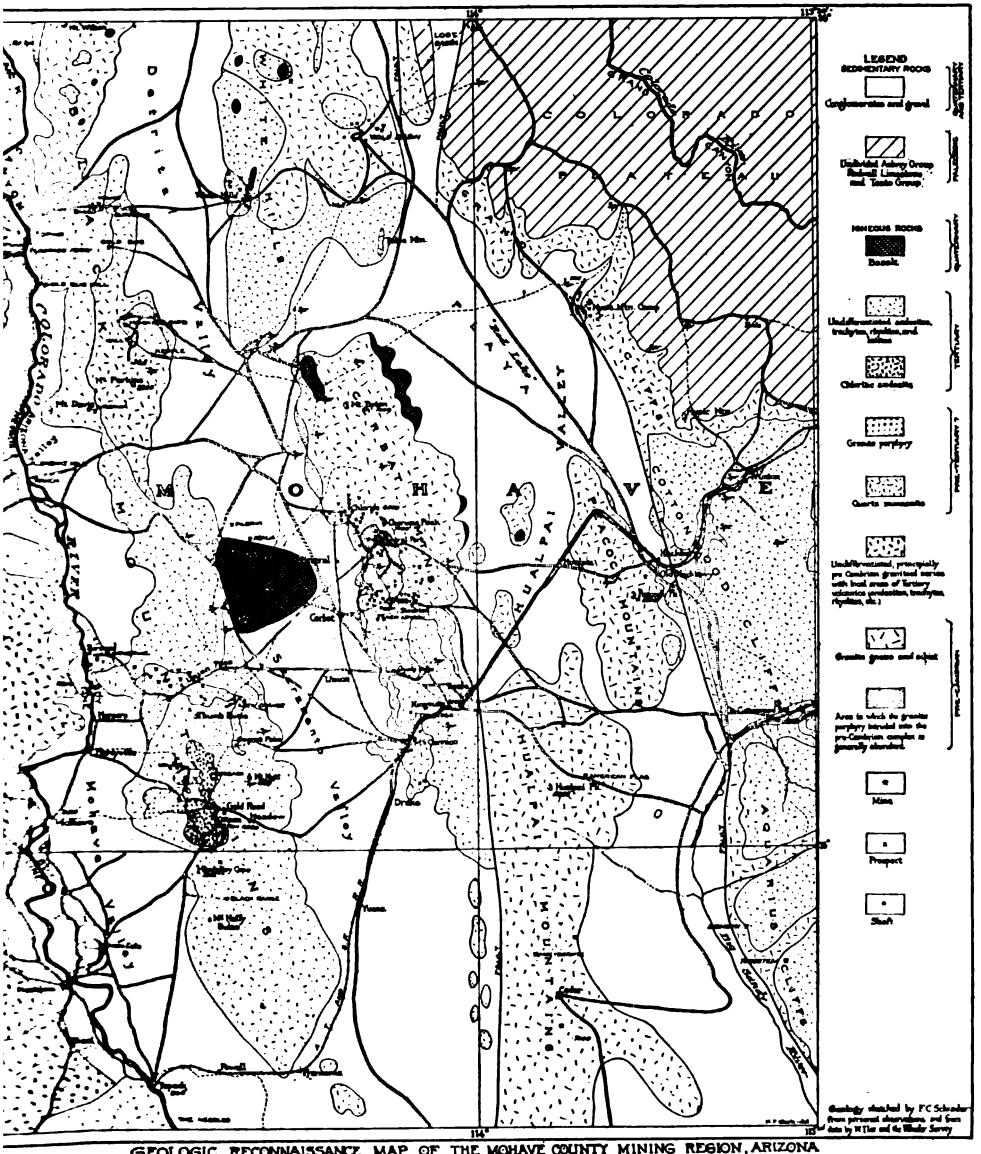
porphyritic granitoids and their related schists, all of igneous origin and with varying degrees of metamorphism. It is traversed by a schistose



GEOLOGIC RECONNAISSANCE MAP OF THE MOHAVE COUNTY MINING REGION, ARIZONA

FIG. 1.

structure trending N. 30° E., and contains nearly all the mineral deposits of the Cerbat Range and Grand Wash Cliffs.

The Tertiary (?) and Quaternary sediments consist of several placer

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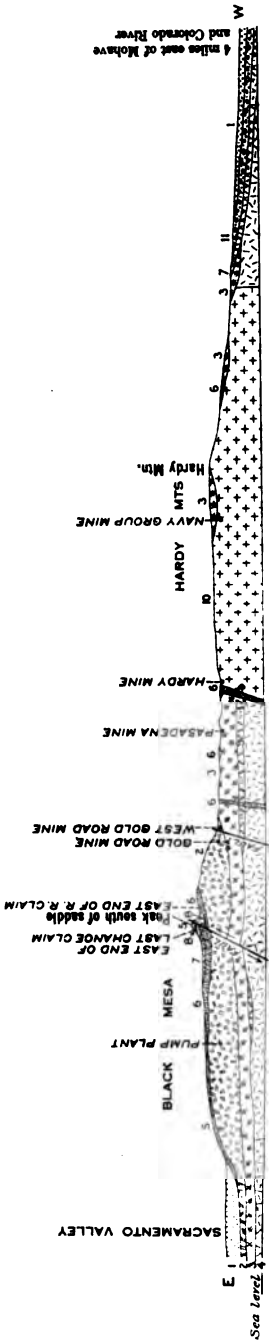

FIG. 2.—GENERALIZED SECTION ACROSS BLACK MOUNTAINS. 1, Sands and gravels; 2, undifferentiated volcanic rocks; 3, green chloritic andesite; 4, gneissoid granite; 5, basalt; 6, rhyolite; 7, young andesite; 8, rhyolite tuffs; 9, andesite and andesite tuffs; 10, granite porphyry and micropegmatite; 11, conglomerate.

Bulletin 397, U. S. Geological Survey.

gold-bearing detrital formations or "wash", locally 2,000 ft. in thickness, partially filling the intermontane valleys.

Locally intruding the pre-Cambrian rocks are pre-Tertiary igneous masses and dikes thought to be of late Jurassic or early Cretaceous age. They occur chiefly in the Cerbat Mountains and are connected with the genesis of the deposits. The most important are granite porphyry, a light gray medium-grained rock, and lamprophyric rocks, the latter occurring mainly as dark, complementary, narrow dikes accompanying the acidic intrusives.

The Tertiary volcanics consist mainly of andesites, trachytes, rhyolites, and latites, lying in broad superimposed sheets, flows and beds locally aggregating 3,000 ft. in thickness (Fig. 2). They are best developed in the Black Mountains, particularly in the southern part (Fig. 3). They contain most of the mineral deposits of the range and played an important part in their genesis.

ORE DEPOSITS OF THE DISTRICT

General Description

The discovery of mineral and the beginning of mining in the Mohave area date from the finding of ore at the Moss mine, 4 miles northwest of Gold Road in the early sixties. From 1904 to 1914[1] the production was nearly $16,000,000, of which $11,500,000 is in gold, nearly all derived from the Tom Reed and Gold Road mines. Besides gold and silver, zinc, lead, copper, tungsten, molybdenum, and bismuth are produced. The distribution of the districts or camps, about 30 in number, is shown in Fig. 4.

[1] *Mineral Resources, U. S. Geological Survey*, 1904–1914.

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The deposits are contained in two distinct groups of fissure veins. The first group consists of the veins of the Cerbat Range which occur chiefly in the pre-Cambrian rocks and are genetically connected with the Mesozoic intrusives, especially granite porphyry and lamprophyric rocks. They are quartz fissure veins in which the quartz carries principally silver but also gold and ores of the other aforenamed metals. They were deposited in depth by hot waters. Their deep-seated character and close association with the major geologic structures indicate continuity in depth. They seem likely to continue productive long after the gold deposits now attracting so much attention in the volcanic rocks of the Black Mountains shall have become exhausted. Oxidation extends to depths of about 300 ft. At present the sulphide ores are principally utilized, though the rich secondary oxidized silver ores furnished most of the early-day production.

The second group comprises the veins of the Black Mountains which occur chiefly in the Tertiary volcanic rocks and whose filling besides quartz includes calcite, adularia, and fluorite. They are deeply oxidized. The valuable constituent is almost wholly free gold.

The Cerbat Mountains Group

General Description.—The deposits of the Cerbat Mountains are mostly located at from 9 to 20 miles north of Kingman. Their production for the year 1915, according to the Chloride Mining Bureau, is $3,000,000. They occur in two sets of well-defined fissure veins, with steep dip-forming conjugate systems, one striking about N. 20° W., parallel with the dominant jointing, and the other N. 60° W. perpendicular to the schistosity of the rocks. Many of the veins have a length of nearly a mile. The structure is irregularly massive. Among the primary-ore minerals, the most important are pyrite, chalcopyrite, arsenopyrite, galena, and sphalerite; more rarely, molybdenite, gold-silver telluride and stibnite. The decrease in galena and increase in pyrite noted in the lower levels, suggests a gradual change in the primary filling. Silver and lead predominate in the Chloride, Mineral Park, and Stockton Hill districts; gold, zinc, and silver in the Cerbat district. The primary ore is leaner in gold and silver than the oxidized ore, and many mines which near the surface were silver mines, with increase in depth carried more lead, and at still greater depths have become cupriferous. The so-called "copper belt" of the area extends from Mineral Park northwestward toward Chloride for a distance of several miles. It contains the Pinkham and Midnight copper mines, and in the Mineral Park end of the belt a recently discovered "copper porphyry" deposit which is attracting attention.

The water level is found at a maximum depth of about 400 ft. In general, the ores above the water level are oxidized, but in many places

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FIG. 3A.—TOM REED MINE AND VICINITY, LOOKING NORTHEAST (IN 1907). DAR

Bulletin 397, *U. S. Geological Survey.*

FIG. 3B.—FOOTHILLS OF GREEN CHLORITIC ANDESITE, LOOKING EAST FROM LELAN
RANGE, 4 MILES DI

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VOLCANIC ROCKS OF RANGE IN BACKGROUND, INTRUDED BY LIGHT-COLORED RHYOLITE.

MINE. MINES NEAR VIVIAN, AND GREEN CHLORITIC ANDESITE IN FOREGROUND; MAIN ANT, IN BACKGROUND.

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(*e.g.*, in the Tennessee vein) galena, and also, locally, pyrite, appears near the surface in association with oxidized ores. In a few mines oxidized ores are found below water level, but not to great depth. The secondary,

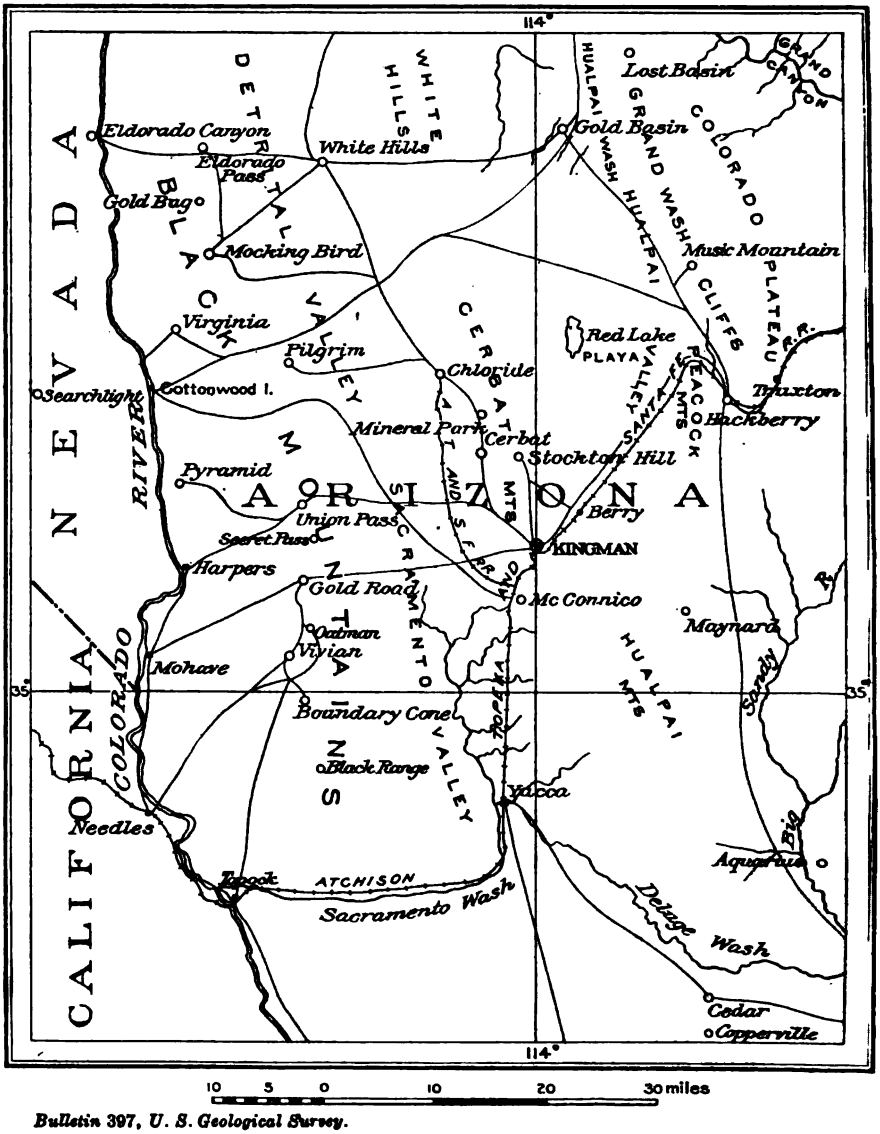

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FIG. 4.—MAP SHOWING MINING CAMPS IN THE MOHAVE COUNTY MINING REGION, ARIZONA.

or oxidized ores, consist chiefly of native silver, horn silver, and cerusite. Ruby silver and argentite are also present with oxidized ore, but do not occupy any well-defined zone between the oxidized and primary ores.

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Many of the oreshoots coincide with intersections or forkings of veins. Good examples were noted in the Pinkham, Elkhart, Rainbow, Pay Roll, and Tennessee mines.

Tennessee Mine.—In the Chloride district, a dozen or more mines are opened to depths of 200 to 1,000 ft. or more, and expose large quantities of good gold-silver and other ores. Among them the Tennessee mine, situated a mile east of Chloride and owned by the United States Smelting, Refining & Mining Co., has long been one of the greatest lead-zinc producing properties of the State. It is credited with a present monthly production of $150,000. It has good orebodies on the 400, 800, 900, 1,000, 1,200, and 1,400-ft. levels, the last-named being the present limit of development. During a considerable portion of the time in recent years it has shipped about 200 tons of ore daily, mostly to Needles. The present daily output is said to be about 300 tons, mostly from the 1,170, 700 and 500-ft. levels.

The mine is on the Tennessee vein, which is regarded as a part of the great lead-bearing "lode" on which the Schuylkill and Elkhart mines to the north are situated. The vein is 12 ft. or more in width, and is locally banded. It dips about 68°E. in pre-Cambrian gneiss, with granite and schist near by and a pegmatite footwall reported in the lower levels. The orebodies which occur as lenses in the vein average about 5 ft. in width. The ore consists mainly of galena and blende, but carries a fair amount of silver and some gold and copper. At present the zinc ore is shipped to the company's smelter at Bartlesville, Okla., and the lead ore to Midvale, Utah.

The mine has been productive almost from the surface down. From between the surface and the 400-ft. level, thousands of tons of rich galena ore have been shipped. Here the main oreshoot had a horizontal extent of about 250 ft., and in places was 15 ft. in width. On the 400-ft. level, an orebody 21 ft. in width with 5 in. of pure galena was mined for the distance of about 40 ft. From the fourth to the fifth level there was a predominance of blende, but from the fifth to the sixth level galena increased to the proportion found in the upper part of the mine.

The 500-ft. level contained good ore for a distance of 800 ft., and the raise from it showed 12½ ft. of almost pure galena. On the 600-ft. level, the vein contained about 10 ft. of good ore. Besides the aforedescribed deposits, large bodies of good zinc ore, some 12 ft. in width, on the 200-ft. and 500-ft. levels, have been left standing in the mine. According to recent reports there has just been opened up on the 1170- and 1400-ft. levels fine bodies of ore averaging about 25 per cent. each in lead and zinc. The body on the 1,170-ft. level has an average width of 8 ft. and a known horizontal extent of 250 ft.

Midnight Mine.—The Midnight mine, located about 2 miles south of Chloride, and adjoining the Pinkham mine, is said to have been recently

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purchased by Salt Lake parties for $250,000. It has produced considerable high-grade copper ore, which contained also important amounts of silver and gold. It is opened to the depth of 300 ft. and is said to have 25,000 tons of pay ore blocked out in the workings, including workable bodies of relatively pure zinc ore. On the 200-ft. level, where the lode is 40 ft. in width, the average zinc content is 15 per cent.

Mineral Park.—The copper porphyry deposit recently discovered near Mineral Park and owned by the Copperfield Copper Porphyry Co., occurs in "porphyry" which seems to be the intrusive granite porphyry aforedescribed, the abundant source of mineralization in this part of the field. The deposit is said to have a width of 1,000 ft. and a length of ½ mile. It contains seams and small bodies of chalcocite and native copper disseminated through the porphyry, which, throughout the greater portion of a 160-ft. crosscut tunnel, carries from 3 to 30 per cent. of copper, with a width of 6 ft., averaging 25 per cent. The deposit is reported to contain by estimate 100,000 tons of 5 per cent. ore. Ore removed in doing development work is reported being shipped to the Humboldt smelter.

Golconda Mine.—The deposits of the Golconda mine operated by the Union Basin Mining Co., in the Cerbat district, occur chiefly in the Golconda vein in the pre-Cambrian complex and seem to be associated with the Mesozoic intrusives. They have produced from essentially surface workings several hundred tons of rich ore containing chiefly gold, silver, and lead with some copper and zinc. The drift on the 300-ft. level is said to have been driven 200 ft. on a 4-ft. oreshoot that averaged about 50 per cent. of zinc, and more recently the mine is reported to be daily shipping to Bartlesville about 100 tons of high-grade zinc ore on which net returns of 9 c. per pound of zinc is realized. Some ore averaging about $12 to the ton is also treated in a 30-ton oil-flotation plant at the mine. The present monthly production is said to be about $250,000.

The mine is reported to have commercial ore on the 1,100-ft. level and a large amount of good ore in all other levels. The present production is derived mainly from the 800-ft. level. On the 900-ft. level, the oreshoot has a known extent of 850 ft. and a large tonnage of high-grade ore is being stoped. From this level a crosscut is being extended to the Tubb vein which parallels the Golconda vein 120 ft. distant on the west and has produced considerable lead-silver ore. On the 700-ft. level, the stopes are working in a 12-ft. shoot of excellent milling ore.

During the year 1915, the mine is reported to have paid two dividends of $85,000 each, which is about 20 per cent. on the issued capital, and having proved the continuity of the oreshoot in depth the company is now erecting a 200-ton oil-flotation plant for treatment of zinc ores. There is said to be $400,000 worth of zinc in the tailings on the dump and in the old stopes in the mine. The mill will be operated by electric power supplied by the Desert Power and Water Co. from its oil-burning

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plant at Kingman for about $12 per horsepower per month. The power line is also being extended to Chloride and the Tennessee mine. The introduction of electric power into the Cerbat Mountain districts seems likely to result in production from many mines now dormant but which, like the Tennessee and Golconda, are known to contain workable deposits. The prospect of cheaper power in the near future is said to be good.

Other Ore-bearing Districts.—Deep development is also being done by the Middle Golconda Co. on the adjoining Big Bethel and Silver claims, which are believed to contain the north extension of the veins of the Golconda mine. Here the main vein is 50-ft. wide and contains much good-grade zinc ore.

A few miles to the east, the Arizona-Butte Mines Co. is building a very complete mill to treat zinc-lead-gold-silver ores of the Banner and other Stockton Hill mines.

South of Kingman in the Yucca, Cedar Valley, and Aquarius Cliffs districts, respectively, plants are in operation producing concentrates of tungsten, molybdenum and bismuth ores.

Lost Basin District.—In concluding remarks on the eastern part of the region, attention is here called to certain copper deposits known for a decade or more in the Lost Basin district on the northeast. The occurrence of these deposits on the trend of the great northwest-southeast mineralization belt of Arizona which contains Bisbee, Ray, Globe, Prescott, Jerome, and other important districts, and under similar geologic conditions as productive deposits in most of those districts, seems to render them worthy of mention at this time.

With a width of about 9 miles, the Lost Basin district extends from the Colorado River at the mouth of the Grand Canyon southward in the Grand Wash Cliffs for the distance of about 20 miles, with Hualpai Wash roughly forming the western boundary. The topography is mostly rugged. The country rock consists of the pre-Cambrian granitic complex, which here is considerably schistose, and the overlying Paleozoic sediments of the Grand Canyon section (Aubrey group, Redwall limestone and Tonto group).

Important gold- and silver-bearing veins occurring mostly south of the middle part of the belt in the pre-Cambrian rocks have long been worked from time to time, and some are now producing. They strike about north. Their ores are fine in texture and are excellent cyaniding ores.

The copper deposits extend from the middle of the belt eastward nearly to the summit of the Grand Wash Cliffs and edge of the Colorado Plateau. They consist of copper-bearing quartz veins and lodes which trend northwest-southeast, some being exposed by erosion through a vertical range of several hundred feet or more. They occur chiefly in the granitic rocks, but some of them, notably on the east, are in the

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Paleozoic limestone and other sediments. The croppings are prominent and some are extensive. They consist principally of masses of brown and blackish iron, copper and manganese-stained quartz containing malachite and azurite. Some of the ore is reported to assay from 17 to 20 per cent. in copper and to contain also gold and silver.

The Black Mountains Group

The deposits of the Black Mountains are mostly on the western slope of the range. They occur in well-defined fissure veins, but differ in most respects very markedly from those of the Cerbat Range. They are found chiefly in the Tertiary volcanic rocks, and belong to the great group of deposits found in this class of rocks throughout the West.

Until recently the most favorable ore horizon was regarded as in the green chloritic andesite and the undifferentiated volcanics, with profitable though subordinate deposits occurring also in the upper rhyolitic series. Recent developments, however, seem to indicate that, in the Tom Reed-Gold Road district at least, the main ore zone probably extends to a deeper horizon, in the so-called older andesite or still lower rocks.

The veins in general trend northwest-southeast with steep northeast dip. They are fairly regular, but the walls are usually rough, broken and frequently full of stringers branching off from the vein. There is a general absence of fluccan or gouge. The gangue primarily was mainly calcite and dolomitic carbonates, but these minerals have largely been replaced by quartz and adularia, a variety of orthoclase free from sodium, semi-translucent and which is so intimately intercrystallized with the quartz that it is not recognizable to the eye. The gangue contains also many inclusions of brecciated altered country rock.

A striking feature of the gangue in many places, particularly in the Tom Reed-Gold Road district, is its characteristically laminated or platy, bladed and cellular structure, pseudomorphic after calcite, barite or other spar in which many contiguous or connecting plates are variously arranged. This material is aptly termed by the miner "fish-scale quartz," from the adjacent plates partly overlapping one another. The plates range from minute up to an inch in diameter and from the thickness of paper to $\frac{1}{10}$ in. in thickness. Much of the quartz intimately associated with the better-grade ore is of greenish or yellowish-green color and waxy luster, which has led to inquiry concerning the source of the color. The cause of the color is not definitely known, nor easy to determine. From preliminary tests it seems to be mainly silicates of iron, manganese, and perhaps other minerals, chlorite, actinolite, rhodonite, dioptase, etc., which may be an important source of the black iron and manganese oxides common as stain, small bodies and pockets in the croppings and more oxidized ores. It is noticeable that the greenish quartz

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occurs more frequently in a crustified or banded form than does the uncolored gangue, which method of forming more readily favors the entering of various salts and minerals into its composition. In the Miller mine on the Hardy vein, 2 miles west of Gold Road, the greenish color of the quartz seems to be due largely to fluorite which is present in considerable quantity in the vein, much of it being replaced by quartz.

The deposits seem to have been formed near the surface by thermal solutions which circulated through the lavas at the close of igneous activity. They seem to belong to the late Tertiary epoch of metallization. They are oxidized to depths of 600 to 700 ft. and, as a rule, contain little or

Bulletin 397, U. S. Geological Survey.

FIG. 5.—GOLD ROAD MINE, MAIN SHAFT LOOKING SOUTH.

Silicified lode and vein wall croppings on both sides. Edges of heavy flows of volcanic rocks in left background.

no sulphides. Gold is almost exclusively the valuable constituent, usually no base metal being present. The gold as a rule is free, but occurs in very minute particles and is best recovered by the cyanide process. Gold telluride is reported from a few mines.

There is no gossan nor iron hat in the outcrops of the veins. In general, the veins weather in relief only where the filling consists chiefly of quartz or a mass of cemented silicified rock. There the croppings form prominent reefs. Likewise, the vein walls are frequently strongly silicified and hardened with the result that they too weather in forms rising to heights of 20 ft. or more above the surface and extending for considerable distances as seen at the Gold Road mine, Fig. 5. This hardened wall

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rock, or so-called "ledge matter," is sometimes netted by stringers of quartz branching off from the vein. It denotes arresting or damming back of copious mineral-bearing solutions that circulated at the locality, and generally indicates workable deposits in the adjacent underlying portion of the vein as described later under the Gold Road mine. Many of the deposits, as exemplified by the Tyro, the Gold Road, and other veins, carry relatively unimportant values near the surface.

Of the ten or more districts in the range the most important is the Tom Reed-Gold Road district.

Tom Reed-Gold Road District

General Description

The Tom Reed-Gold Road district lies about 25 miles southwest of Kingman, mainly on the west slope of the range. In keeping with present usage, the term as here used comprises what was formerly known as the Gold Road and Vivian districts, and the area is approximately co-extensive with the southern part of the San Francisco district of early days. The district has a north-south length of about 10 miles and a width of 6 miles. The principal camps and centers of activity are Oatman, the settlement of the Tom Reed and neighboring mines, situated in the west slope of the range 27 miles from Kingman, and Gold Road, 2 miles north of Oatman. For more than a year Oatman continued to be the center of attraction in the Southwest.

Mineral was first discovered in the district, as aforedescribed for the Mohave County region, in the early sixties at the Moss mine 4 miles northwest of Gold Road. The mine soon produced $240,000, in gold from rich surface ore.

Production in the district has continued more or less steady since the discovery of the Gold Road mine in 1902. Recently discoveries in the Tom Reed mine and vicinity have been attracting attention to the district, with the result that the value of the plants and machinery now installed at the various mines is said to aggregate nearly $2,000,000. Some 50 odd plants are in operation. The greater portion of them have been installed since the first of the year 1915, during which time nearly 200 companies have been organized to operate in the district, of which 150 are fully equipped and most of the others are receiving machinery. Thirty or more properties hitherto dormant have become active, and the population, which is gathered from all the mining camps in the West, has increased from 600 to more than 7,000 and is gradually increasing. Oatman, which is said to recall Goldfields' boom, is described as a well-equipped, substantial town, a cleanly, orderly, model camp, where living is such as one expects to find only in large towns. It is electrically lighted, has three news-

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papers, schools, churches, a stock exchange, and well-stocked stores and business houses of all kinds. It is rapidly becoming the outfitting center for a large territory containing many new districts for a distance of nearly 100 miles north and south along the range. Wildcatting is checked by the laws of Arizona.

Owing to the demand for building space, a new camp, Old Trails, has grown up in the broad wash south of Oatman and there are a dozen other surrounding town sites and additions. Mail, passenger, and express service is by automobile chiefly from Kingman. A similar service exists between Oatman and Needles, 20 miles distant on the southwest, in which the Colorado is crossed by boat. Freight is delivered in the district at a cost of $11.50 per ton by motor truck, road locomotive, and mountain tractor from Kingman, and from Topock, formerly Mellen, located 25 miles distant at the east end of the railroad bridge crossing the Colorado. Early construction of a branch railroad from Topock to Oatman, which is quite feasible, is reported under consideration.

Good water for domestic use is pumped from neighboring springs found mostly in the porous rhyolite tuff or water rock. Seemingly, ample water for milling purposes is being found in deep mining. It also is palatable, being suitable for domestic and all other purposes. Should an additional water supply be required, it can be pumped from the Colorado River 14 miles distant, preferably from wells sunk in the gravel beds on the bank of the river. For this purpose, it is said, a company is being organized.

Development

With a capitalization of more than $53,000,000, operations are being actively prosecuted by 125 separately organized mining corporations and the activity seems to be warranted by substantial results of nearly all deep development. More than half of the companies now in the field are sinking or prepared for deep mining. Ten have good milling ore opened up and four have large producing mines of proven merit. Mining operations are steadily increasing in volume and area; more than 2,200 miners are actually employed in the district and more than $25,000 per day is being expended for wages and equipment.

The approved method of prospecting is sinking to depths of 300 to 500 ft. and then crosscutting and drifting. Practically no surface work is carried on. Gas-engine hoists, compressors, and Jackhamer drills are the usual equipment. Usually also much lateral development must be done before pay ore in large quantities is found and the mine proved. The automobile, a prominent feature in the present activity, has taken the place of the burro in prospecting.

The cost of mining and milling is about $6 per ton, of which $1.25 is for power. The power at the larger mines is electric power supplied

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by the oil-burning plant at Kingman. At the Gold Road mine, treating 200 tons of ore daily, the best record obtained for mining and milling is reported to be slightly less than $3 per ton. At the Tom Reed mine, however, where 20 stamps are used, the cost is about $6. There is said to be no profit in treating $5 ore in the district on a small scale. Both the Gold Road and Tom Reed mines treat their ore by the cyanide process, and have installed the counter-current decantation system.

From what has been said of the Tyro and Gold Road veins, and from the large number of other widely distributed profitable orebodies being found at depth and the cost of mining and milling, this is not a camp for the small operator but seems rather to offer encouraging possibilities for capital to engage in deep mining. The district has received the approval of many eminent mining engineers, a number of whom have become investors there and are now directors in some of the larger companies.

Topography

The district lies mainly in the Black Mesa Mountains, which, with an average elevation of 4,000 ft., extend from Gold Road 20 miles southward to the end of the range east of Needles. Their rugged forms are due chiefly to deep dissection of a huge volcanic plateau known as Black Mesa.

The district ranges in elevation from 2,000 ft. on the west and about 3,000 ft. on the east to 4,500 ft. at the top of the range. The range portion, which is about 4 miles in width, is marked by deep canyons, steep slopes, and peaks. In a horizontal distance of about 1½ miles, the surface declines from the elevation of 4,500 ft. at the crest to 2,500 ft. on Silver Creek just below Gold Road. The edges of the harder lava beds present steplike cliffs (Fig. 3 *A*).

The principal outliers are the Hardy Mountains, a group of hills situated about 3 miles west of Gold Road. They are about 3 miles in diameter and rise about 600 ft. above the surrounding country. Two miles to the north is a smaller group, the Moss Hills, while Leland Mountain at Vivian represents similar features on the southwest.

Geology

The Tertiary volcanic rocks prevail, particularly in the eastern or range portion of the district. They practically constitute the range, dip gently eastward toward its axis and are in places covered by younger rhyolite, andesite and basalt. In the southern part the green chloritic andesite is dominant, while on the west occur also local areas of the pre-Cambrian gneiss, younger granite porphyry and micropegmatite, greenstone agglomerate, and overlying sheets of supposed Tertiary conglomerate and younger gravel and lava flows. Locally intervening between the pre-Cambrian and the overlying volcanics are occasional remnantal patches of tilted and metamorphosed Paleozoic limestone and shale be-

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longing to the Grand Canyon Section. These sedimentary rocks are not as yet known to have any bearing on the deposits or mining other than to indicate to the miner where encountered the general lower limits of the volcanics.

Recent mine developments have disclosed the geology of the ore-bearing volcanics to be more complicated and seemingly of more importance to the district from a gold-producing standpoint than was at first supposed.

In the vicinity of Vivian, and extending from there toward Oatman, occurs the older or basal andesite, which is light gray, calcitic, 300 ft. in thickness, and rests mainly on the pre-Cambrian complex and Paleozoic sediments. The older andesite, however, is not known to be of wide extent in the district, a fact seemingly overlooked by Bancroft and others. It is seemingly absent from Secret Pass where the next higher rock, the green chloritic andesite rests directly upon the pre-Cambrian granite, and from the Hardy Mountains where the green chloritic andesite similarly rests upon the Mesozoic granite porphyry or micropegmatite.[3] It is not known to be present at the Gold Road mine, and according to Sperr[4] the rock underlying the green chloritic andesite in the deep workings of the Tom Reed mine does not correspond to the older andesite described at Vivian. The older andesite is unconformably succeeded by another series of flows, the green chloritic andesite which contains an important part of the mineral deposits in the Tom Reed-Gold Road district (Figs. 1 and 3*B*). The flows aggregate a known thickness of 800 ft. The rock consists mainly of a greenish, fine-grained groundmass containing abundant whitish feldspar phenocrysts. It is very chloritic and calcitic. It is intruded by black latite and younger lavas.

The intrusive character of the green chloritic andesite or rocks grouped with it is well shown at the head of the wash, just west of the Leland mine, where dikes from 2 to 20 ft. in width, given off from the main mass, extend ⅛ mile or more westward into the older andesite. A black, fresh-looking specimen of it collected by the writer from the Leland mine proved by microscopic study and chemical analyses to be latite, and it contains chlorite in abundance throughout.[5]

The intrusive nature of the green chloritic andesite and the association of ore deposits with its intrusive phases in various parts of the district are also abundantly corroborated by later work of Sperr, Probert, Bancroft, and other engineers. Probert[6] believes it to be both intrusive and

[3] *Bulletin No.* 397, *U. S. Geological Survey*, p. 35, and Fig. 2 (1909).

[4] J. D. Sperr: The Tom Reed-Gold Road Mining District, Arizona, *Engineering and Mining Journal*, vol. 101, No. 1, pp. 1–5 (Jan. 1, 1916).

[5] *Bulletin No.* 397, *U. S. Geological Survey*, pp. 36–37 (1909).

[6] Frank H. Probert: Oatman, Arizona—A Prohibition Camp, *Mining and Scientific Press*, vol. 112, No. 1, pp. 17–20 (Jan. 1, 1916).

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extrusive, that dikes and sills of it occur in the older andesite and that mineralization is dependent upon this association.

Bancroft[7] writes that in the vicinity of the mines which he examined in localities rather widely scattered in the district, he found evidence of the intrusive nature of this formation, and that the orebodies are largely formed within the intrusive.

More recently, according to Smith,[8] the bottom as well as the collar of the Tom Reed shaft at 1,075 ft. in depth was in the green chloritic andesite which in the bottom of the shaft was ore-bearing, and he suggests that the rock may here be intrusive. The supposition of the rock being here intrusive, probably as a neck, would help to account for the unusual thickness of the formation at this point, which seems to be local, since elsewhere in the Tom Reed mine and in the neighboring United Eastern, Pioneer and other properties the workings, according to Schader,[9] passed through the green chloritic andesite and into the older underlying andesite at shallower depths and have workable ore in the lower rock.

Therefore, according to the observations of six or more investigators, the green chloritic andesite (formation) contains rocks which vary considerably mineralogically from the normal andesite, rocks with which the ore deposits in general seem to be associated and which are known to be intrusive into the older andesite. The most important of these rocks seems, to the present writer, to be the dark latite occurring at the Leland mine and elsewhere. It seems to intrude not only the older andesite but also the green chloritic andesite as sheets, necks and dikes, and to be intimately connected genetically with the ore deposits. More recently, Sperr,[10] whose observations in the district have been extensive, regards all the commercial ore as occurring in the andesites intimately associated with latites. The intrusive nature of the rocks associated with the ore deposits obviously favors continuity of the deposits in depth.

The deposition of the green chloritic andesite was followed by a period of great fissuring and faulting accompanied and followed by eruption of the next higher group, the undifferentiated volcanic rocks 2,000 ft. in thickness, containing the Gold Road and other important veins, and by intrusions of younger rocks, especially latite and rhyolite in the form of dikes, necks, and rounded plug or stocklike masses, and seemingly the formation of many of the larger fissure veins. The undifferentiated volcanics are succeeded by a series of younger light-colored tuffaceous

[7] Howland Bancroft: Geology of Gold Road District, *Mining and Scientific Press*, vol. 3, No. 1, p. 21 (July, 3, 1915).

[8] Howard D. Smith: The Oatman District, Arizona, *Mining and Scientific Press*, vol. 3, No. 5, p. 172–175 (July 31, 1915).

[9] Carl F. Schader: Personal letter, Feb. 6, 1915.

[10] J. D. Sperr: "Conversational Geology" at Oatman, *Engineering and Mining Journal*, vol. 101, No. 26, p. 1119 (June 24, 1916).

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rhyolites locally 1,000 ft. in thickness and known as the "water rock," which is succeeded by dark reddish andesite which in turn is followed by black olivine basalt, the youngest of the effusive rocks, which remains as a capping over a large part of the Black Mesa Mountains.

With the extensive development recently done in the district, the rocks merit detailed study with reference to their sequence and bearing on the genesis of mineralization. Such a diagnosis seems certain to prove of great economic value in preventing useless expenditure of money in some directions and leading to profitable development in others.

Ore Deposits

General Description.—The deposits, which are numerous, are chiefly gold-bearing fissure veins or lodes of the character already described for the Black Mountains. The veins vary from 5 to 70 ft. in width and from a few hundred feet to several miles in length. In general they are strong and persistent. They strike northwest with steep dip to the northeast. They are almost devoid of metallic sulphides, the gold being free. They occur chiefly in the lower part of the undifferentiated volcanic series, the green chloritic andesite, the granite porphyry and micropegmatite, other underlying rocks and also along certain contacts, where latite and rhyolite are generally the intrusives. Some of the deposits are very rich, but the large bodies of low-grade ore constitute the main resource. Ore having a metallic content of $10 or less is considered low-grade.

The older andesite, from the ill behavior and feathering out of certain vein deposits on entering it from the green chloritic andesite, was originally regarded by the writer as unfavorable for mineral, or essentially barren, particularly in the Vivian district. Owing to its tufaceous brecciated and fragmental nature it is almost devoid of lava-cooling shrinkage cracks and fissures, which elsewhere form favorable repositories for ore deposition. According to Palmer[11] "the occurrence of any oreshoots in the earlier (older) andesite is yet to be demonstrated."

Also E. W. Brooks limits the area of commercial mineralization in this part of the field to the green chloritic or "younger andesite." Later developments, however, it is gratifying to note, in the Oatman and Vivian camps, report workable ore deposits in the older andesite also. It is hoped that with development similar reports may be received from several mines near Vivian which, though well-equipped for operations nearly a decade ago, have remained inactive. That the writer has never doubted that major veins probably occur in and below this formation is evidenced by the following statement: "The veins cut through the great mass of Tertiary volcanic rocks which characterize the range and un-

[11] Leroy A. Palmer: The Oatman Distr[illegible] Arizona, *Mining and Scientific Press*, vol. 113, No. 6, p. 195 (Aug. 5, 1916).

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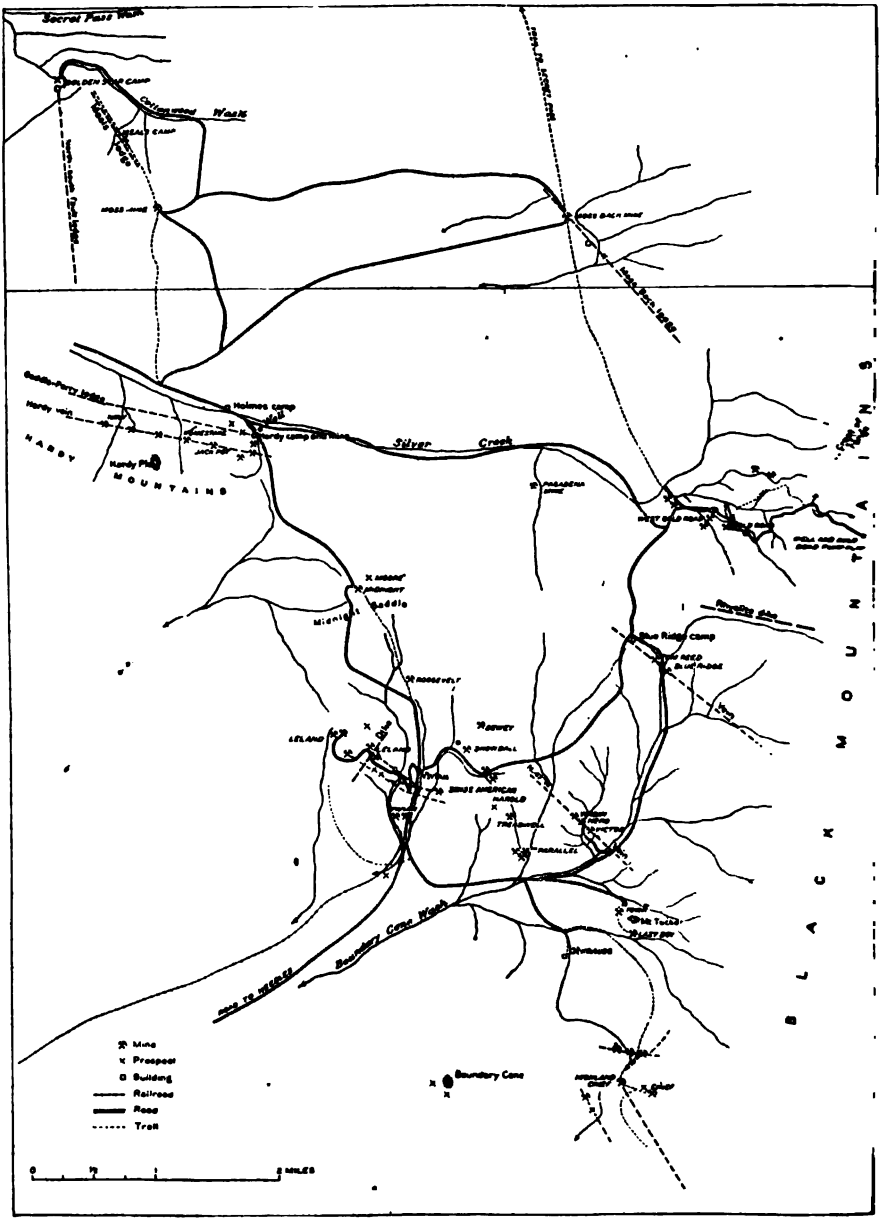

Bulletin 397, *U. S. Geological Survey.*

FIG. 6.—MINES AND VEINS IN THE TOM REED-GOLD ROAD DISTRICT.

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doubtedly continue in depth into the underlying pre-Cambrian granitic rocks.[12]

According to Palmer,[13] "some ore of value has recently been found in the pre-Cambrian."

Since the deposits are confined to the vein filling and do not as a rule form metasomatic replacements in the wall rock, as at Cripple Creek and other camps, the selective preference which any bounding wall rock by reason of its more favorable physical or chemical properties for replacement may exert in favor of ore deposition seems to be practically nil. Accordingly, there is no apparent reason, other conditions being equal, why the deposits should not be equally developed in any one of several formations through which the fissure vein with like strength may extend.

The deposits consist of two types—those in which the gangue is chiefly quartz and adularia and those in which it is chiefly calcite. The source of the quartz and adularia is referred to the siliceous magmas and that of the calcite to basic or andesitic magmas with possible contributions derived from underlying limestones. The former carry the best values, occur mostly in the undifferentiated volcanic rocks and in granite porphyry and have a general northwest-southeast trend. The latter seem to occur mainly in the green chloritic andesite and trend more nearly north-south. Among the most important of the former type are the Gold Road and Tom Reed veins; among the latter, the Pasadena, Mossback and Meals veins (Fig. 6). In some cases the veins are associated with boldly cropping silicified dikes of which the deposits in certain instances may be in part replacement.

According to Platts,[14] the most productive veins, such as those in the Tom Reed, United Eastern, and Big Jim mines, are in a complicated series of fissures, part of which strike about N. 45° W., and others N. 60° W., producing with each other a conjugated system with numerous intersections near which many large orebodies are found.

Zones.—Surficially, the veins seem to mostly fall into four main zones[15] which, named in order from north to south, are the Gold Road, Tom Reed, Vivian, and Black Range zones. The Tom Reed zone is the best developed and contains the most interesting discoveries.

There seem also to be two or more horizons or vertical ore "zones." The largest and richest orebodies seem in general to lie in a "zone" of enriched oxides between the 300-ft. and 500-ft. levels. Below this zone

[12] *Bulletin No.* 397, *U. S. Geological Survey*, p. 48 (1909).

[13] Leroy A. Palmer: *Op. cit.*, p. 195.

[14] J. B. Platts: Geology of Oatman, *Mining and Scientific Press*, vol. 112, No. 23, p. 814 (June 3, 1916).

[15] Leroy A. Palmer: The Oatman District, Arizona, *Engineering and Mining Journal*, vol. 101, No. 21, p. 895 (May 20, 1916).

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the ores decrease in value, but continue to be of workable grade beyond the deepest point yet penetrated by any working. The richness of this zone as suggested by Smith[16] is probably due to secondary enrichment, by contributions leached from shallower depths, in support of which the presence of vugs and manganese oxide in the upper part of the veins is cited. This view is also seemingly corroborated by the tendency of the zone to parallel the contour of the surface. For instance, its occurrence at about the same depth in the Gold Road mine as in the Oatman camp, though at correspondingly greater elevations and higher geologic horizons. The gold was probably precipitated in large part along with the manganese oxide.

If the thickness of 600 or 800 ft. assigned to the green chloritic andesite be correct, this ore zone in the Oatman camp or, more generally speaking, in the triangular area of several square miles comprised between the Tom Reed, Pioneer, and Pasadena mines, should lie mainly in this formation.

There seems to be also present, notably in the Oatman camp and vicinity, a shallow or surface ore zone of leached oxides to which pay ores found at or near the surface are generally confined. It extends to depths of about 150 ft., between which and the zone of enriched oxides, or 300-ft. level, lies a 150-ft. intermediate zone of leached or relatively barren ground, although the valuable oreshoots, according to Sperr,[17] almost without exception come at least within 100 ft. of the surface.

These two zones have probably suffered about the same amount of leaching, the upper zone certainly not less than the intermediate or barren zone. The upper zone seemingly owes its greater ore content to the more siliceous, and consequently resistant, character of the ore which accordingly better withstands the process of leaching.

Differing from the view of enrichment by leaching and redeposition in the main zone is that of Platts[18] which holds that the ore is essentially a primary deposit formed by heat ascending solutions, that from the nature of the gangue it is evident that acid solutions could not exist, and that, except for the oxidation of the pyrite, there is no evidence of the action of surface water on the ore.

It seems quite possible, as suggested by one writer, that the groundwater table in the district may be in part dependent upon the neighboring Colorado River. If this view be correct, physiographic study will probably be able to correlate certain horizon features of the vertical section as leaching, etc., with relatively prolonged pauses in the historical downcutting of the river. It does not, however, seem safe to assume that the water table at Oatman coincides with the level of the Colorado River,

[16] *Op. cit.*, p. 173.

[17] J. D. Sperr: "Conversational Geology" at Oatman, Ariz., *Engineering and Mining Journal*, vol. 101, No. 26, p. 1119 (June 24, 1916).

[18] J. B. Platts: *Op. cit.*

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elevation of the gathering zone on the east, which probably extends to the Hualpai Mountains, or longitude of Kingman, the ground-water table is not a level surface but gradually rises from the Colorado River, eastward, and at Oatman it probably stands several hundred feet or more above the level of the river.

Structure of Oreshoots.—The ore occurs chiefly as a series of more or less tabular or lenticular oreshoots and pay streaks dipping and plunging variously within the vein, with which they exhibit a greater or less degree of parallelism. The shoots vary from 1 ft. wide to the width of the vein. They usually carry gold for their full width. They range up to nearly 1,000 ft. in length and depth, and there is a general similarity or repetition of the shoots in the same vein. They seem to have been formed by thermo-aqueous processes that followed igneous activity. In general, the quartz and values favor the hanging wall, which of the two walls is generally the best defined, and contains stringers branching off obliquely from the vein, while the spar or calcite favors the footwall. The gold is mostly associated with the quartz-adularia gangue and not rarely where sulphides have existed, it, according to Platts,[20] occurs in hematite (which is pseudomorphic after pyrite) in the quartz.

According to Palmer,[21] the first indications of the vein encountered in sinking are small stringers of quartz and calcite scattered through the andesite, usually accompanied by slight pyritization in the vein-wall andesite which yields a little free gold in the pan, while in the oreshoots the vein matter shows pronounced hematite and manganese stains. It is said that the problem in development is not so much the finding of veins as the discovery of oreshoots in the veins, that nothing sufficiently tangible has yet been found to use as the basis for a theory to guide the operator in the search for ore.

Though no rigid rule can be laid down to guide the operator in search for ore, nevertheless, from the apparently well-established facts that the metallic values have been largely imported by the replacement quartz-adularia solutions and that more gold is found where the replacement of calcite is most nearly complete, in formulating plans of exploration much benefit in most cases should be derived from a correlative study of the criteria indicating the probable courses followed by these solutions, namely, quartzose vein croppings, silicified wall rock, the quartz pseudomorphic structures, etc., which have been described. It was the quartz

[19] L. A. Palmer: The Oatman District, Arizona, *Mining and Scientific Press*, vol. 113, No. 6, p. 196 (Aug. 5, 1916).

[20] J. B. Platts: *Op. cit.*

[21] L. A. Palmer: *Idem.*, vol. 101, No. 21, p. 896.

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adularia or siliceous waxy-appearing character of the deposits seen in the Tom Reed mine and the recognition of their marked similarity to the then-producing deposits of the Gold Road mine that apparently led to the resumption of operations in the Tom Reed mine following the writer's visit in 1907.

Gold Road Mine

Some of the principal mines in which the deposits have been worked are the Gold Road, Tom Reed, Leland, Pioneer, Victor-Virgin, Midnight, and United Eastern. Their general distribution is shown in Fig. 6. The most important producers are the Gold Road and Tom Reed mines.

General Description.—The Gold Road mine, owned by the United States Smelting, Refining and Mining Co., is situated at Gold Road, on the western rugged slope of the range about 1 mile below the crest, at an elevation of about 2,900 ft., mainly on the western part of the Gold Road vein. From its croppings, the Gold Road vein has long been known, but the discovery of values which resulted in the opening of the mine was made about 1902. The mine soon began to be worked systematically and paid dividends of 5 per cent. on the capitalization of $500,000. Several years ago it was acquired by the present owner for $1,500,000.

The mine is opened to a depth of 1,000 ft. or more, mainly by shafts and drifts, and ore has been mined in quantities for a distance of 4,000 ft. on the vein.

Most of the work done up to 1907 was comprised in the ground extending about 200 ft. from the main or Gold Road shaft (then known as the Gold Road mine), in either direction on the vein, and to the depth of 500 ft. Since then, however, extensive developments have been made, especially to the east on the Billy Bryan and adjoining ground.

The gross production of the property is estimated at more than $6,000,000. To the end of 1907 the production was about $2,250,000, most of which was made during 1905 and 1906. Since 1909 the output has averaged about $800,000 annually, which figure in some years was much exceeded. The present monthly production is nearly $80,000 in gold bullion.

Geology.—The country surrounding the mine is occupied by the undifferentiated volcanic rocks consisting mainly of heavy sheets of volcanic flows (Fig. 5). The series comprises andesite, trachyte, latite, rhyolite, and dacite, which aggregate nearly 2,000 ft. in thickness and are difficult to separate. The series extends from a point about 1,000 ft. west of the main shaft eastward nearly to the crest of the range. On the west it gives way to the underlying green chloritic andesite, which in the West Gold Road mine, situated near the contact, has been penetrated to a depth of 455 ft. On the east it is overlain by the upper rhyolite. The

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contact on the west is probably a fault contact, as some of the green chloritic andesite occurs in the main shaft workings. The dominant dynamic structure is a pronounced close sheeting, which strikes N. 40° W., with vertical dip. The series is intruded by dikes of rhyolite and the younger dark andesite. Much of the rhyolite carries gold values. A dike of it, 40 ft. in width, located about 600 ft. south of the mine, is said to average about $5 in gold to the ton, the gold occurring chiefly in contained stringers of quartz and calcite.

Ore Deposits.—The Gold Road vein extends from a point about 700 ft. west of the main shaft southeastward through the Gold Road, Rail-

Bulletin 397, U. S. Geological Survey.

FIG. 7.—GOLD ROAD VEIN AT EAST OF LINE ROAD CLAIM, LOOKING NORTHWEST.

road, Billy Bryan and Last Chance claims to beyond the crest of the range, a distance of nearly 1¼ miles. It dips about 80° NE. approximately parallel with the close sheeting in the country rock. It varies in width from 22 ft. on the west to less than 1 ft. on the east. Between the bottom of the mine and the crest of the mountains it has a known vertical range of about 2,300 ft., and its croppings have a vertical range of about 1,300 ft. They consist essentially of iron and manganese-stained quartz, silicified rock, and calcite. In places they form conspicuous reefs or knobs rising 20 or 30 ft. above the surface, as at the main shaft or Gold

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Road mine (Fig. 5). Here the associated hard silicified prominent wall-rock croppings on both sides of the vein have a lateral extent of 20 to 60 ft. The best ore usually underlies these prominent croppings, which seem to represent pool-like courses along which mineralizing solutions deposited more freely than elsewhere. Where the croppings weaken or break down, the underlying portion of the vein generally becomes lean or barren, though the fissure and its walls and filling may continue unchanged.

The vein consists essentially of quartz and adularia, with some calcite and brecciated altered rock, and is locally more or less crustified (Fig. 7). Since the more siliceous portions frequently exhibit a perfection of crustification not found in the calcic portions, and it is difficult to see how this structure could be derived from massive calcite by process of replacement, the quartz-adularia filling in the well-banded portions is regarded as largely primary rather than replacement. It seems to have been deposited in reopenings of the calcite vein and extension of the fissure itself eastward into the axis of the range.

In 1907, the vein in nearly all workings within 200 ft. of the main shaft and from the 300-ft. level to below the 700-ft. level was mostly good milling ore from wall to wall, the amount of waste in mining as shown by the dump (Fig. 5) being very small. Elsewhere, however, as seen on the Billy Bryan and other ground, portions of the vein, sometimes for a considerable extent, are relatively barren.

The vein is strongest on the west, where, as developed in the main shaft workings, it is uniformly about 10 ft. in width. It is usually in sharp contact with well-defined firm walls of the country rock, consisting of andesite, trachyte, latite, and rhyolite. It is, in general, "frozen" to the walls and is locally enriched where stringers extend from it at acute angles into the hanging wall. As a rule, hard and rough walls indicate good ore, and conversely, soft and smooth walls generally correspond with lean ore. The ore consists chiefly of fine-grained, light-greenish or waxy quartz. Much of it is platy or hackly with a peculiar chalcedonic or drusy appearance. Much of it is pseudomorphic after calcite and many of the pseudomorphic plates are thickly studded with minute quartz crystals of a still younger growth. The gold, as seen microscopically, and in places by the naked eye, is very finely disseminated, principally in the quartz-adularia portion of the gangue. On the 600-ft. level, however, the vein contained more spar or calcite than on any other level, and here the oreshoot is reported to have had an uninterrupted extent of 1,100 ft. Some sulphide ore containing pyrite, the first encountered in the mine, was found here, and it is reported to have contained higher values than the overlying oxidized ore, which is probably due to arrest in the downward progress of leached concentrates. On the 700-ft. level, the country wall rock, consisting of green chloritic andesite, was more or

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less pyritic and seemed to indicate that the mine was entering the sulphide zone. The mine has not produced much ore from depths shallower than the 300-ft. level. The lowest-grade ore taken out was obtained between the surface and the 300-ft. level, where it fell to $5 a ton. The ordinary mill-run ore, it is said, rarely falls below $8 or $9 and ranges from that up to $22 a ton. It averages about $10 to the ton, but $100 ore and upward is occasionally encountered on the hanging wall, where nearly all the high-grade ore occurs.

On the Billy Bryan claim, nearly ½ mile from the main Gold Road shaft, the vein and the associated rocks show essentially the same characteristics and relations as in the Gold Road mine. The variations and phases of the rocks are well displayed in the large dump at the mouth of the drift. The principal rock corresponds to trachyte. It is in general considerably altered. The croppings of the vein are prominent and wall-like. The vein varies from 1 to 20 ft. in width and dips, in general, steeply to the southwest instead of normally to the northeast. It consists mostly of spar or calcite, which contains bands of greenish quartz. Beyond the southeast limits of the Billy Bryan claim, and, in fact, to the end of the Railroad claim, quartz prevails and carries good values. The quartz is greenish with glassy luster and locally is very brittle, closely crustified, wavy, and crinkled, its structure resembling the fine flow structure of certain lavas. Weed,[22] having access to later development, described the vein filling as "the usual mixture of calcite flakes and waxy quartz varying to a dense waxy yellow quartz in curly or crinkled bands and shreds sometimes showing a faint blackish stain."

In the eastern part of the Line Road claim, and in a considerable portion of the Billy Bryan and Railroad claims, the vein has produced very rich shipping ore from surface workings, and it is under these workings, notably on the Billy Bryan and Railroad claims, that the Gold Road Co. in 1908 encountered good ore in depth.

In its extension eastward through the Last Chance claim, and across the crest of the range, the vein contains some good-looking croppings and openings, but it is split by horses at several points and locally narrows to unworkable dimensions. In August, 1915, the mine was reported to have good-sized bodies of $20 ore on the 800-ft. level, and in November it is said to have encountered on the 900- and 1,000-ft. levels a body of higher-grade ore than any hitherto found on the lower levels.

Recently it is reported that the vein as stoped averages about 12 ft. in width. Three oreshoots are being stoped on the 500-ft. level and at greater depths. These are respectively 1,200 ft., 70 ft., and 700 ft. in length, being separated by small intervals of barren ground.

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[22] Walter Harvey Weed: The Kingman District of Arizona, *Mining World*, vol. 32, No. 23, pp. 1113–1114 (June 4, 1910).

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Tom Reed Mine

General Description.—In the last year or two, owing to new discoveries, interest in the district has centered mainly in the Tom Reed mine and neighboring properties, wherefore this part of the district is known as the Tom Reed district or camp, Oatman district or camp, or simply Oatman. Here the geology and ore deposit are similar to those of the Gold Road camp, except that the deposits occur largely in lower geologic horizons in the green chloritic andesite and still older rocks (Fig. 3*A*). Some of the veins are "blind," being covered by later flows of rhyolite and younger rocks whose contact is traceable by the softened character of the contiguous weathered portion of the underlying rock.

The Tom Reed (formerly Blue Ridge) mine is situated at Oatman about 2 miles south of the Gold Road mine and about 200 ft. below it. It lies on open ground in Blue Ridge wash, near the base of the central part of the range, at an elevation of about 2,700 ft. (Fig. 3*A*). It is one of the well-known mines of the country and contains well-defined ore-shoots, which for nearly a decade have been worked with great profit.

It was discovered about 1900 and was soon after owned by a party composed of Ely Hilty and others. About 1901, the Gold Road Co. sunk two shafts on the property, the Ben Harrison and the Tom Reed shafts, each to a depth of about 100 ft. with good results. About 1904 the mine was purchased by the Blue Ridge Gold Mines Co., which installed a mill and operated the mine and mill for about a year and a half, milling on an average about 30 tons of $7 ore a day. In 1906 the Blue Ridge Co. was succeeded by the present owner, the Tom Reed Gold Mines Co., which resumed operations in 1907, and in 1908 the mine was reported to be working and making gold bullion shipments regularly, since when it has been a steady producer. A little later a 12-ft. wide body of $12 ore is said to have been encountered on the 300-ft. level.

The property comprises a group of 11 or more claims, adjoining one another in part end on and extending along the vein for a distance of about 3 miles. The mine is located near the middle of the group.

Ore Deposits.—The country rock is mainly the green chloritic andesite. The vein, the Tom Reed (formerly Blue Ridge) vein, strikes about N. 50° W. and dips about 70° NE. It nearly parallels the Gold Road vein on the north and the Victor-Virgin vein on the southwest, to both of which it is geologically and mineralogically similar. On the Tom Reed property, it has an extent of about 3 miles, Fig. 6, and it is said to be continuous with the Pasadena vein on the northwest, in which event it has a length of about 4½ miles, being probably the longest vein in the district. At the Tom Reed mine it ranges up to about 40 ft. or more in width with the fissure walls usually ill defined. It outcrops boldly for the length of nearly two claims. The croppings consist principally

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of the usual dark iron and manganese-stained quartz, silicified rock, and calcite.

The vein is mainly of the quartz-adularia-calcite type. Of the early-day ore, a considerable portion is reported to have run $25 in gold to the ton for the first 30 ft. in depth and about $12 from that point down.

In the Ben Harrison shaft and its workings, the vein has a width of 16 to 22 ft. On the 100-ft. level the vein consisted chiefly of 16 ft. of crushed quartz and rock with neither wall well defined; but toward the hanging-wall side there was 6 ft. of good-looking, more or less porous, clear quartz ore.

On the 150-ft. level the vein consisted mainly of crushed rock, but the footwall side of the drift was in quartz; the hanging wall contained vugs 6 in. to 1 ft. in diameter, containing blackish, porous, oxidized quartz ore.

Development.—The mine is opened to a depth of 1,400 ft. It has more than 30,000 ft. of underground work, with the longest drift extending more than 4,000 ft. out from the shaft and in ore nearly all the way. The production to June, 1916, was nearly $6,000,000. That for the year ending March 31, 1916, was $661,870.68, the average value of the ore being $22.12 to the ton and the extraction 98.6 per cent. Dividends paid during the year amounted to nearly $164,000, or 18 per cent. on the par value of the outstanding stock. By estimate 11,000 tons of ore were blocked out in the stopes at the end of the year. The mine has paid more than $2,500,000 in dividends. The net realization on the mine by June, 1913, was $3,019,569.75. By June 24, 1914, the 44th dividend had been declared. Later the mine was reported to be paying for the last several years monthly dividends of from 6 to 7 per cent. on the par value of the stock, and the ore then blocked out, by estimate $2,000,000 worth, was said to be sufficient to continue their payments for several years to come. By 1907 the production considerably exceeded $120,000, and that since 1910 is more than $5,000,000. The annual production for the last few years is reported to average about $1,200,000 in bullion, besides which a large tonnage of ore is accumulating at the mine, especially of $10 ore in the workings.

Up to June, 1913, much of the ore produced had been drawn from an orebody between the third and fifth levels, where about an equal amount remained, and this same orebody had been proven to a further depth of 200 ft. below the fifth level. The seventh-level drift, at this time 233 ft. in length, was all in ore of about the same average value as that on the fifth level. Recently, in the Black Eagle section of the mine, the crosscut on the 400-ft. level is said to have passed through 35 ft. of good ore, 30 ft. of which averages $25 to the ton, and the ore tonnage on the 600-ft. level is very large.

Later, good orebodies were reported on the 500, 600, 700, 900, 1,075,

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1,200 and 1,400-ft. levels. The ore on the 1,200-ft. level is said to be similar in character and grade to that on the upper levels. Progress work on the 1,400-ft. level it is said has revealed a vein width of 12 ft. with a large orebody having a known extent of 300 ft., averaging $12 to the ton and containing some high-grade ore.

Explorations on the 1,075-ft. level for 225 ft. west of the shaft have shown the oreshoot throughout that distance to have a width of about 18 ft. and to range from $22.50 to $40 to the ton. On and below this level the vein is reported to be disturbed by a fault, but at the 1,175-ft. level it has fully recovered its former size and values.

Character of Ore.—The ore is similar to that of the Gold Road mine. It consists of a mixture of flaky calcite, waxy quartz, adularia, brecciated altered rock, and pinkish argillaceous material which is frequently of high grade. According to Weed,[23] a dense quartz whose color and luster closely resembles that of beeswax constitutes the richest ore. The ore is not hard and most of the gold is free, especially in the ore from the lower levels, but it requires fine grinding to free the finely disseminated gold and expose it to the action of the cyanide. The gold is seldom visible even in rich ore.

In milling, the total sliming system of cyanidation is employed, followed by treatment of the ore in Pachuca vats and Dorr thickeners. "Dorr thickeners," according to Smith, "appear to be particularly suited to conditions in this district where little silver is present, weak solutions are used, and the slime settles quickly." The ore amalgamates about 50 per cent. of its value on the plates and a high extraction is obtained at reasonable cost by cyanidation. In 1910, the average extraction was $42.46 to the ton. In 1912 it was $23.22. The amount of ore treated in 1911 was 39,447 tons; in 1913, 948,110 tons, with an average extraction of $24.09, and a recovery of 97.05 per cent. The average value of the ore mined in the fiscal year 1914 to 1915 is $21 to the ton. The gold is generally pure, the proportion of silver present being very small. The mill treats about 4,000 tons of ore a month.

Other Mines

Among the new properties which are attracting most attention is the United Eastern which adjoins the Tom Reed mine on the northwest and is often referred to as the "Bonanza." It is but a year and a half old, and is reported to have in sight, according to the estimates of conservative mining engineers, $11,000,000 worth of $26 ore. The mine contains more than 2,000 ft. of drift and has good orebodies on all levels between the depths of 300 and 700 ft. The vein is reported to be 43 ft. wide on the 555-ft. and 665-ft. levels, and the entire width is pay milling ore. Of this

[23] *Op. cit.*

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width, 30 ft. proven for the distance of 650 ft. averages $40 to the ton and carries considerable free gold. Since August, 1915, daily shipments of $30 ore removed during development are being made to the Gold Road mill. A new 200-ton cyaniding plant, in which gyratory crushers and ball mills instead of stamps will be used, is being installed at the mine. The equipment will be adequate for sinking to the depth of 2,000 ft.

Many other new properties like the United Eastern are being opened up and in many of them good ore is being found at depths of from 200 to 500 ft. A score or more are worked by incorporated mining companies.

Extensive developments are being undertaken at the Pioneer (formerly German-American) mine by the Oatman Pioneer Mines Co. which by coöperation and use of its efficient machinery and 400-ft. level will immediately facilitate the exploration and working of adjoining properties. The Pioneer is on one of the three main lode outcrops of the district, and is said to have $1,000,000 worth of commercial ore in sight between the 400-ft. level and the surface. It is working on two veins of the 400-ft. level, of which the northeast, or Pioneer vein, has an 8-ft. oreshoot having a known extent of 600 ft. and averaging $16 to the ton, and on the southwest, or Snowball vein, the crosscut has penetrated a width of 16 ft. of good-grade ore with the outer wall not yet reached. On the 200-ft. level, a 2-ft. shoot of high-grade ore is being worked.

The Boundary Cone mine adjoining the Pioneer on the east has good ore on the 750-ft. level, and is said to contain a 5-ft. shoot of ore that averages $100 to the ton on the 200-ft. level. On the 500-ft. level, where the oreshoot has been proved for a distance of 90 ft., it has a width of 12 ft. and averages about $20 to the ton. The mine is credited with a considerable production of high-grade ore, some of which contained crystallized gold.[24]

In the Big Jim mine, a mile northwest of Oatman, the vein on the 400-ft. level is said to be 51 ft. in width, of which 46 ft. averages about $8 to the ton and 8 ft., on the hanging-wall side, $15 to the ton with some pay streaks which are very rich. On the 485-ft. level, the same oreshoot has been opened for the extent of 200 ft. and is good milling ore, most of which averages more than $12 to the ton. Here the ore is said to be more oxidized and silicified than on the upper levels. The vein parallels the projected course of the Tom Reed vein, but whether it is the northwestward extension of the Tom Reed vein which may here be faulted to the northeast, or a new vein, has not yet been determined. The mine is daily shipping about 30 tons of ore removed in development.

In the Carter mine, ½ mile south of Oatman, the main oreshoot, about 15 ft. in width descending from the 150-ft. level to the 400-ft. level, is said to contain 5 ft. of ore which averages about $30 to the ton, and 8

[24] W. P. DeWolf: The Tom Reed-Gold Road, *Salt Lake Mining Review*, vol. 17, No. 13, p. 16 (Oct., 1915).

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ft. that averages $8 to the ton. This property is said to have shipped, several years ago, some very rich ore taken from between the surface and the 150-ft. level. This same vein is thought to extend through the adjoining Telluride and Lucky Boy properties, where on the 300-ft. level the entire width of 25 ft. is good milling ore.

The Gold Reed mine, a mile south of Oatman, has 4 ft. of $32 ore on the 375-ft. level.

On the Times property, the vein recently opened at 270 ft. in from the portal of the Martin tunnel, reveals 5 ft. of ore, averaging nearly $24 to the ton.

On the north, the Gold Ore mine, ½ mile northeast of Gold Road, is credited with a 9-ft. vein on the 500- and 550-ft. levels containing a 6-ft. shoot of ore which for the distance of 300 ft. averages about $12 to the ton, and a considerable portion of it nearly $100 to the ton. More than 75,000 tons of ore are said to be blocked out above the 550-ft. level. Daily shipments of ore averaging nearly $25 to the ton are made to the Gold Road mill.

The old Moss mine, where the original discovery of mineral in the Mohave County region was made, is being developed by the owners of the Gold Road mine and $60 ore is being mined from the 200-ft. level.

In the Ivanhoe mine, 2 miles northwest of Oatman, the vein, whose footwall is a partially mineralized 75-ft. "quartz porphyry dike cutting andesite and underlying sedimentaries," on the 250-ft. level, has a width of 60 ft., and on the 500-ft. level 6 ft. of milling ore has just been crosscut on the footwall side. Some high-grade ore has been shipped to the Gold Road mill.

Attention is also being attracted to the Secret Pass district, 6 miles north of Gold Road, which with a small mill is making considerable production mostly from high-grade surface ore. Here the occurrence is unusual, the ore, according to Payne,[25] being found chiefly as replacement deposits in the rock walls of a fissure occupied by a dike which the ore seems to postdate. The bullion, which is shipped to the U. S. Mint at San Francisco, is said to average about $15 to the ounce in gold. The population during the last few months of 1915 increased from 100 to 400, with prospecting extending over an area of several square miles.

At about 6 miles south of Oatman, promising deposits are reported in the Black Range zone, where a dozen companies are operating. Concerning the rocks in this part of the field, which have been roughly grouped with the undifferentiated volcanics, little is known as yet.

The deposits occur in a series of veins of which the one on which the principal mines are located is a prominent quartz outcrop known as the Nellie vein. It has been traced for the distance of several thousand feet and opened to the depth of 300 ft. on the Black Range, Nellie and Green

[25] C. Q. Payne: Oral communication.

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300-ft. level, of which 3 ft. averages about $30 to the ton. Here, also, the country wall rock extending along the vein, for a width of 60 ft. or more, is impregnated with replacement deposits and averages nearly $6 in gold to the ton.

In referring to the future of the Tom Reed-Gold Road district, some men favorably compare it with Goldfield, Cripple Creek or other large camps and hold that it will become one of the greatest gold-producing districts in the United States. Among the more conservative and seemingly reasonable views is that of Palmer,[26] who believes that it will become comparable with the Tintic district, Utah, that it will become a large low-grade camp with several producing mines which will yield dividends for many years to come.

Fields Similar to the Tom Reed-Gold Road District

Recent investigations[27] have shown that the southern end of the Black Mountains containing the Tom Reed-Gold Road district is the easterly one of a number of similar volcanic areas which extend interruptedly westward on either side of the railroad through the distance of nearly 100 miles to the longitude of Barstow, Cal. In these areas—which embrace the Mohave and Chemehuevis Mountains, the ranges west of Von Triger, Clipper Mountain, the Cady Mountains, the Newberry-Ord district, and the well-known Calico Mountain group and others—the geological and mineralogical conditions are very similar to what they are in the Tom Reed-Gold Road district. The areas lie from a few miles to 25 or 30 miles back from the railroad. Their volcanic rocks, which in character, recurrence, and succession, are in general identical with those in the Tom Reed-Gold Road district, range up to 2,000 ft. or more in thickness, are frequently well-mineralized, and contain strong veins. Most of the areas have been but little prospected, but some, as that of the Calico Mountain group, are productive.

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[26] *Op. cit.*, p. 900.

[27] N. H. Darton and others: Guidebook of the Western United States, Part C, The Santa Fe Route, *Bulletin No.* 613, *U. S. Geological Survey*, pp. 142 to 162 and sheets 21, 22, and 23 (1915).

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Influence of the Movement in Shales on the Area of Oil Production

BY RICHARD A. CONKLING,* A. B., TULSA, OKLA.

(New York Meeting, February, 1917)

A SHALE layer, buried beneath two or three thousand feet of strata, in some instances, will upon folding become thicker in the synclines and thinner on top of the anticlines.

This can be accounted for, in part, by the stretching on the crests of the fold and the compressing in the troughs; but this will by no means account for all of it, as is shown in the example herein set forth.

It is my firm belief that the rest of the thickness is due to flow. What causes the movement, however, does not concern us here, so long as there is movement, for this article purposes to show its effect upon the producing area of different sands.

It is the author's hope, in this way, to drop a hint or two that may be valuable to the oil geologist, in making estimates of future productions and values of oil properties. An example will be given from property in the famous Cushing Pool in northeastern Oklahoma, where the author has had occasion to make a detailed study before recommending some property to his company, and has then been able to watch the results of this work.

The author never mapped the surface structure in this pool, as this had been done by Frank Buttram when he was with the Oklahoma Geological Survey. We are using, therefore, a section of his map in this article for our surface contours. The main east dip comes farther east than this section shows.

From the 10-ft. contours in Fig. 1, it will be seen that the surface folding is very slight, but in the contour map of the Bartlesville sand (Fig. 2), which was the greatest producing sand, we see much more complex folding. It seems that the folding becomes greater with the depth. Folding may have started in Middle Pennsylvanian times and continued through the Upper Pennsylvanian which is found at the surface. This may, or may not, have been the cause of the increase of folding with depth. However, the fact that mostly concerns us is the interval of shale between the Bartlesville sand and the Tucker sand. The Tucker is the next productive sand below the Bartlesville. Not enough wells have been drilled to the Tucker sand to make a contour map of it, but

* Head Geologist, Roxana Petroleum Co.

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the figures beside the wells on Fig. 2 show the interval between the top of the Bartlesville and the top of the Tucker.

The entire area shown in Fig. 2, except the southeast corner, was productive in the Bartlesville sand. On the west edge of Section 10, a syncline will be observed. Water was encountered in this syncline at the bottom of the Bartlesville, but it did not rise high enough to ruin the wells for some time. The producing area of the Tucker sand is much smaller, however, as the surface of the sand dips more steeply

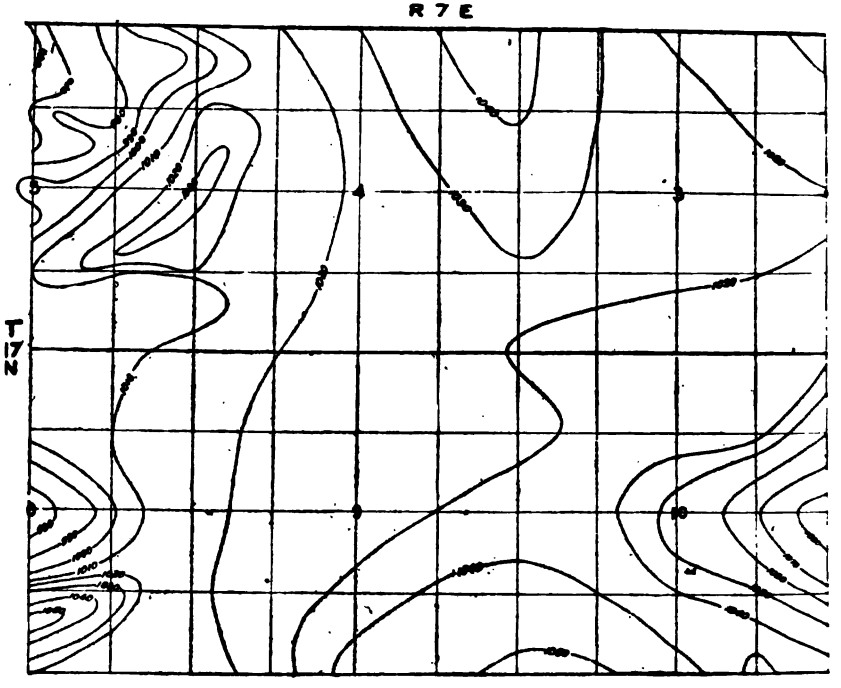

FIG. 1.—SECTION OF CUSHING FIELD. CONTOURS ON PAWHUSKA LIMESTONE.

and goes more rapidly into water. It will be seen from the figures on the north line of Sections 8 and 9, and from the cross-section (Fig. 3) taken along that line, that the Tucker sand dips almost twice as steeply as the Bartlesville sand. The latter dips 70 ft. from Well No. 12, Maley Yarhola, to Well No. 4 of the Gypsy Oil Co., ¼ mile east, while the former dips 123 ft. in the same distance. The difference is entirely due to the thickening of the shale interval between the two sands.

The producing area of the Tucker sand is, therefore, only a little over ¼ mile across. On top of this narrow anticline, however, was drilled the biggest producer of light oil known to the writer, namely, the No. 11 Jackson Barnett of the Gypsy Oil Co. of Oklahoma, which came in at 14,000 bbl. per day.

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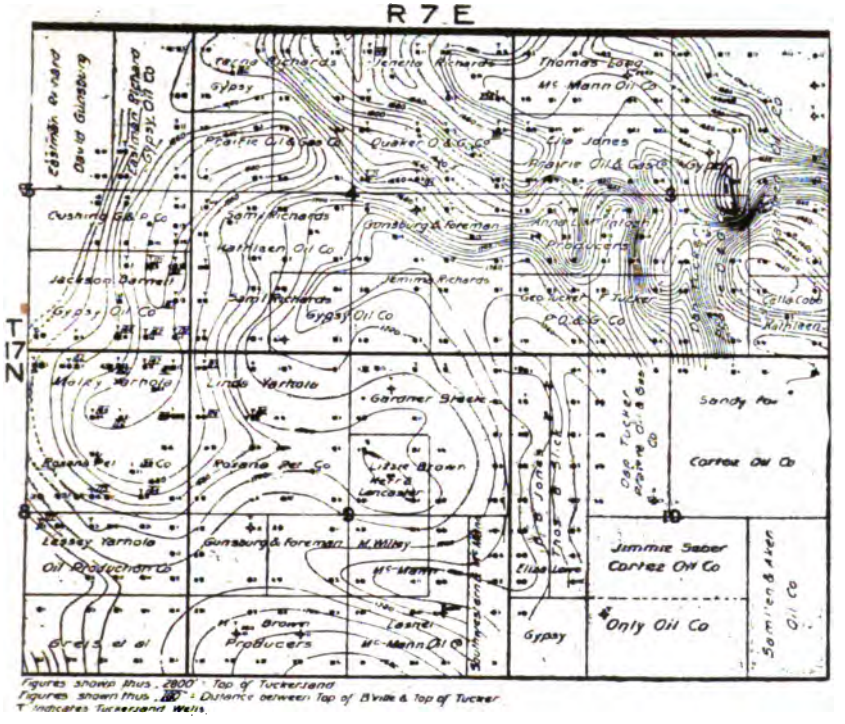

FIG. 2.—SECTION OF CUSHING FIELD. CONTOURS ON BARTLESVILLE SAND.

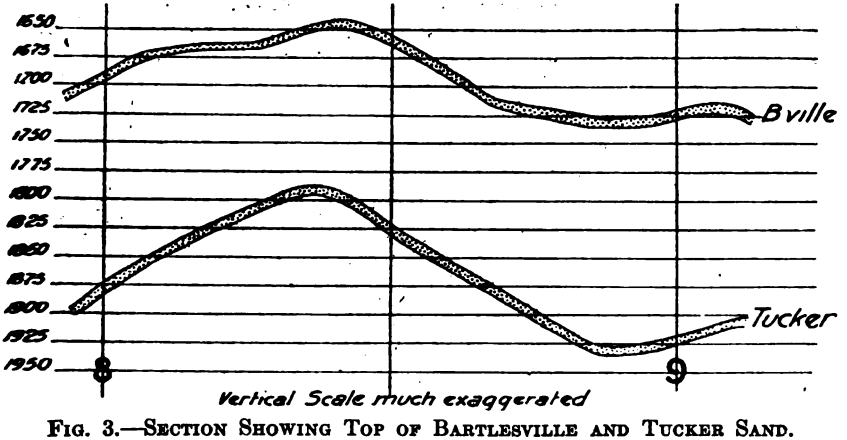

FIG. 3.—SECTION SHOWING TOP OF BARTLESVILLE AND TUCKER SAND.

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After a few wells had been drilled to this sand, the exact limit of production was figured out, and our company is drilling accordingly. By taking this thickening into account, millions of dollars were made for our company on purchasing of land.

It will be seen that the same is true of the other domes on the maps. Of course these details cannot be told from the surface, but if one recognizes the conditions that these facts set forth, one need not go far astray when estimating the possible deeper-sand production in a developed field.

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offered thereafter should preferably be in the form of a new paper.

The Genesis of Asbestos and Asbestiform Minerals*

BY STEPHEN TABER, PH. D.,† COLUMBIA, S. C.

(New York Meeting, February, 1917)

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Introduction

The term asbestos, as commonly used, includes half a dozen minerals ll having a well-developed fibrous structure, but differing in chemical mposition and in some of their physical properties. In its strict pplication the name is limited to the fibrous varieties of the monoclinic nphiboles. Commercially, however, the most important of the asbesform minerals is chrysotile, a fibrous variety of serpentine. About 95. er cent. of the asbestos used in manufacturing is chrysotile, and it mmands a much higher price than any of the other fibrous minerals w on the market.

Although the production of asbestos has increased rapidly in recent ars, comparatively little has been published concerning its origin. ie present paper is preliminary in its nature, and therefore does not etend to exhaust the subject. The ideas herein developed are the result field investigation, laboratory experiments in the growth of fibrous

* Manuscript received by the Secretary on June 26, 1916.

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crystals, and the examination of asbestos specimens from the more important producing districts, as well as a careful study of the work of previous investigators.

Asbestiform Minerals

Chemical and Mineralogical Relations

The table given below contains the group name and species of the various fibrous minerals to which the term asbestos is commonly applied. The name of each mineral is accompanied by its chemical formula as given in Dana's *System of Mineralogy*.

Asbestiform Minerals
Amphibole Group
Orthorhombic Section
Anthophyllite (Mg,Fe) SiO_3
Monoclinic Section
Tremolite $CaMg_3(SiO_3)_4$
Actinolite $Ca(Mg,Fe)_3(SiO_3)_4$
Crocidolite $NaFe(SiO_3)_2.FeSiO_3$
Serpentine $H_4Mg_3Si_2O_9$
Chrysotile
Picrolite

All of the minerals listed in the table have both fibrous and non-fibrous varieties. The fibrous varieties never show the outward form of crystals except as pseudomorphs, but the fact that they are crystalline is proved by their optical properties and other evidences of regular internal molecular structure. Well-developed euhedral crystals of tremolite and actinolite are quite common, especially where these minerals have developed in metamorphic limestones or have grown in cavities. Euhedral crystals of anthophyllite are very rare, they have never been reported for crocidolite, and serpentine has no crystal form of its own, though frequently occurring as a pseudomorph after other minerals.

Tremolite and actinolite usually assume more or less elongated prismatic forms. Often the crystals occur in aggregates of parallel prisms or in groups having a radial arrangement. There is every gradation in structure from the normal prismatic forms through the slender and columnar into the fibrous. The habit of anthophyllite is similar, except that radiating groups are relatively more abundant and the structure is more inclined to be fibrous. Crocidolite is usually fibrous but is sometimes massive or earthy.[1] Serpentine is generally massive in appearance, frequently fibrous, and sometimes foliated. Examined under the micro-

[1] E. S. Dana: *A System of Mineralogy*, Ed. 6, p. 400 (1914).

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scope, even the massive varieties commonly show a more or less finely fibrous structure, though occasionally the serpentine is in the form of minute scales. The amphiboles possess a well-developed prismatic cleavage intersecting at angles of 54° to 56°, while the chrysotile variety of serpentine is said to have a prismatic cleavage of 50°.

In chemical composition these minerals are characterized by the absence of aluminum and by the presence of magnesium in all except crocidolite, which contains instead a notable percentage of sodium. Tremolite and actinolite are distinguished by the presence of calcium as an essential element. The fibrous amphiboles are all normal metasilicates, and serpentine is a hydrous orthosilicate. These minerals are all secondary, being formed from other minerals, and for the most part they are confined to metamorphic rocks, which have had in some instances an igneous and in others a sedimentary origin.

Physical Properties

The commercial value of asbestos and the uses to which the different varieties may be put are dependent upon the physical properties; fineness, length and flexibility of fiber, tensile strength, and heat- and acid-resisting properties.

All of the different varieties of asbestos may be split up into exceedingly fine fibers, and when the finest obtainable fibers, having a diameter of 0.002 mm. or less, are examined under the microscope, they are usually seen to be made up of still smaller fibers. The number and fineness of the smallest fibers distinguishable under the microscope increase with every increase in the power of magnification, and there is no apparent limit to this subdivision.

Bunches of amphibole asbestos, of the slip-fiber type, are sometimes found with a length of 2 or 3 ft., but when these bunches are split up into smaller and smaller bundles, the length of the bundles tends to decrease with their diameter. This indicates that the long bundles probably consist of shorter interlaced and overlapping bundles of fibers. In chrysotile veins the fibers seldom have a length of over 2 or 3 in., and the bulk of the material mined averages less than ½ in. The longer fibers are commonly cut by planes of jointing, sometimes almost invisible, and sometimes marked by the presence of foreign material.

Chrysotile fibers are the most flexible, but fibers of crocidolite are almost if not equally as good in this respect. The other varieties of amphibole asbestos are, on the whole, less flexible, and, while this property varies greatly in each of the different species, the fibers of anthophyllite seem to be somewhat more flexible than similar fibers of tremolite or actinolite. Picrolite is similar to chrysotile except that the fibers are coarser and more brittle. The latter mineral is of no commercial im-

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portance. Little is known concerning the tensile strength of the different species of asbestos, but the experiments of Haussman[2] indicate that crocidolite has the greatest tensile strength, and chrysotile probably ranks second.

Chrysotile easily withstands temperatures of 2,000° to 3,000° F., while with some varieties a temperature of 5,000° F., produces no visible effect.[3] At red heat it gives up water and becomes brittle. Anthophyllite under the same conditions remains practically unaltered. Tremolite and actinolite fuse at somewhat lower temperatures, while crocidolite fuses so easily that it is useless for many purposes where asbestos is commonly employed.

Chrysotile is readily attacked by relatively weak acids, being decomposed with the separation of silica; the amphibole varieties, especially tremolite and anthophyllite, are very resistant even when subjected to the action of concentrated acids.

Types of Asbestos

Three types of asbestos fiber are recognized—cross-fiber, slip-fiber and mass-fiber. Cross-fiber asbestos occurs in veins with the fibers extending transverse to the strike of the vein. Usually the fibers are approximately perpendicular to the inclosing walls, frequently they are more or less oblique and occasionally they are curved or abruptly bent. Slip-fiber is found along fault planes, often accompanied by slickensides, and the direction of the parallel fibers records the direction of displacement. The amount of displacement is usually small. Slip-fiber asbestos is commonly distributed in thin layers that are not as a rule continuous for any considerable distance, but occasionally it is found in masses a foot or more in thickness. All gradations are to be found between the cross-fiber and slip-fiber types. Mass-fiber asbestos occurs in fibrous bundles or groups varying in size and orientation. The fibers may be parallel but are usually divergent and often radiating. In the type occurrence at Sall Mountain, Georgia, mass-fiber asbestos makes up practically the entire rock mass. Gradations between the mass-fiber and slip-fiber types probably exist, although, so far as the writer is aware, none has been described.

Anthophyllite has been reported as occurring in all three ways, and it is the only variety of asbestos known to occur as mass-fiber. The asbestiform varieties of tremolite and actinolite are practically limited to the slip-fiber type although two occurrences of true asbestos in the

[2] J. F. L. Haussman: *Handbuch der Mineralogie*, p. 734 (1847).

[3] Fritz Cirkel: Chrysotile-Asbestos, Its Occurrence, Exploitation, Milling and Uses, Ed. 2., *Canada Department of Mines, Mines Branch, Report* No. 69, p. 30 (1910).

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form of cross-fiber have been described.[4] Crocidolite has been reported only in cross-fiber veins, but there is every reason for believing that it may be found also as slip-fiber. Very little is known as yet concerning the occurrence of crocidolite. Most of the commercially important deposits of chrysotile are of the cross-fiber type, but slip-fiber is also very common.

Origin of the Fibrous Structure

Views of Previous Investigators

The wonderfully fibrous structure of the asbestos minerals has aroused interest and curiosity ever since they were first discovered, and yet little by way of explanation of this peculiar structure is to be found in geologic literature. Why these minerals sometimes possess an asbestiform structure and sometimes do not is a question on which few geologists have advanced an opinion.

Merrill[5] expresses the belief that the fibrous structure in the case of anthophyllite "and of the true asbestos as well, is due, in many instances at least, to a process of shearing—is, in fact, an exaggerated form of the process of uralitization."

Heddle[6] and Merrill[7] have described instances in which the fibrous structure of anthophyllite has been accentuated by weathering. Hopkins[8] apparently adopts this view in part for the deposits at Sall Mountain, Georgia.

Heddle[9] argued that chrysotile must be pseudomorphic, since serpentine is a mineral incapable of assuming a chrystalline form, but, after describing the chrysotile found near "Hesta Ness, which terminates the south side of the bay of Gruting, in Fetlar, Shetland," he suggests that "As the magnetite here is itself saturated with a serpentinous basis, there is a possibility that the fibrous structure of the chrysotile may be the result of its protrusion, by an exfiltration process, through the interstices of the granular magnetite, which interstices acted upon the mineral in a manner similar to the holes in a draw-plate."

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[4] M. F. Heddle: Chapters on the Mineralogy of Scotland, *Transactions of the Royal Society of Edinburgh*, vol. 28, pp. 503–504 and 531 (1877–78). Also quoted by G. P. Merrill in Notes on Asbestos and Asbestiform Minerals, *Proceedings of the U. S. National Museum*, vol. 18, pp. 286–287 (1895).

[5] G. P. Merrill: Notes on Asbestos and Asbestiform Minerals, *Proceedings of the U. S. National Museum*, vol. 18, p. 289 (1895).

[6] M. F. Heddle: *Op. cit.*, pp. 502 and 504.

[7] *Op. cit.*, p. 288.

[8] O. B. Hopkins: Report on the Asbestos, Talc and Soapstone Deposits of Georgia. *Geological Survey of Georgia, Bulletin* No. 29, p. 106 (1914).

[9] M. F. Heddle: *Op. cit.*, pp. 535–536.

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Dana,[10] in his *Systematic Mineralogy*, gave the following explanation of the origin of fibrous structure: "When a solution is spread thinly over a large surface, minute crystalline points encrust the whole, and if the solution be gradually supplied, as crystallization goes on, it is obvious that the minute points may elongate into crowded prisms of fibres, producing a fibrous structure. Such a structure is common in narrow seams in rocks, and the fibres are usually elongated across the seam."

Cirkel[11] quotes this explanation and then adds: "It seems that this paragraph refers to chrysotile; because the crystals of the same are elongated across the seam in the precise mode described." Dana's hypothesis is probably the correct explanation of the fibrous structure seen in the stalactitic, mammillary, and similar forms that have been deposited in open spaces from a thin layer of solution, but there is no evidence that any of the varieties of asbestos have been formed in this way.

The hypothesis advanced by Cirkel[12] is that, "chrysotile may have been an extreme example of crystallization which took place under conditions of high temperature and extreme pressure."

While the writer has not had the opportunity of making an exhaustive search of the literature, he believes that the various explanations of fibrous structure listed above include the more important hypotheses so far advanced by scientific investigators.

Discussion of the Evidence

Microscopic examination shows that bundles consisting of thousands of small fibers of chrysotile or amphibole asbestos behave as crystal units, an entire bundle exhibiting the same optical properties as any one of the component fibers. The fact that all of the asbestiform minerals have prismatic cleavage suggests that the fibrous structure may be due to an abnormal development of the cleavage or at least that the separation of the fibers takes place along cleavage planes. This hypothesis is supported by the prismatic form shown by most of the smallest obtainable fibers of amphibole asbestos, when they are examined under the microscope, but fibers of chrysotile usually possess irregular polygonal or rounded cross-sections. The highly fibrous structure of the asbestiform minerals is not, however, a crystallization phenomenon in the sense that it is due solely to the inherent physical properties of the crystal molecule, for all minerals having asbestiform varieties occur in non-fibrous as well as fibrous forms. Moreover, there are intermediate gradations between

[10] J. D. Dana: *Systematic Mineralogy: Section II, Theoretical Crystallogeny*, p. 124 (1850).

[11] Fritz Cirkel: *Op. cit.*, p. 93.

[12] *Loc. cit.*, p. 93.

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the non-fibrous and the asbestiform varieties of serpentine and of the amphiboles with the possible exception of crocidolite.

Why is it, then, that these minerals sometimes have a fibrous or asbestiform structure, while at other times they are non-fibrous? The answer to this question, the writer believes, is to be found chiefly in the physical conditions under which the minerals were formed.

The primary minerals of igneous rocks never show an asbestiform structure even in the case of minerals possessing perfect prismatic cleavage, and this statement is true in general for all minerals that have grown in free contact with supersaturated solutions. The latter generalization will be objected to by those who believe that cross-fiber veins have been deposited from solutions circulating along open fractures, but the writer has found no evidence supporting such an hypothesis. This question, however, is later discussed in detail under the origin of cross-fiber veins.

There are many minerals that have fibrous varieties. Most of them do not have a prismatic cleavage, some have no cleavage at all, and at least one such mineral, limonite, is always amorphous and never crystalline. The fibrous varieties of these minerals differ from asbestos chiefly in having fibers that are coarser, more brittle and not so easily separable. With fibers of equal size the flexibility and tensile strength are probably determined very largely by chemical composition.

In the case of those minerals that do not have a prismatic cleavage and do not normally have a columnar or prismatic habit, the fibrous structure is obviously due to physical conditions which have prevented crystal growth except in one direction. If a mineral having perfect prismatic habit and cleavage develops under physical conditions that limit growth to a direction parallel to the principal axis, then the fibrous structure may be accentuated to such an extent as to make the mineral truly asbestiform. A careful comparison of the occurrences of all the common fibrous minerals indicates that the commercially important deposits of the asbestiform minerals have been formed under the same physical conditions that result in the development of a well-defined fibrous structure in other minerals.

That the peculiar structure of asbestiform minerals is usually due to the accentuation of a normal prismatic habit and cleavage through the limitation of crystal growth by physical conditions is the author's thesis. Recently he conducted a series of laboratory experiments with the object of determining the essential conditions for the development of fibrous crystals. Some of these experiments have been referred to in another paper,[13] and it is planned to publish a detailed account of the others in the near future. All questions connected with the problem have not been

[13] Stephen Taber: The Growth of Crystals under External Pressure, *American Journal of Science*, Ser. 4, vol. 41, pp. 532–556 (1916).

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cleared up, but some of the facts established have an important bearing on the present discussion.

Conclusions

The shape of a growing crystal is controlled by one or more of three independent factors, namely: (1) the tendency to assume a regular polyhedral form because of the forces of surface tension and molecular orientation; (2) the relative and absolute magnitude of the external forces resisting growth in different directions; and (3) the accessibility of the material from which the crystal is built.

1. Some crystalline substances normally have a prismatic or columnar habit because of the intermolecular forces controlling the development of crystal faces, and under favorable conditions slender acicular or hair-like crystals may result; but a highly fibrous or asbestiform structure is never developed even in the case of minerals having perfect prismatic cleavage, without the assistance of one of the other two factors.

2. If crystals are under unequal pressure, growth may be limited to the direction of least pressure. When crystals grow through the addition of new material from solutions, growth takes place only where the solutions are supersaturated with respect to the crystal surfaces with which they are in contact, and growth must continue regardless of resisting forces as long as supersaturation is maintained. But pressure increases the solubility of all substances that go into solution with decrease in volume, and therefore any increase in pressure must be accompanied by a corresponding increase in the concentration of the solution, if growth is to continue. Crystals subjected to unequal pressure may even go into solution on the surfaces that are under the greater pressure, while at the same time deposition is taking place along the line of least pressure; and if the normal habit of the crystals is columnar, then those crystals that are oriented with their longer axes parallel to the least pressure will tend to grow at the expense of those that are less favorably oriented.

In the same way unequal pressure may control the direction of growth and therefore the shape of crystals, when crystallization from a state of fusion takes place, with increase in volume. There are, however, only a few substances, such as ice, that solidify with expansion in volume. Unequal pressure may also determine the shape of secondary minerals formed without going into solution, for, when the alteration of one mineral to another is accompanied by increase in volume, pressure tends to prevent the alteration, and therefore the effect of unequal pressure may be to limit alteration and the growth of the new mineral to the direction of least pressure.

Unequal pressure always sets up shearing stresses, and in the case of cleavable minerals these stresses are relieved most easily by slipping

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along cleavage planes, thus making the cleavage more pronounced. The effect of shearing on minerals with prismatic cleavage may therefore be a factor of some importance in producing a fibrous structure.

The development of fibrous structure is undoubtedly in some instances to be attributed to growth under unequal pressure, but this is not the commonest cause of the phenomenon.

3. The study of fibrous minerals and their occurrence in nature, as well as the results of laboratory experiments in the growth of fibrous crystals, leads to the conclusion that in most cases the development of fibrous structure has been due primarily to the fact that the material from which the growing crystals were built was accessible only in one direction. This is the most probable reason why secondary minerals are so frequently fibrous. For example, when serpentine is formed from olivine, the alteration sets in from the exterior of the crystal and from cracks, and the resulting serpentine is usually in the form of microscopic fibers that develop normal to these surfaces. The fibrous form is gradually assumed by the microscopic crystals, because growth takes place only at their base where they may receive additions of new material as alteration of the olivine progresses. In a partly altered olivine the microscopic veinlets of serpentine are frequently similar in appearance and structure to the larger veins of cross-fiber chrysotile.

The efficacy of this method of producing fibrous structure in crystals, when growth takes place through the addition of new material from supersaturated solutions, has been proved experimentally.[14] Cups of porous porcelain were partly immersed in concentrated solutions of copper sulphate, alum and other salts. The solutions were drawn up through the capillary pores allowing evaporation to take place from the exposed surfaces. After a day or two, crystal growth began with the formation of irregular spots or thin crusts on the upper surfaces of the cups, and these gradually increased in size and thickness. Later, groups of fibrous crystals could be observed under the crusts slowly pushing them outward. The fibers continually increased in length as long as the material for growth was available, and at the end of 8 months some were over 2 cm. in length.

In these experiments crystallization also occurred at a few favorable places within the walls of the cups, and, as growth continued, the crystals developed sufficient pressure to produce rupture. The fractures thus formed were gradually extended and widened by the growth of fibrous veins, closely resembling in structure the cross-fiber veins of chrysotile and of asbestos, as well as the similar veins of fibrous calcite, gypsum and other minerals. In some of the experiments where alum was used, it was possible to change the color of the solution by adding chrome alum in varying amounts, and thus produce banded veins.

[14] Stephen Taber: *Op. cit.*, pp. 545–546.

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Cavities or open spaces were usually present under the central portions of the larger crusts and within the veins, and these openings were sometimes partly lined with crystals normal in habit instead of fibrous. This lack of uniformity is to be explained by the fact that supersaturation of the solution was produced and maintained by evaporation, and this process was retarded or prevented where the crusts protected the solution from contact with the air. Better results were obtained when supersaturation was induced by cooling, so that practically uniform concentration could be obtained over the entire crystallizing surface.

The experiments so far conducted by the writer indicate that fibrous crystals may be produced in the manner described above only from substances that go into solution with decrease in volume.[15] Most, if not all, of the rock-forming minerals belong to this class.

The development of the fibrous structure is probably aided by small adjustments or slips along the surfaces of the fiber prisms and cleavage planes when the latter are present, since under the conditions of growth it is not conceivable that new material can be added at the end of each prism of a large group continuously and at the same uniform rate. Indeed observation shows that such slipping does take place in the groups of crystals grown in the laboratory, and both megascopic and microscopic examination of fibrous minerals furnishes evidence of similar slipping.

The fibers obtained in the experiments just described are brittle and therefore rather difficultly separable, but occasionally it is possible to procure individual fibers having a length of several millimeters and a thickness of less than 0.001 mm. Such fibers of copper sulphate and alum are flexible and somewhat elastic. The columnar or fibrous ice crystals found in clayey soils grow in practically the same way as the fibrous crystals described above, and Merrill has observed the growth of fibrous incrustations of gypsum on the walls of caves,[16] but did not attribute their fibrous structure to the manner of growth.

Origin of Cross-fiber Veins

Many different theories, some of them rather hypothetical, have been advanced by geologists to explain the origin of veins of cross-fiber asbestos. These theories have been reviewed recently in three publica-

[15] Attempts were made to obtain similar results with ammonium chloride and other salts that go into solution with expansion in volume, but no veins were formed and the crusts were enlarged only through the addition of new material on the outer exposed surfaces, the solutions reaching these surfaces through capillary pores in the crystalline mass. In some instances tubular, hair-like crystals were formed, 0.01 mm. or less in diameter, and these apparently grew only at their outer ends, the new material being furnished by solutions drawn up through the capillary tubes.

[16] G. P. Merrill: On the Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings of the U. S. National Museum*, vol. 17, pp. 77–81 (1894).

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tions, and it is not necessary, therefore, to give them in detail here. That no general agreement has been reached, is shown by the diversity of opinion expressed in the conclusions drawn by the authors of the three monographs just cited.[17] Most of the theories are limited to chrysotile veins in serpentine and could not be applied to veins of crocidolite and other varieties of amphibole asbestos. Since the origin of chrysotile is closely related to the origin of the massive serpentine with which it is associated, the origin of chrysotile veins will be considered separately.

Chrysotile Veins

All of the different theories attempting to explain the origin of chrysotile veins fall under one or the other of the following classes:

1. The veins were deposited in open fissures: (*a*) by circulating solutions; or (*b*) through the segregation of material; or (*c*) infiltration of serpentinous solutions from the walls.

2. The veins were formed by the replacement of the wall rock along small fissures that served as channels for the circulation of solution.

3. The veins are portions of the serpentine that have crystallized *in situ*, the crystals growing outward from preëxisting fractures through which water entered to alter the rock to serpentine.

All of these theories presuppose the presence of fractures, and much ingenuity has been used in accounting for the fractures. They have been attributed to: (1) the contraction of an igneous magma upon cooling and solidifying; (2) dynamic causes; (3) exfoliation, possibly aided by the increase in volume that accompanies the alteration of peridotite to serpentine; (4) shrinkage resulting from the loss of silica or other constituent; and (5) a partial dehydration of the serpentine.

It seems to the writer that there are serious objections to all of the theories outlined above, for, while some of them may explain many of the phenomena connected with the occurrence of asbestos veins, not one offers a complete and adequate explanation of all of the known facts. It is conceivable that some chrysotile veins may have been formed in open fissures, but it is mechanically impossible that all of them could have been formed in this way. Dresser[18] has pointed out the absurdity of this theory as applied to the chrysotile deposits of southern Quebec, Canada, where the veins, in places occupying over 10 per cent. of the entire rock, run in all directions from vertical to horizontal, and occasionally reach

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[17] Fritz Cirkel: *Op. cit.*

J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey, Memoir* No. 22 (1913).

O. B. Hopkins: *Op. cit.*

[18] J. A. Dresser: *Op. cit.*, p. 65.

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a length as great as 100 ft. Pratt[19] and Diller[20] have described chrysotile veins in the Grand Canyon, Arizona, over 4,000 ft. below the rim, which extend horizontally parallel to the bedding of the inclosing rocks for distances of 150 ft. or more.

Moreover, there is evidence, in many instances at least, that the formation of the chrysotile veins and the alteration of the inclosing rock to form serpentine are contemporaneous processes. In the Black Lake district of Quebec, where the alteration of peridotite to serpentine is not complete, the chrysotile veins are bordered on both sides by bands of massive serpentine, and the width of the veins is proportional to the width of the inclosing bands. By careful measurements Dresser[21] found the ratio of the chrysotile vein to the entire band of serpentine to be 1:6.6. The alteration of peridotite to serpentine is accompanied by such an increase in the volume of the rock that it would be impossible for fissures to remain open during the process.

Little if any evidence has been offered in support of the replacement hypothesis. Where chrysotile veins occur in limestone, as they do in Arizona, it is possible for some replacement of the inclosing rock to take place, but this occurrence is exceptional. Nearly all chrysotile veins are found in massive serpentine, having a chemical composition that is practically identical with that of the vein material, and no reason has yet been given to explain why serpentine should replace serpentine of the same chemical composition. Moreover, chrysotile veins do not possess any of the characteristics that commonly distinguish replacement veins from other veins. They have their own peculiar structure and never show any trace of an inherited structure. Serpentine frequently occurs as a pseudomorph after other minerals, but pseudomorphs are never found in chrysotile veins. Replacement veins are characterized by great irregularity in width and lack of sharp boundaries, while chrysotile veins frequently show remarkable uniformity in width, and always have well-defined walls.

The theory that "the veins are crystallized portions of the serpentine walls, and that the crystals (fibers) have grown outward from the original crevices which are now represented by partings of iron ore found near the center of the veins,"[22] was evidently advanced in order to avoid some of the more obvious objections to the preceding hypothesis, but it completely fails to explain many of the characteristic phenomena of chrysotile veins. If the fibers grew outward in the way here postulated, some of

[19] J. H. Pratt: Asbestos, *Mineral Resources*, 1904, *U. S. Geological Survey*, pp. 1137–1140 (1905).

[20] J. S. Diller: Asbestos, *Mineral Resources*, 1907, *U. S. Geological Survey*, pt. II, pp. 720–721 (1908).

[21] J. A. Dresser: *Op. cit.*, pp. 59–60.

[22] J. A. Dresser: *Op. cit.*, p. 66.

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them would unquestionably penetrate the massive serpentine for greater distances than others, thus giving irregular boundaries to the veins, but chrysotile veins have sharply defined boundaries and are easily separated from the wall rock.

In discussing the deposits of southern Quebec, Dresser states: "the facts are self-evident that zones of the country rock have been altered to serpentine and proportionate parts of these have taken the form of asbestos veins."[23] But, if the veins are merely "portions of the serpentinized bands which have crystallized *in situ*"[24] why is the ratio of chrysotile to massive serpentine limited? Why are the veins limited to a width of 1, 2 or, in rare instances, 3 in.?

This theory does not satisfactorily explain the angular inclusions of massive serpentine that frequently mark the central parting of chrysotile veins and are also to be found irregularly distributed through them. It affords no explanation of the occasional presence of more than one parting or of bends in the fibers such as are shown in Figs. 4, 5 and 6. It does not explain the gradation of cross-fiber into slip-fiber as illustrated in Figs. 3 and 7. Most of the objections here given are equally applicable to the other theories of vein formation so far discussed, and, in view of all the facts, the conclusion is inevitable that none of these theories furnishes a true explanation of the origin of chrysotile veins.

As already noted in this paper, fibrous veins with structural features analogous to those of chrysotile veins have been produced in the laboratory, where their formation and growth could be carefully observed. The results obtained from these experiments, supplemented by a study of fibrous veins in different kinds of rock, have led the writer to certain definite conclusions. These conclusions have been formulated into a theory of vein formation that seems to explain satisfactorily the phenomena connected with chrysotile veins. Briefly stated, this theory is that all cross-fiber veins are formed through a process of lateral secretion, the growing veins making room for themselves by pushing apart the inclosing walls; and that the fibrous structure is to be attributed largely to the physical conditions which have limited crystal growth to a single direction. In the case of the asbestiform minerals, the fibrous structure is accentuated by a normal prismatic habit and cleavage.

In individual occurrences it may be difficult or impossible to determine why the fibrous mineral was taken into solution, the cause of its redeposition and the other details of origin; but, in the case of serpentine, pressure due to expansion in volume is probably the controlling factor.

Serpentine is a secondary mineral resulting, as a rule, from the altera-

[23] J. A. Dresser: Asbestos in Southern Quebec, *Trans.*, vol. 50, p. 957 (1915).

[24] J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey, Memoir* No. 22, p. 65 (1913).

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tion of preëxisting silicates of magnesia. The alteration is seldom one of simple hydration, for usually some material is added or subtracted, and in many cases the constituents out of which serpentine is formed are derived from several different minerals. Serpentine is therefore found in rocks that are either of sedimentary or of igneous origin. Locally it may be the dominant mineral thus forming serpentine rocks. Such rocks are derived chiefly from igneous rocks of the pyroxenite-peridotite family which contain a high magnesian content. Although serpentine is secondary after many minerals, the most important source is olivine followed by the magnesian pyroxenes and amphiboles.

The alteration to serpentine is accompanied by an increase in the volume of the rock mass, which results in the development of pressure whenever there is resistance to expansion. This pressure can not be explained, however, by attributing it to a chemical reaction taking place with increase of volume, as, for example, when plaster of Paris combines with water and sets, because, in many cases where serpentine is formed from anhydrous minerals, there is an actual decrease in the total volume of the system, if the volume of the water is taken into consideration. For instance, when serpentine is formed from olivine according to the equation:

$$3Mg_2FeSi_2O_8 + 6H_2O + O = 3H_4Mg_3Si_2O_9 + Fe_3O_4.$$

Neglecting the oxygen, the volume changes in this reaction are as follows: 281.3 parts of olivine combine with 108 parts of water to give 321 parts of serpentine and 44.5 parts of magnetite. Hence there is an increase in the volume of solid amounting to approximately 30 per cent., while there is a decrease of about 6 per cent. in the volume of the system as a whole. The pressure resulting from this reaction is, therefore, due to the fact that water in a liquid or gaseous state is able to penetrate the rock mass through capillary and subcapillary openings while the solid serpentine formed does not escape in like manner.

The water may penetrate the rock through fractures, along the contact between mineral grains and along the cleavages of minerals. As the alteration proceeds, readjustments in the rock mass, due to increase in volume, result in the formation of new fractures. Some of the water possibly travels for short distances through a process of diffusion or trading, by which a molecule of serpentine gives up its water to a neighboring anhydrous molecule and then takes up more water.

The maximum pressure that may be developed locally as a result of the alteration must be enormous, since, in most cases, it is limited solely by the resistance offered to expansion. Pressure alone can have no direct effect in preventing a reaction that takes place with decrease in volume; and, whether the reaction will take place or not is determined entirely

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by the partial pressure of the water vapor and the vapor pressure of the serpentine.

The development of pressure in this way is nicely illustrated by the following laboratory experiment:

A porous porcelain cup was partly filled with anhydrous cupric chloride ($CuCl_2$), after which the open end was sealed with paraffine. The cup was then placed, together with a beaker containing water, under a bell jar, and allowed to stand undisturbed. Water vapor was absorbed through the capillary pores of the cup with the formation of the hydrous salt ($CuCl_2.2H_2O$). At the end of 5 days small fractures were observed, and these gradually increased in length and width as the volume of the salt continued to increase. The fracturing of bottles by salts that take up water of crystallization from the atmosphere is a common phenomenon in chemical laboratories.

The pressure developed as a result of the increase in rock volume when serpentine is formed is important as a factor in the formation of chrysotile veins, chiefly because the solution of the serpentine and its redeposition in the form of chrysotile seem to be determined very largely by pressure. The pressure is at a maximum where serpentine is being formed, and here the mineral is most readily taken into solution. The separation of serpentine from solution is accompanied by expansion in volume, and therefore takes place only where the forces opposing expansion are not prohibitive.

All existing fractures or joints, whether dynamic in origin or due to contraction on cooling, are favorable places for the deposition of chrysotile, because as a rule less force is required in pushing apart the walls of existing fractures than is necessary in forming new ones. The alteration of a rock to serpentine begins along the existing fractures which divide the rock mass into blocks of variable size. As the alteration slowly penetrates inward from the surfaces of a block, strains are set up between the expanded outer shell and the unaltered central part. These strains tend to produce exfoliation fractures, and, although the resisting pressure is probably sufficient in most cases to prevent the formation of open fissures, this pressure is locally reduced by the tendency to fracture, thus permitting the separation of serpentine from solution and the growth of new veinlets of chrysotile.

The chrysotile deposits of Thetford, Canada, furnish good illustrations of veins that have originated in this way,[25] and Merrill has described veins in the serpentine at Montville, N. J., that are difficult of explanation under any other hypothesis. At the latter locality the serpentine has been formed through the hydration of nodules of lime-magnesian

[25] J. A. Dresser: Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey Memoir*, No. 22, pp. 58–59 and Fig. 7 (1913).

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pyroxene occurring in dolomite, and the narrow chrysotile veins are in a general way parallel to the surface of the original nodule of pyroxene.[26]

Since the pressure due to expansion prevents the formation or maintenance of appreciable open spaces, water must reach the unaltered rock, at least partly, through very small capillary and subcapillary openings. Under these conditions circulation is extremely slow and the movement is probably limited almost entirely to a single direction, *i.e.*, toward the unaltered portion of the rock. It is not necessary to assume the presence of solutions circulating toward the chrysotile veins in order to explain their growth, as the transfer of serpentine through the short distance from the place of its formation to the walls of the growing vein is probably due chiefly to diffusion of the material through the solution.

If pressure due to expansion is the principal factor in bringing about solution of the serpentine, there must be a close relation between the increase in volume of the rock mass and the volume of material removed in solution and redeposited as chrysotile. This probably explains why the percentage of cross-fiber chrysotile in serpentine is less than the increased volume of the rock, and why the width of chrysotile veins in partly altered rock sometimes bears a definite ratio to the thickness of the inclosing bands of massive serpentine. When the rock is homogeneous and the alteration proceeds uniformly, a definite and limited proportion of the resulting serpentine is removed in solution, for the local pressure is relieved by the transfer of this excess material.

When the formation of serpentine is accompanied by little or no increase in the volume of the rock because of the complete removal of a portion of the products, no chrysotile may be formed. It is possible that in individual cases the resistance to expansion and various other factors prevent the formation of chrysotile veins.

The difference in the specific gravities of massive serpentine and chrysotile is unquestionably due to the difference in their modes of origin. According to Dana[27] the specific gravity of massive serpentine varies from 2.50 to 2.65, while that of chrysotile is only 2.219.

While it seems probable that pressure is the controlling factor in the solution of serpentine and its redeposition as chrysotile, this is not essential to the writer's theory of the origin of cross-fiber veins. The solution of mineral matter and its redeposition in fibrous veins may result from one or more of several different causes. In the laboratory experiments referred to above, supersaturation was induced by either evaporation or cooling. The essential conditions for the growth of cross-fiber veins are: (1) the growing fibers must be in contact at their base with supersaturated solutions; and (2) that the solutions must reach the grow-

[26] G. P. Merrill: On the Serpentine of Montville, New Jersey. *Proceedings of the U. S. National Museum*, vol. 11, pp. 105–112 (1888).

[27] E. S. Dana: *System of Mineralogy*, Ed. 6, pp. 269–271 (1914).

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ing veins through the wall rock. So long as the veins are in contact with supersaturated solutions, growth will continue and the walls will be pushed apart regardless of resisting forces; but the greater the resisting pressure, the greater the concentration must be in order to produce supersaturation.

The force that enables growing veins to make room for themselves by pushing apart their inclosing walls is not due to the tendency of a crystalline substance to assume a regular polyhedral form, for the columnar or fibrous structure of most minerals occurring in cross-fiber veins is not a crystallization property, but is caused by the conditions of growth, as already explained. Under suitable conditions, the fibrous structure may develop in substances that crystallize in any of the systems of crystallization. In most cases it is not the normal habit, and therefore is unstable. The writer believes that the force is due chiefly to the expansion in volume which accompanies the separation of most solids from solution. When a substance separates from solution with increase in volume, the pressure developed depends on the magnitude of the forces resisting expansion, and may be many times the force required to crush the substance. It is altogether improbable that pressure alone could expel solutions occupying subcapillary pores in rocks, and, in serpentine and other rocks found inclosing cross-fiber veins, the openings are almost entirely subcapillary in size. As previously stated, the transfer of material is probably due to diffusion rather than to circulating solutions.

Structural Features of Cross-fiber Veins

Nearly all of the structural features characteristic of the cross-fiber veins found in rocks have been reproduced in the course of laboratory experiments. The diagrammatic sketches shown in Figs. 1 to 7 are all drawn to scale from veins of chrysotile or from veins of other fibrous minerals, but, so far as the phenomena that they illustrate are concerned, they might just as well have been drawn from some of the veins produced in the laboratory.

Cross-fiber veins commonly show a central parting or break in the continuity of the fibers (see Fig. 4), but in some veins the fibers extend from wall to wall without interruption. The central parting is formed whenever fibers start to grow from both walls of a fracture, and apparently this always happens when veins develop along preëxisting fractures, unless growth is limited to one side only. In laboratory experiments, growth is frequently limited to one side of a vein by the development of the fissure in such a direction as to cut the other side off from the solution furnishing material for growth. That this may also occur in the case of chrysotile veins is indicated by the occasional presence of the parting very close to one of the walls, instead of near the center. Since growth

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in such veins has been very largely limited to one side, it is more than probable, in some instances, that growth may be entirely confined to one side. In some of the laboratory veins, the absence of a parting is due to simultaneous growth at both ends of the fibers—a fact proved by changing the color of the solutions. This occurs only when vein growth and the inception of fracturing are contemporaneous. The absence of partings is most common in the small lenticular chrysotile veins that frequently narrow and pinch out without intersecting other veins, thus indicating that they were not formed along preëxisting joints. The absence of partings is also more noticeable where the inclosing serpentine is practically free from mineral impurities.

Occasionally two or more partings may be observed in the same chrysotile vein, and these are sometimes symmetrically arranged with respect to the center. Such partings may result from a pause in the process of growth or from a slight displacement of the walls along their contact with the vein. A parting may frequently be observed separating two stages of growth in the fibrous or needle-like ice columns found on clayey soils after a cold night, and a similar parting in fibrous gypsum has been noted by Merrill.[28]

When cross-fiber veins are examined closely, it may be seen that the partings are never plane surfaces. The specimen of chrysotile[29] sketched in Fig. 1 shows how very irregular the partings may be in individual cases. The same phenomenon can be observed on a smaller scale under the microscope, where the fibers look as though they had been pushed past one another, causing the ends to interpenetrate along the line of parting. The inequality in the length of fibers is due to the more rapid growth of those fibers that are most favorably situated for receiving additions of new material. The stresses resulting from unequal growth are relieved by slipping along cleavage planes and prism boundaries, and this tends to accentuate the development of the fibrous structure. Additional evidence of the unequal growth of fibers in chrysotile veins is furnished by the similar displacement of bands roughly paralleling the walls and marked by a slight difference in color from the rest of the vein material.

The partings in chrysotile veins are commonly marked by the presence of granular crystals of magnetite and chromite and angular inclusions of the massive serpentine wall rock, as in Figs. 2 and 7. In size, the fragments of wall rock range from the smallest grains to masses that are larger than the veins themselves. In other words, there is every gradation between branching veins and veins with inclusions of wall rock.

[28] G. P. Merrill: On the Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings U. S. National Museum*, vol. 17, p. 81 (1894).

[29] This specimen of chrysotile, from the deposits on Ash Creek near Globe, Ariz., was furnished through the courtesy of the U. S. Geological Survey.

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The inclusions are also irregularly distributed through the veins without reference to partings, and occasionally a large number of fragments may be seen strung out in a broken train that may extend from the wall to a

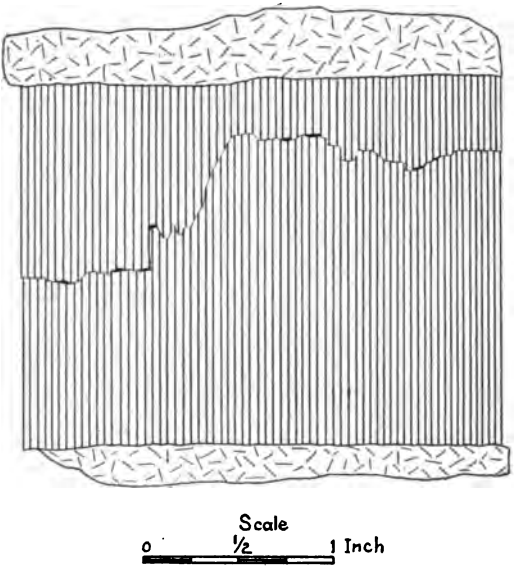

FIG. 1.—CHRYSOTILE VEIN FROM NEAR GLOBE, ARIZ. PARTING IS VERY IRREGULAR AND MARKED BY OCCASIONAL INCLUSIONS OF MASSIVE SERPENTINE WALL ROCK.

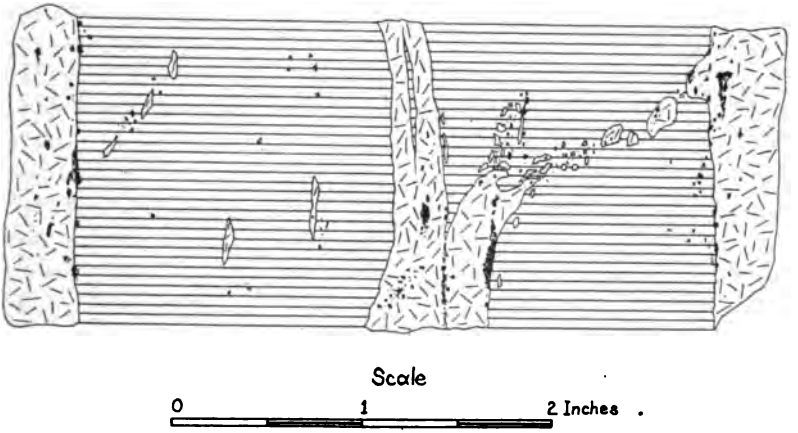

FIG. 2.—CHRYSOTILE VEIN FROM THETFORD, CANADA, CONTAINING CENTRAL INCLUSION OF MASSIVE SERPENTINE AND SMALLER INCLUSIONS OF SERPENTINE AND MAGNETITE EXTENDING IN A BROKEN TRAIN TOWARD ONE WALL. THERE IS A VEINLET OF CHRYSOTILE IN THE LARGE INCLUSION.

central parting, as in Fig. 2. Some of the larger inclusions contain veinlets of chrysotile.

Cross-fiber veins grown in the laboratory are frequently branching,

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and they also contain numerous inclusions of wall material distributed through the veins in exactly the same way as in veins of chrysotile and other fibrous minerals, such as crocidolite, gypsum and calcite. The inclusions that mark a central parting represent fragments formed when rupture occurred, and their position is due to the growth of the vein on both sides as new material was added through the walls. When growth

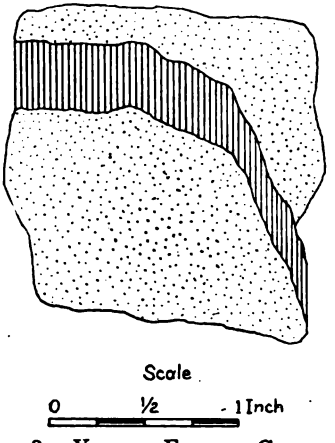

FIG. 3.—VEIN OF FIBROUS CALCITE IN LIMESTONE FROM ST. LAWRENCE COUNTY, NEW YORK.

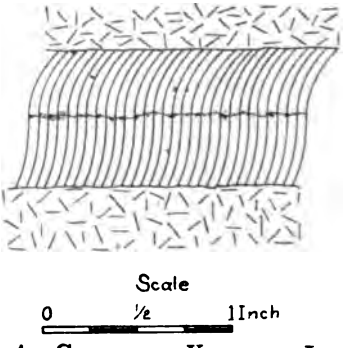

FIG. 4.—CHRYSOTILE VEIN FROM LOWELL, VERMONT, WITH CURVED FIBERS.

is more rapid on one side, the line of parting, together with the inclusions, is closer to the opposite side. Vein matter occasionally begins to crystallize out along an incipient fracture or line of weakness close to the vein, and in this way a fragment is gradually separated from the wall and included in the growing vein.

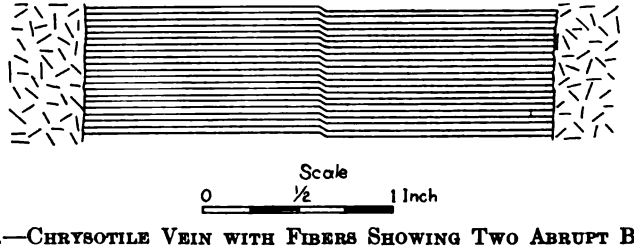

FIG. 5.—CHRYSOTILE VEIN WITH FIBERS SHOWING TWO ABRUPT BENDS.

The fibers are always parallel to one another, and, in most veins, they are approximately normal to the walls (see Fig. 2), but frequently they are more or less oblique. Laboratory experiments prove that the fibers always extend in the direction in which the walls move as they are pushed apart by vein growth. In most veins the fibers are normal to the walls because the walls are usually forced directly apart, but when the walls

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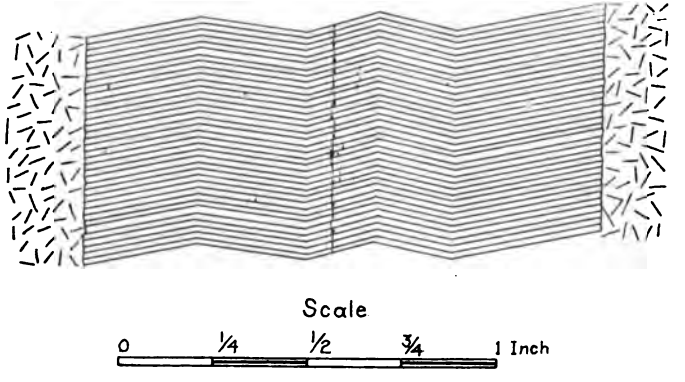
have also a lateral displacement because of the simultaneous growth of adjacent non-parallel veins or other causes, the fibers grow in the direction of the resultant motion. If the course of a vein is not straight, the fibers may be normal to the walls at one place and oblique at another, as in the calcite vein from St. Lawrence County, New York, sketched in Fig. 3. In this way cross-fiber veins may grade imperceptibly into slip-fiber veins.

As long as the relative motion of the walls of a growing vein is in a straight line the fibers are straight; any change in the direction of motion is immediately recorded by the slowly lengthening fibers. If the change in the direction of relative motion is gradual and continuous, the fibers are curved, as in Fig. 4, and if the change is abrupt, it results in the development of sharp bends. In the chrysotile vein shown in Fig. 5, the initial movement of the walls had a lateral component, which soon



IG. 6.—CHRYSOTILE VEIN WITH FIBERS SHOWING FOUR ABRUPT BENDS AND A CENTRAL PARTING.

isappeared as the vein continued to grow. The structure here suggests iat the formation of the vein fracture was due, at least in part, to shear-ig stresses. Most of the fibers in this vein extend from wall to wall ithout a break, and their tensile strength does not seem to be appreciably fected by the bends.

Sometimes the fibers of a vein record several changes in the relative ovement of the walls, and this gives a banded appearance due to the ıequal reflection of light where the fibers run in different directions. ıe specimen of chrysotile from Thetford, Canada, sketched to scale Fig. 6, is a beautiful example of this kind of banding. The beginning the vein is recorded by a parting within the central band, and there e numerous inclusions of chromite and massive serpentine distributed ong the line of parting. The movement of the walls during the first ge of vein growth is recorded by the direction of the fibers in the central nd, and during the enlargement of the vein there were two sudden

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changes in the relative motion of the walls with the formation of two additional bands on both sides of the central band, thus making five in all.

The structural phenomena discussed in the preceding paragraphs all furnish evidence tending to confirm the writer's theory of the origin of cross-fiber veins, and they are difficult or impossible of explanation under any other hypothesis of vein formation so far advanced.

Veins of Crocidolite and Other Fibrous Amphiboles

All of the asbestiform minerals belonging to the amphibole group have been reported as occurring in cross-fiber veins, though the number of such instances is small, and, as yet, little information has been published concerning the details of individual occurrences. For this reason, it is not possible at the present time to discuss, except in a general way, the factors controlling the origin, solution and redeposition of the minerals, but the data now available indicate that these veins must also have been formed through a process of lateral secretion, and that they have made room for themselves by forcing apart the inclosing wall rock. Crocidolite is the only asbestiform mineral other than chrysotile known to occur in cross-fiber veins in commercial quantities, and only one occurrence of this mineral, so far reported, is of any importance.

Crocidolite has long been known to occur in West Griqualand, South Africa, and most specimens found in mineral collections have come from that source. The deposits are located in a range of quartzose schists called the asbestos mountains. The mineral is characterized by a dull lavender-blue color, due to the presence of considerable iron protoxide. In places it is altered by oxidation and infiltration of silica to a compact siliceous stone with a bright yellow color, and chatoyant luster, which has given it the popular name of *tiger-eye*. A brief account of this occurrence is given by Cirkel[30] in his monograph on chrysotile asbestos. He states that the crocidolite "is generally found in veins, seldom less than 2 in., and more often 4 in. and 5 in. wide, formed of closely compacted parallel fibers which run from wall to wall of the vein without break or fault. Several veins have been found, regular in extent, and the fiber always lies at right angles to the sides of the deposit. The inclosing rock is a dark brown shale."

A considerable number of specimens of crocidolite from South Africa have been examined megascopically and also under the microscope, by the present writer. All of the structural phenomena characteristic of

[30] Fritz Cirkel: Crysotile-Asbestos, Its Occurrence, Exploitation, Milling, and Uses, Ed. 2, *Canada Department of Mines, Mines Branch, Report* No. 69, pp. 239–240 (1910). The quotations here given appear to have been taken from a paper by H. T. Olds, Notes on Blue Asbestos, read before the *Institution of Mining and Metallurgy*, London, *Transactions*, vol. 7, pp. 122–123 (1899).

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chrysotile veins may be found in veins of crocidolite. Specimens in the collections of the University of South Carolina show that the vein fibers are sometimes oblique instead of perpendicular to the walls, partings are frequently present and may be marked by inclusions of hematite, magnetite, quartz and fine-grained wall rock; bent and curved fibers are also common. Most of these features may likewise be observed in specimens of tiger-eye. In view of all the facts stated above, it must be concluded that, although consisting of different material, the crocidolite veins have grown in the same way as those of chrysotile.

Anthophyllite and true asbestos are rarely found in cross-fiber veins because the conditions prevailing at the time of their formation are not, as a rule, favorable for the solution and transportation of these minerals to a new place of deposition. According to Hopkins,[31] the veins of cross-fiber anthophyllite found in Georgia "are in every way similar to chrysotile, the fibers being perpendicular to the inclosing walls, and sometimes continuing from the one to the other, but more commonly jointed one or more times." This description indicates that their origin is similar to that of other cross-fiber veins.

Origin of Slip-fiber

All asbestiform minerals, with the possible exception of crocidolite, are known to occur as slip-fiber, and true asbestos seldom occurs in any other way. The origin of the slip-fiber type is a question that has aroused much less diversity of opinion than the origin of the cross-fiber type. Merrill[32] and others have advocated the efficacy of pressure and shearing in the production of a fibrous structure in minerals. The common association of slip-fiber with slickensided surfaces and other evidences of faulting have apparently convinced most investigators that shearing is the essential factor in its formation. The various ways in which unequal pressure may bring about the development of a fibrous structure have been previously discussed, but there is reason for believing that all of the so-called slip fiber has not been formed directly as a result of unequal pressure.

Cirkel believed that the slip-fiber found in the Broughton district, Quebec, had been formed from cross-fiber chrysotile as "the result of the secondary readjustment, which took place immediately after the crystallization of the fiber in veins. Both rock and veins must have been in a semi-magmatic condition during this period, and pressure may have aided this process of physical alterations of the mass in a marked degree."[33]

[31] O. B. Hopkins: *Op. cit.*, pp. 104–105.

[32] G. P. Merrill: On the Serpentine of Montville, New Jersey, *Proceedings of the U. S. National Museum*, vol. 11, p. 105 (1888), and Notes on Asbestos and Asbestiform Minerals, *Proceedings U. S. National Museum*, vol. 18, p. 289 (1895).

[33] Fritz Cirkel: *Op. cit.*, pp. 94–95.

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It is possible if not probable that some slip-fiber has been formed from cross-fiber, but there is absolutely no reason for postulating that the rock was "in a semi-magmatic condition."

Dresser[34] reached a somewhat different conclusion as to the origin of the Broughton deposits. He notes that the serpentine at Broughton was probably derived from pyroxenite, while that at Thetford, where cross-fiber predominates, was derived from peridotite. After calling attention to the fact that the asbestos is limited to "the sheared and shattered" portions of the rock mass, he argues that there is some connection between the two. He further[35] suggests that when the pyroxenite was altered to serpentine, "the upper portions may have had a development of asbestos in the form of 'mass-fibre' or asbestos irregularly distributed through the rock, possibly due to a greater action of magmatic waters near the top of the sills; and that this fibrous structure weakened

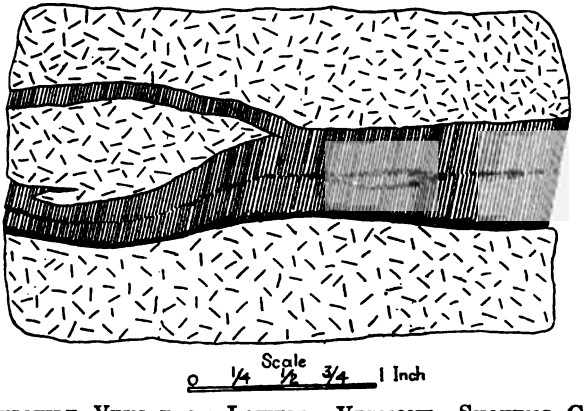

FIG. 7.—CHRYSOTILE VEIN FROM LOWELL, VERMONT, SHOWING GRADATION OF CROSS-FIBER INTO SLIP-FIBER ALONG THE LOWER WALL.

the resisting power of the rock and the shear zone was thus localized." The chief objection to the latter hypothesis is, that while chrysotile is common both as cross-fiber and as slip-fiber, it is not known to occur as mass-fiber.

The investigations of the present writer indicate that much so-called slip-fiber has really been formed in the same way as cross-fiber, as is evidenced by the presence of partings, inclusions and fibers that are oblique rather than parallel to the inclosing walls. In fact, every gradation may be found between cross-fiber veins in which the fibers are nor-

[34] J. A. Dresser: A Preliminary Report on the Serpentine and Associated Rocks of Southern Quebec, *Canada Department of Mines, Geological Survey Memoir* No. 22, p. 69 (1913).

[35] J. A. Dresser: *Idem.*, p. 70.

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mal to the walls, and slip-fiber veins in which the fibers are approximately parallel to the walls. In some instances the gradation may be observed in a single vein, as in the specimen[36] from near Lowell, Vt., sketched in Fig. 7. In this vein some of the fibers terminate at the abrupt bend, but others continue without a break, although they are less flexible where approximately parallel to the vein walls than where they are more nearly normal.

Origin of Mass-fiber

Asbestos of the mass-fiber type, so far as known, is always anthophyllite. Several occurrences of this kind are found in the vicinity of Sall Mountain, Georgia, which may be considered the type locality, as the deposits have been worked on a small scale for several years. More recently similar deposits have been opened up about 14 miles southeast of Kamiah, Idaho.

The deposits at Sall Mountain are roughly elliptical in shape, and the largest, which has been practically mined out, had a length of about 75 ft., a width, near the middle, of 50 ft., and apparently pinched out at a depth of 50 ft. It is estimated that considerably over 90 per cent. of the rock mass is realized as fiber.[37] The fibers are arranged in small groups or bundles, and range up to an inch in length, though averaging only about ½ in. The fibers show a strong tendency to form spherical bunches with radial structure, but because of mutual interference these bodies are, as a rule, only imperfectly developed, and therefore, in most cases, the rock consists of a mass of fibrous bundles and sheaves oriented in all directions. Occasionally, however, cross-fractures show well-formed rosettes of radiating fibers. Individual fibers sometimes appear jointed or broken. They are low in tensile strength and brittle, readily breaking into short lengths so that none of the material is of spinning grade. Hopkins states that, because of lack of flexibility, the fibers are broken so many times during the milling process "that they are exceptionally ¼ in. long, while the bulk is 1/10 in. and less."[38]

According to Diller, the rock found near Kamiah, Idaho, "is very like that mined at Sall Mountain, Georgia, except that in Idaho the fibers are somewhat coarser and the radial groups larger."[39]

Hopkins, who studied and described the Georgia deposits in great

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[36] This specimen was obtained through the courtesy of the U. S. National Museum.

[37] J. S. Diller: Asbestos, *Mineral Resources*, 1907, *U. S. Geological Survey*, pt. II, p. 717 (1908).

[38] O. B. Hopkins: Report on the Asbestos, Talc and Soapstone Deposits of Georgia, *Geological Survey of Georgia, Bulletin* No. 29, p. 88 (1914).

[39] J. S. Diller, Asbestos: *Mineral Resources*, 1909, *U. S. Geological Survey*, pt. II, p. 729 (1910).

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detail, concludes that the origin of the mass-fiber is due largely to the alteration of enstatite.[40]

Since this alteration involves an increase in volume, it probably takes place in the katamorphic zone. The change of enstatite to anthophyllite is paramorphic, and therefore the presence of solutions is not essential, as in the case of reactions involving the addition or subtraction of material. This probably accounts for the difference in the origin and structure of serpentine and of mass-fiber anthophyllite. The alteration of a rock mass to serpentine begins along fractures and all accessible surfaces, gradually penetrating toward the unaltered interior; while the alteration to anthophyllite begins at a large number of centers, more or less evenly distributed through the rock, and then spreads out radially from these centers. The lack of flexibility in mass-fiber anthophyllite is attributed largely to the fact that physical conditions have not controlled and limited its growth to the same extent that they have controlled the formation of the cross-fiber and slip-fiber types. The separation of the fibers is made easier by the hydration of the anthophyllite in the belt of weathering.

[40] O. B. Hopkins: *Op. cit.*, p. 104.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

A Study of the Silica Refractories*

BY J. SPOTTS McDOWELL, B. S.†

(New York Meeting, February, 1917)

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* An investigation conducted by the author at the laboratories of the Massachusetts Institute of Technology from October, 1915, to February, 1916, and published with the consent of Professor H. O. Hofman, Acting Head of the Department of Mining and Metallurgy. The supplementary report represents work done later while associated with the Harbison-Walker Refractories Co. at Pittsburgh, Pa.

† Research Department, Harbison-Walker Refractories Co., Pittsburgh, Pa.

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Acknowledgments

To a very considerable degree I am indebted to the kindness of others for the material herein presented. The subject was suggested to me by Kenneth Seaver, Chief Engineer of the Harbison-Walker Refractories Co., Pittsburgh, Pa., and represents, in fact, the continuation of an investigation begun by him.*

The materials studied were furnished by the Harbison-Walker Refractories Co., and the test brick manufactured at the Hays Station plant of that company under the supervision of R. H. H. Pierce, Chief Chemist. The conditions under which the test brick were made were devised by members of the Harbison-Walker Refractories Co.'s engineering department.

Whatever value the microscopic determinations may possess is due entirely to the careful instruction and kindly counsel of Professor C. H. Warren of the Massachusetts Institute of Technology. I am particularly grateful to Victor Dolmage, who was extremely generous with his time in assisting with some of these determinations. The photomicrographs are the work of W. L. Whitehead. The crushing and cross-breaking tests were made with the assistance of I. H. Cowdrey of the Mechanical Engineering Department of the Institute of Technology. I wish to express my thanks to all those named, as well as to others who have rendered assistance in various ways.

Introduction

The magnitude of the thermal expansion of silica brick, and its inability to withstand rapid temperature changes, present problems of considerable importance in the manufacture of siliceous refractories.

In the study represented by the experimental data herein described, these problems are considered from the point of view afforded by the results of recent investigators: Fenner and his associates of the Geophysical Laboratory, Washington, D. C., who have established the stability relations of the silica minerals, and Endell and co-workers in Berlin, who have studied the changes brought about in the constitution of siliceous refractory materials upon the application of heat.

In the process of manufacture, the quartzite from which the brick is made changes in part into cristobalite and tridymite. The attempt is herein made to determine by microscopic methods the degree of transformation in various specimens of test bricks manufactured at slightly varying temperatures of burning, with varying coarseness of grain, and to determine the effect of repeated burning. The effect of variations

* Kenneth Seaver: Manufacture and Tests of Silica Brick for the Byproduct Coke Oven, *Trans.*, vol. 53, pp. 125–139 (1916).

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in these conditions upon crushing strength and modulus of rupture has also been considered.

Coincidently, a study has been made, from the available literature, of the properties of the silica minerals and the silica refractories. From the knowledge so gained, combined with the experimental data regarding the constitution of the test bricks, deductions have been made as to the conditions of manufacture under which silica brick of decreased thermal expansion and increased power to withstand rapid temperature changes might conceivably be produced. It is hoped that these deductions may serve to indicate possible starting points for future investigations.

The first part of this report comprises data compiled from various publications. Although much of this has no bearing upon the specific problem in hand, it is believed to be of interest in any thorough study of the properties of siliceous refractory materials.

THE SILICA MINERALS

Stability Relations

The crystal modifications of silica important in this connection are quartz, tridymite and cristobalite, each of which possesses α and β phases. Any one of these three modifications may be converted into either of the others by appropriate heat treatment. The formation of tridymite seems always to require the presence of a flux or catalyzer, while cristobalite is formed in the absence of a catalyzer. The inversion temperatures[1] are as follows:

870°C ± 10 quartz $\rightleftarrows$ tridymite
1,470° ± 10 tridymite $\rightleftarrows$ cristobalite
575° α quartz $\rightarrow$ β quartz; 570° β quartz $\rightarrow$ α quartz
117° α tridymite $\rightarrow$ β_1 tridymite; 163° β_1 tridymite $\rightarrow$ β_2 tridymite; reversions on cooling not sharp.
274.6° – 219.7° α cristobalite $\rightarrow$ β cristobalite
240.5° – 198.1° β cristobalite $\rightarrow$ α cristobalite

At ordinary temperatures each mineral exists only in the α phase, changing into the β phase as the temperature is raised. The α to β transformations are all rapid, while considerable time is required for the transformation of one mineral into another.

In the presence of a flux, when any form of silica is heated a sufficient length of time below 870°, quartz is always formed;[2] between 870° and 1,470° tridymite and from 1,470° upward to the melting point of silica, cristobalite. *Heated without a flux*, the inversion from quartz to tridymite does not occur, but the transformation is direct to cristobalite,

[1] C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, p. 383 (1913).

[2] *Ibid.*, p. 358.

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and the temperature of incipient formation of cristobalite is depressed to about 1,250°; under the same conditions, tridymite is converted into cristobalite, with the inversion temperature raised to about 1,570°. Fenner's experiments on quartz powder heated without a flux gave the following results:

108 hr. at 1,250°: a very small per cent. of cristobalite formed.
90 hr. at 1,360°: resultant product two-thirds cristobalite, one-third quartz.
1 hr. at 1,570°: transformation into cristobalite nearly complete.

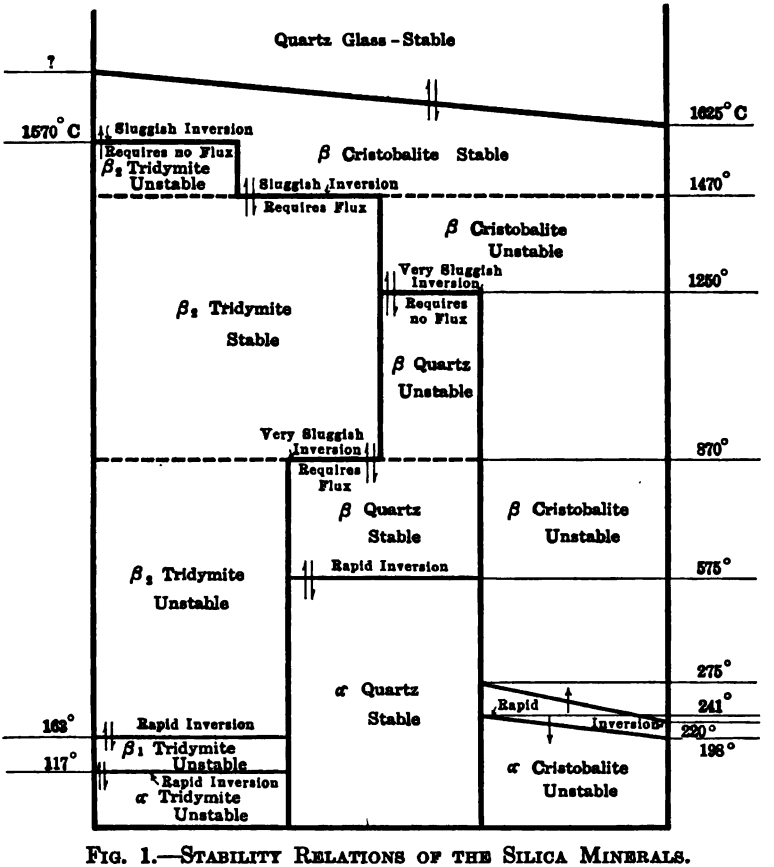

FIG. 1.—STABILITY RELATIONS OF THE SILICA MINERALS.

Quartz glass heated above 1,200° becomes gradually converted into cristobalite.[3] The percentages of cristobalite formed under various conditions of heating are shown in the accompanying table, taken from the work of Rieke and Endell.

[3] Rieke and Endell: Devitrification of Quartz Glass, *Silikat Zeitschrift*, vol. 1, No. 1, p. 8 (Jan., 1913).

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TABLE 1.—*Cristobalite Formed by Heating Quartz Glass under Various Conditions (Rieke and Endell)*

Time of Heating, Hours	Temperature				
	1,200°	1,300°	1,400°	1,500°	1,600°
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
½	—	—	5	34	60
1	0	2	17	50	90
2	0	5	25	100	
4	0	10	—		

Quartzite heated three to five times to 1,450° in a porcelain kiln is converted chiefly into cristobalite, with here and there scattered residual

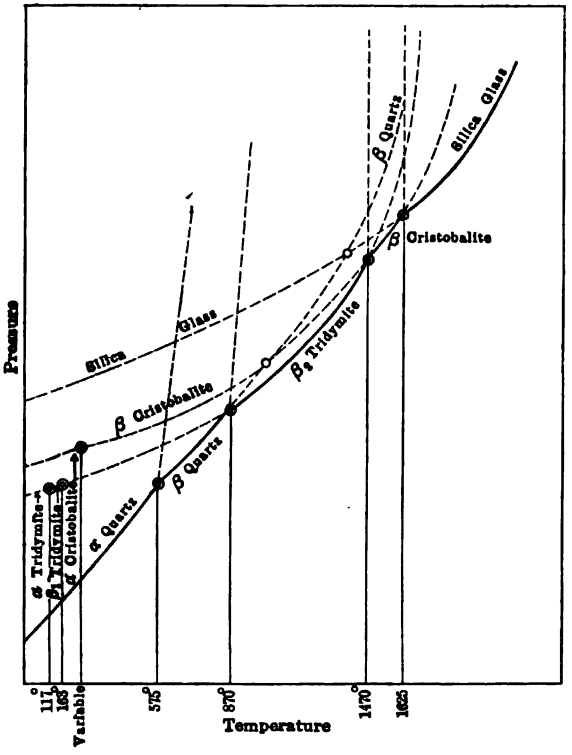

FIG. 2.—STABILITY RELATIONS OF THE SILICA MINERALS (AFTER FENNER).

quartz grains; after 10 burnings at 1,450° wedge-shaped twin crystals of tridymite begin to appear.[4] Silica brick, which consists principally of

[4] K. Endell: *Stahl und Eisen*, vol. 33, p. 1855 (Nov., 1913).

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cristobalite when placed in use, gradually changes at high temperatures in the furnace to tridymite, as explained later on in this paper.

The stability fields and existence ranges of the various forms of silica are shown in Fig. 1; an equilibrium diagram (after Fenner) in Fig. 2.

TABLE 2.—*Optical Properties**

Composition	Crystal System	Crystal Habit	Elongation	Optical Orientation	nNa		Optical Character	Remarks
					α	γ		
α Quarts.....	Hexagonal Trapezohedral Tetartohedral	Pyramidal	Y	$c = \gamma$	1.544	1.553	+	Stable below 575°.
β Quarts.....	Hexagonal Trapezohedral Hemihedral	Pyramidal	Y	$c = \gamma$			+	Stable between 575° and 870°. Inverts on cooling to α quarts.
α Tridymite...	Orthorhombic	Thin pseudo-hexagonal plates			1.469	1.473	+	Exists below 117°. Usually finely intergrown aggregates. Optic axial angle large.
β Tridymite...	Hexagonal	Hexagonal plates		$c = \gamma$			+	Exists only above 117°. Inverts on cooling to α tridymite.
α Cristobalite.	Probably tetragonal	Equaint grains		$c = \alpha$	1.484	1.487	−	Exists below 275°.
β Cristobalite..	Isometric	Equaint grains						Exists only above 198°. Inverts on cooling to α cristobalite.

* C. N. Fenner: *American Journal of Science*, ser. 4, vol. 36, pp. 351–356 (1913).
N. L. Bowen: *American Journal of Science*, ser. 4, vol. 38, p. 245 (1914).
Rankin and Wright: *American Journal of Science*, ser. 4, vol. 39, pp. 4 and 74 (1915).

Tridymite crystals usually occur as crystalline aggregates of random orientation, as broad, thin hexagonal plates, which appear as needles or laths when turned on edge, and as wedge-shaped twin crystals. The hexagonal plates of usual thinness appear perfectly isotropic when lying on the base; the needle or lath-like sections have weak double refraction, parallel extinction and negative elongation.

Cristobalite does not develop a well-defined crystal form but usually occurs as minute branching skeleton crystals, with the branches often showing octahedral terminal caps. The higher temperature form of cristobalite is isometric; on cooling it becomes weakly birefringent. The birefringence in thin sections is barely discernible with the sensitive tint plate.

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TABLE 3.—*Density*

	Specific Gravity	
Quartz	2.65	
Tridymite	2.270	Artificial tridymite (Fenner).
	2.28	Natural tridymite (Mallard).
	2.32	(Endell).
Cristobalite	2.333	Artificial cristobalite (Fenner).
	2.34	Natural cristobalite (Mallard).
	2.33	(Endell).
Cristobalite	.....	At 1,500° has same density as quartz glass; at 300°, slightly greater density than quartz glass (Rieke and Endell).
Quartz glass	2.21	(Dana, Endell).
	2.194	(Schwarz).

MELTING POINTS

"It appears that the fusing point of quartz is lower than 1,470°, but that at this temperature it passes into cristobalite almost as rapidly as it melts. The fusing point of tridymite should lie between those of quartz and cristobalite."[5] "It is clear that the indicated melting point of cristobalite must be higher than 1,625°, the value found by Fenner, higher than 1,685° even, the value found by Endell and Rieke.[6] . . . . . As Dr. Fenner has suggested, cristobalite may have a variable molecular constitution and a similarly variable melting point according to the conditions under which it was formed."[7] Kanolt[8] finds that pure silica flows distinctly at 1,750°, which is, therefore, the apparent melting point and higher than the true melting point. Quartz glass softens at 1,500° and melts between 1,700° and 1,800°.[9]

THERMAL EXPANSION

The results of LeChatelier's[10] experiments are shown in Fig. 3; those of Day, Sosman and Hostetter[11] in Fig. 4.

[5] C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, p. 383 (1913).

[6] Endell and Rieke: *Zeitschrift für Anorganische, Chemie.*, vol. 79, pp. 239–259 (1913).

[7] N. L. Bowen: Ternary System: Diopside-Forsterite-Silica, *American Journal of Science*, ser. 4, vol. 38, p. 218 (1914).

[8] C. W. Kanolt: Melting Point of Fire-Bricks, *U. S. Bureau of Standards, Technologic Paper* No. 10, p. 14 (1912).

[9] A. E. Marshall: Fused Silica Ware, Its Manufacture, Properties and Uses, *Metallurgical and Chemical Engineering*, vol. 10, pp. 248–249 (Apr., 1912).

[10] LeChatelier: La Silice, *Revue Universelle des Mines*, ser. 5, vol. 1, p. 90 (1913).

[11] Day, Sosman and Hostetter: Determination of Mineral and Rock Densities at High Temperatures, *American Journal of Science*, ser. 4, vol. 37, pp. 1–39 (1914).

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LeChatelier gives the mean linear expansion of quartz glass as 0.0$_6$7; Kaye[12] gives the following figures: 0° to 30°C., 0.0$_6$42; 30° to 100°, 0.0$_6$53; 100° to 500°, 0.0$_6$58; 500° to 900°; 0.0$_6$50; 900° to 1000°; 0.0$_6$80. Quartz glass, partly devitrified by long-continued heating at a high tem-

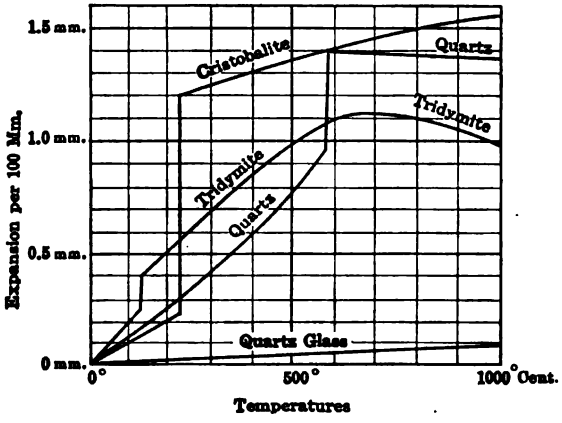

FIG. 3.—RESULTS OF LE CHATELIER'S EXPERIMENTS IN THERMAL EXPANSION.

perature, so that it contains much β cristobalite, if cooled rapidly to 300° remains completely clear and transparent, and but few cracks appear. When further cooled, however, to 230°, the sudden formation of numerous

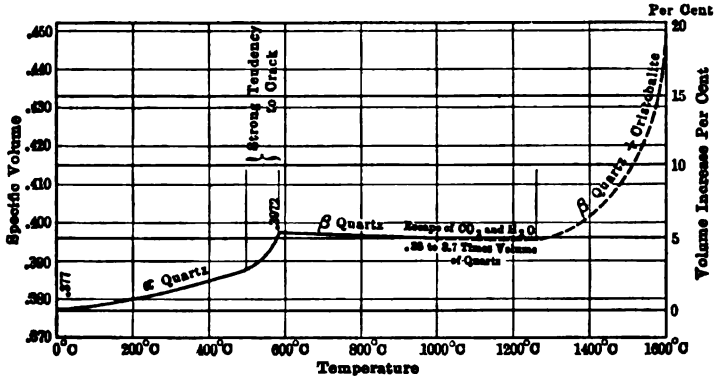

FIG. 4.—THERMAL EXPANSION OF QUARTZ.

fine cracks causes the glass to become white and opaque, coincident with the inversion of β to α cristobalite. The specific volume and coefficient of expansion of β cristobalite are, therefore, nearly the same as those of quartz glass; the presence of a few cracks at 300° indicate that the

[12] G. W. C. Kaye: Expansion and Thermal Hysteresis of Fused Silica, *Philosophical Magazine*, ser. 6, vol. 20, pp. 718–728 (Oct., 1910).

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cristobalite probably has a slightly greater coefficient of expansion than the glass.[13]

Solubilities

Rammelsberg,[14] and Lange and Milberg[15] studied the solubility of quartz, opal and amorphous silica in alkaline solutions, but not that of tridymite or cristobalite. Cramer found the solubility in KOH of quartzite burned once in a porcelain kiln to be 41 per cent.; burned 10 times, 70 per cent. After the first burning, this material probably consisted of a mixture of cristobalite and quartz; after the tenth burning, of cristobalite and tridymite, with a small amount of quartz.[16] Schwarz[17] obtained the results shown in Table 4 on boiling powders with grains approximately 0.04 mm. in diameter.

Table 4

Reagent	Solubility of				Time, Hours
	Quartz, Per Cent.	Tridymite, Per Cent.	Cristobalite, Per Cent.	Amorphous Silica, Per Cent.	
5 per cent. Na_2CO_3 solution	2.11	2.77	....	....	½
5 per cent. HF solution...	30.17	76.30	74.3	96.6	½
1 per cent. HF solution...	5.20	20.30	25.8	52.9	1

Identification of Cristobalite and Tridymite

The method of identification most readily applied is by determination of the indices of refraction by the immersion method, described later on in this paper. Tridymite may also be recognized by its characteristic needles, laths and wedge-shaped twin crystals.

The methods of Rieke and Endell,[18] who have gone into this subject with great care, are given in the following paragraphs. Specific-gravity determinations were of little value in the case of quartz converted by heating into cristobalite or tridymite, for several reasons. The specific gravities of the two last-named are not greatly different; the material always contains some unaltered quartz; and the end products are so

[13] Rieke and Endell: Devitrification of Quartz Glass, *Silikat Zeitschrift*, vol. 1, No. 1, p. 6 (Jan., 1913).

[14] *Annalen der Physik und Chemie, Poggendorf*, vol. 112, pp. 177 (1861).

[15] *Zeitschrift für Angewandte Chemie*, 1897, pp. 393 and 425.

[16] K. Endell: *Stahl und Eisen*, vol. 33, 1770 *et seq.* (Oct., 1913).

[17] R. Schwarz: Chemical Relations of the Various Modifications of SiO_2, *Zeitschrift für Anorganische Chemie*, vol. 76, p. 424 (1912).

[18] Rieke and Endell: Volume Change of Some Ceramic Raw Materials on Burning, *Silikat Zeitschrift*, vol. 1, No. 4, pp. 67, 85 (Apr., 1913).

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filled with microscopic cracks that the results of specific-gravity determinations are always low.

These fine cracks interfered with microscopic investigations. On account of total reflection at the borders, according to Rieke and Endell, the greater part of the end product appears isotropic, and the birefringence can be detected only upon very careful observation.

By the use of the thermal microscope, it was seen that cristobalite becomes isotropic at 225° to 230°. The effect was particularly evident in cristobalite derived from quartz glass or from quartz heated above 1,600°; it could not be used as a means of identification in the case of cristobalite formed below 1,450°, which, on account of the cracks

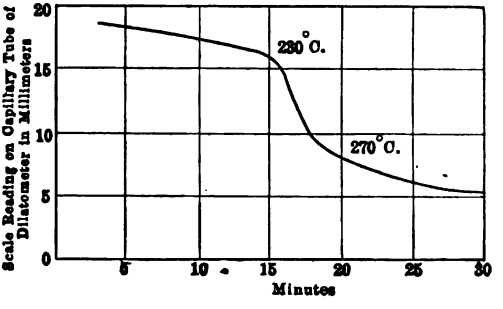

FIG. 5.—CURVE SHOWING EFFECT OF HEAT ON A SPECIMEN OF SILICA BRICK COMPOSED ESSENTIALLY OF CRISTOBALITE.

above mentioned, appears nearly isotropic. The change in volume of cristobalite at 230° is considerable. For that reason, quicksilver dilatometer observations offered a sure way of distinguishing cristobalite from tridymite. A specimen of silica brick, composed essentially of cristobalite, on heating[19] gave the curve shown in Fig. 5.

Rieke and Endell have designated the sudden clearing up shown by cristobalite heated carefully over the flame at about 230°, as the "cristobalite reaction." This effect can easily be observed with the naked eye in the case of cristobalite derived from quartz glass.

THE SILICA REFRACTORIES

RAW MATERIAL

American Deposits

The raw material used for the manufacture of high-grade American silica brick is a true quartzite; the deposits most extensively used are described by Seaver.[20] Named in the order of their importance, they

[19] *Stahl und Eisen*, vol. 33, p. 1856, Fig. 4 (Nov., 1913).

[20] K. Seaver: Manufacture and Tests of Silica Brick for the By-Product Coke Oven, *Trans.* vol. 53, pp. 125–139 (1916).

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are the so-called Medina or Tuscarora sandstone of Huntingdon and Blair Counties, Pennsylvania, the Baraboo quartzite of the Devil's Lake

FIG. 6.—GANISTER FLOE IN THE GORGE OF THE JUNIATA RIVER, NEAR MOUNT UNION, HUNTINGDON CO., PENNSYLVANIA.

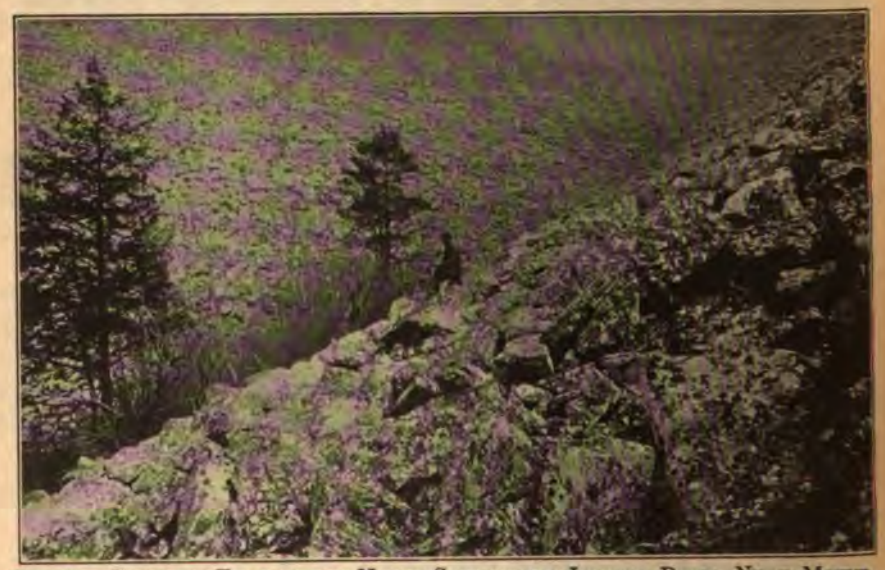

FIG. 7.—GANISTER FLOE ON THE NORTH SIDE OF THE JUNIATA RIVER, NEAR MOUNT UNION, HUNTINGDON CO., PENNSYLVANIA.

region, Wisconsin, and the deposits of eastern Alabama and of Colorado.

Where quarried for refractory purposes, the Medina is white, hard

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and resistant. Its steeply pitching beds form the crests of many of the mountains of central Pennsylvania, and the steep slopes below the summit are covered by great bodies of talus, composed of blocks of the rock, varying in size from pieces weighing a few pounds to those weighing many tons. The talus slopes sometimes cover areas of a thousand acres or more where the beds have been cut through by streams. For the manufacture of silica brick, most of the rock used is taken from these bodies of loose rock, locally called "ganister floes," although rock from the solid measures is sometimes used.

Figs. 6 and 7 are pictures of ganister floes near Mt. Union, Huntington Co., Pa., where the Juniata River cuts through Jack's Mountain.

TABLE 5.—*Typical Analyses**

Constituent	Medina Quartzite from Pennsylvania	Baraboo Quartzite	Alabama Quartzite
SiO_2	97.80	97.15	97.70
Al_2O_3	0.90	1.00	0.96
Fe_2O_3	0.85	1.05	0.80
CaO	0.10	0.10	0.05
MgO	0.15	0.25	0.30
Alkalies	0.40	0.10	0.31

* K. Seaver: *Op cit.*, p. 127–128.

Properties Determining Usability

Not all pure quartzites are adapted to the manufacture of refractories. For refractory purposes three properties are to be considered: mechanical strength, melting point and behavior on burning. Experience shows that to produce a physically strong brick the stone should be hard, dense, and give splintery, angular fragments of heterogeneous size and shape on crushing; one that crushes down to rounded grains can not be used. The strength of the brick and its melting point are conditioned by the analysis of the rock. The silica content may vary from 96 to 98 per cent.; if less silica is present the fusion point is lowered. Attempts to use pure quartzite of 99 per cent. SiO_2 have not met with success. The presence of Al_2O_3 and Fe_2O_3 is necessary to form a bond; their combined percentage is usually about 1.75 per cent., and should not exceed 2.5 per cent. The amount of alkalies should be less than 0.5 per cent.

The melting interval of the stone depends upon its content of bases. For good quartzite it is usually considered to be cone 35 to 36; that is, 1,755 to 1,775°C.[21] Endell[21] obtained the following results, measuring temperatures by means of an iridium-iridium-ruthenium thermoelement

[21] K. Endell: *Stahl und Eisen*, vol. 33, p. 1857 (Nov., 1913).

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in the calibration of which the melting points of gold, palladium and platinum were taken as 1,063°, 1,549° and 1,755°, the values determined by Day and Sosman:[22]

At 1,650°C., fine powder of good quartzite sintered together; pea-sized pieces showed no signs of fusion.

At 1,690°C., fine powder of good quartzite melted together into drops; the pea-sized pieces were superficially melted with visible rounding of the edges.

At 1,710°C., little pieces of pure Brazilian quartz 1 to 2 mm. in cross-section were melted round after 10 min. heating; a piece of ¼-in. cross-section had strongly rounded edges. Pea-sized pieces of good quartzite were strongly vitrified and showed fully rounded edges.

Expansion on Heating[23]

Chemical analysis alone does not afford an adequate index to the behavior of a quartzite on heating. Material suitable for refractory purposes does not weaken or crack perceptibly under the influence of high temperatures, and shows considerable volume increase after a first burning, retaining essentially the same volume later after repeated burning. Unsuitable quartzites, notably the pure, coarsely crystalline variety, become badly cracked or completely disintegrated on heating; they usually expand but little on a first burning and considerably thereafter.

TABLE 6

Material	Volume Increase after Burns				
	1	2	3	4	5
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Good quartzite (average of three samples)	9.4	10.7	11.0	11.3	11.5
Quartzite unsuitable for refractory purposes (average of three samples)	5.3	6.5	6.5	9.7	10.7
Melting point of all samples cone 35 to 36.					

In Cramer's[24] experiments, hard, fine-grained quartzites, which did not weaken in the heat treatment, gave the greatest expansion on the first burn. The volume increase of one specimen of good quartzite heated

[22] Day and Sosman: *Zeitschrift für Anorganische Chemie*, vol. 72, pp. 1–10 (Aug., 1911).

[23] F. T. Havard: *Refractories and Furnaces*, p. 36, McGraw-Hill Book Co., New York, 1912.

[24] *Stahl und Eisen*, vol. 21, p. 772 (July, 1901); *Tonindustrie Zeitung*, vol. 25, p. 864 (1901).

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to cone 16 (1,460°±) was 17.1 per cent. after the first burn, 22 per cent. after the sixth. This enlargement was due to two factors: (1) an increase in porosity and (2) an expansion of the mineral itself, resulting from the gradual transformation of the quartz into cristobalite and tridymite. Only the latter effect is shown in the figures of Table 6, calculated from specific gravities of samples of quartzite heated repeatedly to 1,450° in a porcelain kiln.[25]

Texture

The insufficiency of analyses and melting-point determinations to indicate the value of a given quartzite for refractory purposes led Wernicke and Wildschrey[26] to seek an explanation in extensive microscopic studies of the texture of the rock. Their conclusions are as follows:

Typical quartzites, consisting principally of differently oriented intergrown quartz grains fairly uniform in size, metamorphosed quartzites, or those showing undulatory extinction under the microscope, can not be used. Good quartzites consist of quartz grains, mostly rounded, in a groundmass or cement of amorphous silica or crypto-crystalline quartz. No muscovite was found in such quartzites. Since expansion upon heating presents no difficulties in the case of quartzites containing a cement, only their melting point need be considered. They expand without cracking and acquire nearly their whole expansion on the first burn. In consequence of the fineness of division of the impurities in the cement and the slight sintering caused thereby, they are much stronger after burning than the typical quartzites in which the impurities are not so finely divided. The latter crack on burning and attain their complete expansion more slowly.

Grum-Grzimailo,[27] on the other hand, expresses the opinion that any quartzite with more than 94 to 95 per cent. SiO_2 may be used. He states that the purer the material and the larger the single quartz grains the longer must be the time of burning; the finer the quartz and its impurities the more rapidly will the required changes take place on heating.

Wernicke and Wildschrey's views are not supported by microscopic examinations of the most important American quartzites as shown later in this paper.

Silica Brick

The material herein discussed is termed "silica brick" in American practice, and is made from quartzite of 96 to 98 per cent. SiO_2 content to

[25] Calculated by the writer from data of K. Endell: *Stahl und Eisen*, vol. 33, p. 1774 (Oct., 1913).

[26] Wernicke and Wildschrey: Quartzite and Its Application in the Refractories Industry, *Tonindustrie Zeitung*, vol. 34, pp. 688, 723 and 768 (1910).

Wernicke: Quartzite and Silica Brick, *Stahl und Eisen*, vol. 33, p. 1860 (Nov., 1913).

[27] *Stahl und Eisen*, vol. 31, p. 225 (1911).

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which about 2 per cent. of lime is added to serve as a bond. It is not to be confused with the so-called "quartzite" brick, which contains much less silica, and in which fire clay is the bonding material.

Manufacture

The processes of manufacture are discussed by Seaver,[28] Hofman,[28] and Havard.[28] After grinding in a wet pan, where the lime is added, the charge is molded into brick, dried, and burned in a down-draft kiln.

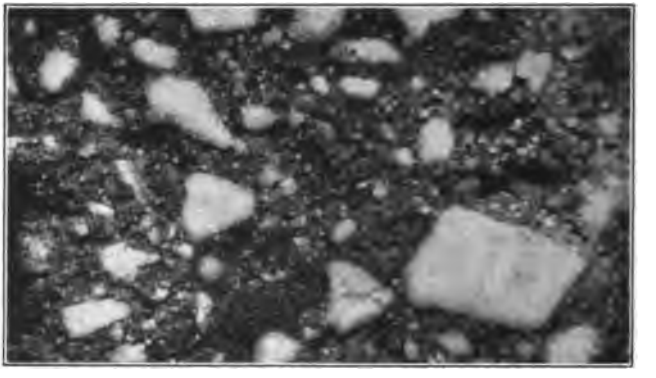

FIG. 8.—PHOTOMICROGRAPH OF POLISHED SURFACE OF 9-IN. SILICA BRICK. × 2.

The fineness of grinding is determined by the character of brick to be made; the grind used for special shapes has a larger proportion of fines

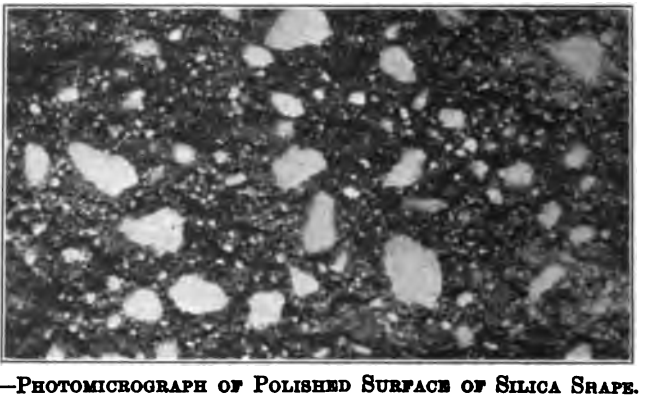

FIG. 9.—PHOTOMICROGRAPH OF POLISHED SURFACE OF SILICA SHAPE. × 2.

than that used for standard 9-in. brick. In either case, the ground material consists of a mixture of grains varying in size from that of a pea to the finest powder.[29] All silica brick are made in molds smaller than

[28] See Bibliography.

[29] See screen analyses, Table 13.

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the proposed size of the finished brick; 3/8 to 7/16 in. per foot is allowed for expansion in burning.

The brick must be burned to such a temperature that as much as possible of the permanent expansion shall be acquired in the kiln, and not later in the furnace. The burning temperature is usually given as cone 15 to 16, or 16 to 17, and the time of burning as 20 days, including the time required for cooling. Lange[30] states that cone 15 should be melted down, cone 16 beginning to melt.

Properties of Silica Brick

Analysis.—A typical analysis[31] of brick made from Pennsylvania rock is as follows:

	Per Cent.
SiO_2	96.25
Al_2O_3	0.88
Fe_2O_3	0.79
CaO	1.80
MgO	0.14
Alkalies	0.39
	100.25

Brick with less than 94 per cent. SiO_2 or more than 3 per cent. CaO do not have the necessary refractory qualities; if less than 1.5 per cent. CaO is present, the brick are weak.

Density.—Principally on account of the progressive transformation of quartz into other modifications of silica upon the application of heat, the density of a silica brick is dependent upon the time and temperature of burning. Within certain limits, long-continued heating or burning at higher temperatures causes the density to diminish. The specific gravity of a silica brick should presumably lie between 2.65 (the value for quartz) and 2.28 (the value for tridymite). Microscopic cracks permeating the material render accurate determinations difficult, and results obtained in the ordinary way by the use of large pieces are quite sure to be inaccurate, even when the sample has been previously boiled.

For more accurate results, the finely powdered substance, boiled to remove adherent air bubbles, is usually tested in a pycnometer. Although somewhat laborious, this method with homogeneous materials gives results accurate in the second decimal place. As burned quartz is not homogeneous (see Figs. 16 and 17), and as the very fine cracks and pores are not always filled by the pycnometer liquid, the accuracy of the determinations in the case of silica brick is considered to be ± 0.02.[32]

[30] *Stahl und Eisen*, vol. 32, p. 1729 (Oct., 1912).
[31] K. Seaver: *Op. cit.*, p. 134.
[32] K. Endell: *Stahl und Eisen*, vol. 33, p. 1773 (Oct., 1913).

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In this way Goerens and Gilles[33] determined the specific gravity of certain samples as 2.44. The figures given for the volume density of silica brick range from 1.50 to 1.88, averaging about 1.67 for a well-burned 9-in. brick. The values quoted by different writers for the "true specific gravity" of this material are quite inconsistent, varying from 2.05[34] to 2.75.[35]

Inconsistencies inexplicable by variations in the materials studied may perhaps be explained on the following grounds:

Let W = weight of specimen in grams.
V = entire volume of body in cubic centimeters.
V_s = volume of rock substance.
V_p = volume of pores.
V_o = volume of open pores, which absorb water.
V_c = volume of closed pores and cracks, which absorb no water.
$V_p = V_o + V_c$.
$V = V_s + V_p = V_s + V_o + V_c$.

It is evident that three possible values may be obtained for the specific gravity.

1. G_b. Bulk specific gravity $= \frac{W}{V} = \frac{W}{V_s + V_o + V_c}$.

2. G. True specific gravity $= \frac{W}{V_s}$. As already pointed out, for silica brick this value should probably lie somewhere between 2.28 and 2.65, depending on the conditions of burning.

3. G'. Apparent specific gravity, determined from the weights of dry specimen and water displaced after saturation. Apparent specific gravity $= \frac{W}{V_s + V_c}$. Its value lies between that of G and G_b and is dependent on the relation of closed to open pore space. When $V_c = O$, $G' = G$; when $V_c = O$, $G' = G_b$.

In many cases, the values quoted for the "true specific gravity" of silica brick are obviously too low. It is not improbable that in such cases the value designated above as G' has been considered to be the true density and so reported.

Porosity.—The porosity of silica brick varies within wide limits, dependent upon the methods of manufacture. The importance of this property lies in its relation to thermal conductivity and permeability

[33] Goerens and Gilles, *Ferrum*, vol. 12, p. 17 (Nov., 1914).

[34] F. T. Havard: *Op. cit.*, p. 37.

[35] Wologdine and Queneau: Conductivity, Porosity, and Gas Permeability of Refractory Materials, *Electrochemical and Metallurgical Industry*, vol. 7, p. 433 (Oct., 1909).

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to gases.[36] Under ordinary conditions pore space acts as a heat insulator.[37] Following the notation of the preceding paragraph,

Total pore space in per cent. of volume of body $= P_t = \frac{100\,V_p}{V} = 100\,\frac{(1 - G_a)}{G}$.

Open pore space in per cent. $= P_o = \frac{100\,V_o}{V} = 100\,\frac{(1 - G_b)}{G'}$.

P_o is determined by absorption tests; P_t, by ascertaining the true specific gravity. Many writers upon silica brick do not make this distinction, but assume that $P_o = P_t$. The figures commonly given for porosity range from 18 to 43 per cent. of volume of the brick.

Strength.[38]—The figures in Table 7 are taken from Havard.

TABLE 7.—*Figures Taken from Havard**

Silica Brick Made in	Crushing Strength, Pounds per Square Inch		
	Side	Edge	End
Illinois	2,345	1,546	2,119
Pennsylvania	2,896	1,044	1,551
Missouri	2,370	1,792	1,803

Seaver[39] studied the effect of reburning on crushing strength and modulus of rupture with silica bricks made in Pennsylvania burned repeatedly to 1,540°±. His results (Table 8) show that with brick burned three times, the strength of the brick is increased by each reburning.

TABLE 8

Number of Burns	Average Modulus of Rupture, Pounds per Square Inch. Brick Stood on Edge, Supports 6 In. apart	Average Crushing Strength, Pounds per Square Inch. Brick Tested Flatwise
1	624	4,313
2	809	4,396
3	1,001	4,500

At high temperatures, the strength of a brick is much less than at

* F. T. Havard: *Op. cit.*, p. 37.

[36] Wologdine and Queneau: *Op. cit.*
Goerens and Gilles: *Op. cit.*

[37] Dougill, Hodsman and Cobb: Thermal Conductivity of Refractory Materials, *Journal of the Society of Chemical Industry*, vol. 34, pp. 465–471 (May, 1915).

[38] See also under Cross-breaking and Crushing Tests.

[39] K. Seaver: *Op. cit.*, p. 138.

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ordinary temperatures.[40] Subjected to a load of 50 lb. per square inch and gradually heated, chrome bricks fail suddenly at 1,450°, magnesia bricks at 1,550°. Silica bricks under the same conditions are unaffected at 1,450°.

Melting Point.—The melting interval of silica brick is generally considered to lie between cones 34 and 36 (1,740 to 1,775° C.)[41]; Havard[42] states that silica brick softens between 1,750 and 1,800°. Kanolt[43] measured the temperature at which pieces of hazelnut size visibly changed their form. For three different pieces the temperatures were 1,700°, 1,705°, 1,700°. Endell[44] tested brick consisting essentially of cristobalite, as well as brick consisting essentially of tridymite. Pea-sized pieces of both became completely melted at 1,700° ± 10.

The apparent discrepancy between the melting temperature of silica brick as determined in the laboratory, and the temperatures to which they are sometimes subjected in practice, is explained as follows:[45] Only the immediate surface of the brick reaches the furnace temperature. This is soon melted when 1,700° is exceeded, forming a viscous glaze, which protects the portion of the brick immediately adjacent. This portion does not reach its melting point on account of the temperature drop toward the exterior of the furnace.

TABLE 9.—*Specific Heat of Silica Brick**

t°	Heyn		Steger	Norton†
	True Specific Heat at t°	Mean Specific Heat between 20° and t°	Mean Specific Heat between 0° and t°	Mean Specific Heat between 20° and t°
0° C.	0.200	0.200		
200° C.	0.237	0.219	0.205	
400° C.	0.270	0.237		
600° C.	0.282	0.250		0.225
800° C.	0.285	0.261		0.236
1,000° C.	0.298	0.262		
1,100° C.				0.255
1,200° C.	0.291	0.266		

*E. Heyn: Thermal Conductivity of Refractory Building Materials, *Mitteilungen aus dem Königlichen Materialprüfungsamt*, vol. 32, p. 180 (1914).

W. Steger: Specific Heats of Refractory Products, *Silikat Zeitschrift*, vol. 2, No. 11, p. 203 (Nov., 1914).

†C. L. Norton: Private communication to K. Seaver.

[40] G. H. Brown: Note on Load Tests Made on Magnesite, Chrome and Silica Brick, *Transactions of the American Ceramic Society*, vol. 14, pp. 391–393 (1912).

[41] K. Endell: *Stahl und Eisen*, vol. 33, p. 1856 (Nov., 1913).

[42] F. T. Havard: *Op. cit.*, p. 29.

[43] *U. S. Bureau of Standards, Technologic Paper* 10 (1912).

[44] K. Endell: *Op. cit.*, pp. 1857, 1858.

[45] K. Endell: *Op. cit.*, pp. 1857, 1858.

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Thermal Conductivity.—The divergence in the results obtained by different observers in the determination of thermal conductivity has occasioned considerable discussion. As Dudley points out, these variations are due to variations in the materials tested, as well as to differences between, or inaccuracies of, the methods employed. For comparative purposes, the coefficients are given in Table 10 for some fire-clay brick also. The coefficient k represents the flow of heat in calories per second, per square-centimeter area, through 1-cm. thickness, for a temperature difference of 1° C. To convert these figures into B.t.u. per 24 hr. per square-foot area per inch thickness per 1° F. temperature difference, they should be multiplied by 69,700.

TABLE 10

Material	Conductivity			Reference
	Temperature		Mean k between t_1 and t_2	
	t_1	t_2		
Silica brick, "Star" brand 95.9 per cent. SiO_2	0	100	0.0021	Calculated from results of Boyd Dudley, Jr.
	0	1,000	0.0031	
Clay brick, "Woodland" brand 52.9 per cent. SiO_2, 42.7 per cent. Al_2O_3	0	100	0.0016	*Transactions of the American Electrochemical Society*, vol. 27, p. 336 (1915).
	0	1,000	0.0025	
"Quartzite" brick, 73.9 per cent. SiO_2, 22.9 per cent. Al_2O_3.	0	100	0.0020	
	0	1,000	0.0027	
Silica brick, 96 per cent. SiO_2, specific gravity, 2.44	0	100	0.0028	Calculated from results of Goerens and Gilles, *Ferrum*, vol. 12, pp. 1, 17 (Oct., Nov., 1914).
	0	1,000	0.0031	
Claybrick, 53.9 SiO_2, 40.2 Al_2O_3	0	100	0.0022	
	0	1,000	0.0027	

Notwithstanding the lack of numerical correspondence in results, Dudley, as well as Goerens and Gilles, finds that the coefficients are greater at high temperatures than at low, and that the conductivity of silica brick is greater than that of clay brick. The latter conclusion is corroborated by Brown,[46] who compared the flow of heat through the following materials:

Silica brick.....95.5 per cent. SiO_2, 1.9 per cent. Al_2O_3.
Quartzite brick.....83.4 per cent. SiO_2, 11.8 per cent. Al_2O_3.
Clay brick.....63.2 per cent. SiO_2, 33.0 per cent. Al_2O_3.

Cylindrical specimens were heated at one end to 1,300° C. while the other end was exposed to the atmosphere at 31° C. Under these condi-

[46] G. H. Brown: Relative Thermal Conductivities of Silica and Clay Refractories, *Transactions of the American Ceramic Society*, vol. 16, pp. 382–385 (1914).

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tions, after equilibrium had been attained, the conductivity of the silica brick was 9 per cent. greater than that of the clay brick, and 3 per cent. greater than that of the quartzite brick.

Wologdine[47] shows that the conductivity of both silica and clay refractories increases with higher temperatures of burning. His figures indicate that silica is a poorer conductor than clay, for material burned at, or lower than 1,300°. Marshall[48] quite properly holds that while Wologdine's data are probably correct for the materials investigated, his figures are not practically applicable in a comparison of American refractories, since commercial silica brick are burned at a higher temperature than 1,300°.

The data given by Boyd Dudley, Jr.,[49] are of particular interest, in that the materials studied are three well-known brands of American refractories. The mean conductivities between any two temperatures t_1 and t_2 can be calculated from his results expressed by the following straight-line formula based upon measurements taken up to about 1,000° C.:

"Woodland" brick..... $k = 0.0015 + 0.0_410\ (t_1 + t_2)$
"Quartzite" brick..... $k = 0.0019 + 0.0_408\ (t_1 + t_2)$
"Star" silica brick..... $k = 0.0020 + 0.0_411\ (t_1 + t_2)$

These formulæ should be used with caution above 1,000°, and the formula for silicia brick should probably not be applied above 1,050°, as Heyn[50] finds a sharp rise in the conductivity of this material between 1,050° and 1,100°.

Expansion.—The problem of the expansion of silica brick is thus stated by Seaver:[51] In the process of manufacture, a permanent expansion occurs under the influence of heat, swelling the brick from the mold or "green" brick size to the required or "burned" size. It is desired that the permanent expansion should, if possible, be completed in the burning, so as to prevent further swelling in subsequent use. "In every burned brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. This thermal or temporary expansion occurring upon heating a burned brick is not to be confused with the permanent expansion produced in the body of the brick during its manufacture. It is argued that if a silica brick is not properly burned.

[47] *Electrochemical and Metallurgical Industry*, vol. 7, pp. 383 and 433 (Sept.-Oct., 1909).

[48] S. M. Marshall: Relative Thermal Conductivity of Silica and Clay Brick, *Metallurgical and Chemical Engineering*, vol. 12, p. 74 (Feb., 1914).

[49] See Table 10.

[50] *Mitteilungen aus dem Königlichen Materialprüfungsamt*, vol. 32, p. 181 (1914).

[51] K. Seaver: *Op. cit.*, p. 133.

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then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion, but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for."

Since the final increments of permanent expansion are acquired but slowly under the influence of high temperatures it is found that reburned brick show a slight increase in volume after each of several heat treatments. It is generally held that the manufacturer can do nothing to diminish the temporary expansion.[52]

The linear expansion in the burning from "green brick" size to size of finished product is 3/8 to 7/16 in. per foot, that is, 3.1 to 3.6 per cent. It may be here noted that the change of volume in the complete transformation of quartz to cristobalite corresponds to a linear expansion of 4.35 per cent.; quartz to tridymite, 5.3 per cent.

With a number of brick burned repeatedly to cone 16 to 17, Cramer[53] found an average permanent linear expansion of 3.2 per cent. on the first burn, an increment of 0.6 per cent. on second burn and 0.2 per cent. on the third. In a test mentioned by Havard,[54] a Pennsylvania silica brick reheated to 1,400° C. had a total linear expansion of 1.7 per cent., of which 0.4 per cent. was permanent, 1.3 per cent. temporary. Stockman and Foote,[55] at the Massachusetts Institute of Technology, studied the temporary thermal expansion of two brands of silica brick with the results shown in Table 11.

TABLE 11

Brand	Average Total Linear Expansion at			
	300°,	600°,	900°,	1,200°,
	Per Cent.	Per Cent.	Per Cent.	Per Cent.
I. Reese & Sons	0.9	1.5	1.9	
Star	...	0.7	0.9	1.1

Practical experience shows that the thermal expansion of well-burned brick amounts to 3/16 to 1/4 in. per foot (1.6 to 2.1 per cent.) of which the greater part occurs below 600° C.

"In laying silica brick it is important to leave room for expansion;

[52] Experimental results hereinafter given indicate that this view is subject to certain qualifications.

[53] *Tonindustrie Zeitung*, vol. 25, p. 876 (1901); *Stahl und Eisen*, vol. 21, p. 772 (July, 1901).

[54] F. T. Havard: *Op. cit.*, p. 37.

[55] Massachusetts Institute of Technology Mining Department Thesis No. 226, 1902.

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thus in a reverberatory furnace for melting sulphide-copper ores, having 12-in. silica brick, ¼ in. is allowed to 1 ft. in the roof, 5/16 in. to 2 ft. in the bottom; the expansion space is occupied by cardboard when the brick is being laid."[56]

Effect of Rapid Temperature Change.—Closely connected with the expansion on heating is the inability of silica brick to withstand rapid changes of temperature. For this reason, both while being manufactured and later when put into use, heating and cooling must take place evenly and slowly. "By quick burning" (in the kilns) "brick may be easily expanded by double or treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick . . . In the early stages of firing the heat must be raised very slowly."[57] If cooled too rapidly, silica brick become friable, crack and spall. An underburned brick is less likely to spall than a well-burned one, but is weak, and has an excessive expansion, largely of a permanent character, when heated in the furnace.

Constitution

The constitution of silica brick and its relation to the phenomena of expansion have engaged the attention of many investigators. K. Endell[58] gives a historical sketch of the most important work done in this connection. In 1890, E. Mallard[59] discovered tridymite crystals in brick which had been in use a year and a half; this observation was confirmed in 1911 by Grum-Grzimailo.[60] The latter found that silica brick, when first made, consisted of grains of unaltered quartz surrounded by an apparently amorphous groundmass, which was held to be glass, together in some cases with varying amounts of tridymite; upon several months' use in the open-hearth furnace, the brick gradually and completely went over into a mass of interlocking twin crystals of tridymite. To avoid continual expansion in the furnace it was proposed to burn the brick in such a way as to effect the complete transformation to tridymite in the kilns.

Holmquist[61] discovered cristobalite in brick which had melted and hung down from the roof of the furnace as stalactites.

Endell[62] shows that the texture and constitution of silica brick are quite different before use, after 10 charges in the open-hearth furnace, after 50 charges, and again after having been melted. After the first burn, about two-thirds of the material was found to be converted into an

[56] H. O. Hofman: *General Metallurgy*, p. 368, McGraw-Hill Book Co., New York, 1913.

[57] K. Seaver: *Op. cit.*, p. 139.

[58] *Stahl und Eisen*, vol. 32, p. 392 (Mar., 1912).

[59] *Bulletin, Société française de minéralogie*, vol. 13, p. 172 (1890).

[60] *Stahl und Eisen*, vol. 31, pp. 224–226 (Feb., 1911).

[61] *Tonindustrie Zeitung*, vol. 35, p. 1325 (1911).

[62] *Stahl und Eisen*, vol. 32, p. 392–397 (Mar., 1912).

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apparently isotropic modification of silica, enclosing unaltered quartz grains of unequal size. This apparently isotropic form was at first believed to be glass, but later shown to be cristobalite.[63] Since the transformation of the original quartz was not complete in the initial burn, the conclusion was reached that only about two-thirds of the expansion takes place in the kilns, and the rest must occur in the furnace. In a brick which had been through 10 charges in the open-hearth furnace, the groundmass was found to be converted to a large degree into tridymite, while the larger quartz grains were transformed into the apparently isotropic condition (cristobalite). After about 50 charges, a good silica brick consisted almost wholly of interlaced tridymite crystals. A brick was examined in which a rind 2 or 3 cm. thick had been melted, while the rest of the piece showed no signs of fusion. Three zones could be distinguished. The one lying nearest the heat had been fully melted, and later, in cooling, crystallized out in part as cristobalite, which was surrounded by a yellow iron-containing glass. The birefringence and other crystal properties of cristobalite formed in this way were much more easily recognized than with cristobalite formed by the transformation of quartz. The colorless cristobalite grains were penetrated by pearly cracks and twinned in a very complicated manner. In the thermal microscope they became isotropic at 220°. Analysis of this fused zone showed a silica content of but 90 per cent.; vaporization of SiO_2 had presumably been favored by intermediate formation of silicon carbide. The central zone showed the typical constitution of a tridymite brick which still contains some cristobalite. As the outer zone was approached, grains of quartz appeared; the latter zone contained considerable unaltered quartz in a groundmass of cristobalite, and in constitution appeared like an unused brick. It had evidently been kept cool by the outside air.

Endell[64] studied also the changes in the constitution of quartzites after repeated burning in a porcelain kiln to 1,450° C. It was found that the finely divided impurities of the quartzite serve as inversion centers and hasten the transformations.

Rock heated three times was converted principally into cristobalite, with some unaltered quartz grains remaining. The latter went over into cristobalite on longer heating and caused a continual expansion. After 10 heatings, there were still a few unaltered quartz particles, and a small proportion of tridymite had appeared in the form of the characteristic wedge-shaped twin crystals. Quartzite burned four times in the porcelain kiln, and later heated 2 hr. to 1,600° was completely transformed into cristobalite.

Seaver and Klein[65] made quantitative determinations of the constitution of silica brick and quartzite burned repeatedly to 1,540° C.,

[63] K. Endell: *Stahl und Eisen*, vol. 33, p. 1856 (Nov., 1913).
[64] *Op. cit.*, p. 1856.
[65] K. Seaver: *Op. cit.*, p. 137.

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that temperature being maintained each time for about 40 hr. Their results are shown in Table 12.

TABLE 12

	Silica Brick			Quartzite	
	Burn			Burn	
	1, Per Cent.	2, Per Cent.	3, Per Cent.	1, Per Cent.	2, Per Cent.
Per cent. cristobalite	77	83	84	49	69
Per cent. quartz and silicates	23	17	16	51	31

No tridymite was to be expected in these tests, since that mineral is not known to be formed under any conditions above 1,470° C.

The relation of the constitution of silica brick to the expansion on heating is indicated by the facts that the volume increase in the transformation of quartz to cristobalite is 13.6 per cent.; quartz to tridymite, 16.7 per cent.

MICROSCOPIC STUDY AND EXPERIMENTAL DATA

MICROSCOPIC EXAMINATION OF QUARTZITES

The accompanying photomicrographs represent the texture of specimens of quartzite from the deposits most important in the American refractories industry. All of these quartzites consist essentially of individual quartz crystals of random orientation closely packed together and interlocking with sharp angles. No trace of a groundmass or binding material is to be seen, except in a few small and isolated areas, which do not appear in the figures. Small inclusions are visible in great number. The accessory minerals are extremely small in amount, and have not been studied.

Fig. 10 shows the Medina quartzite from Mount Union, Huntingdon Co., Pa. This consists essentially of a mass of interfering quartz crystals, each with a sand grain as a nucleus. The rounded or subangular outlines of the original sand grains are marked by a row of inclusions, and are perfectly distinct. The spaces between these are filled with secondary quartz which has attached itself to the original grains in crystalline continuity with them, so that the original quartz grain and the new quartz border have the same extinction positions and interference colors. The borders contain but few inclusions, and consequently are much clearer than the nuclei.

The Baraboo Quartzite from Ablemans, Wis., is shown in Fig. 11. The grain is much coarser than that of the other quartzites examined, and the line along which the interlocking individuals join is peculiarly jagged. Many of the grains show a marked undulatory extinction with crossed nicols; that is, the whole of the crystal is not extinguished at

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the same time, but a wave-like shadow passes over it on rotating the stage.

FIG. 10.

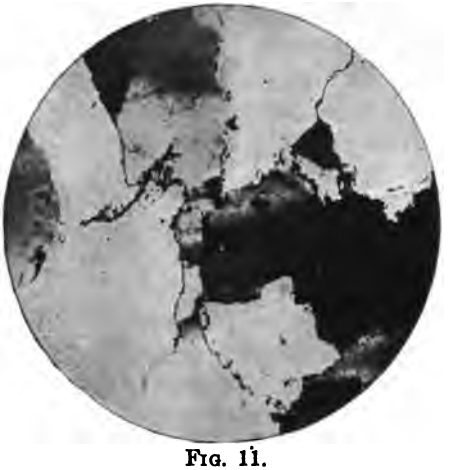

FIG. 11.

FIG. 10.—MEDINA QUARTZITE FROM MT. UNION, HUNTINGDON CO., PENNSYLVANIA (WHITEHEAD). THE FIELD SHOWS A MASS OF INTERFERING QUARTZ CRYSTALS EACH WITH A SAND GRAIN AS A NUCLEUS. THE OUTLINES OF THE ORIGINAL GRAINS ARE WELL SHOWN IN THE TWO CRYSTALS TO THE LEFT. CROSSED NICOLS. × 48.

FIG. 11.—BARABOO QUARTZITE FROM ABLEMANS, WISCONSIN (WHITEHEAD). UNDULATORY EXTINCTION IS SHOWN BY A GRAIN NEAR THE CENTER AND BY ONE NEAR THE TOP OF THE SECTION. CROSSED NICOLS. × 48.

FIG. 12.

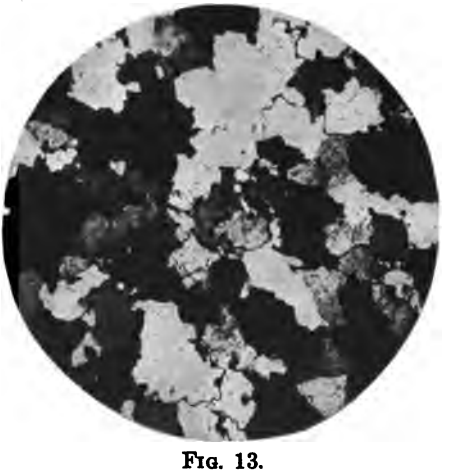

FIG. 13.

FIG. 12.—QUARTZITE FROM NORTHEASTERN ALABAMA, USED IN BRICK MAKING AT WYLAM, ALABAMA (WHITEHEAD). NOTE THE STRIATED APPEARANCE OF SOME OF THE GRAINS. CROSSED NICOLS. × 48.

FIG. 13.—QUARTZITE USED IN BRICK MAKING AT ANACONDA, MONT. (WHITEHEAD). CROSSED NICOLS. × 48.

The crystals of the Alabama quartzite contain numerous inclusions, arranged in nearly parallel lines, which with low magnifications appear under the microscope as fine striations. The Anaconda quartzite is extremely fine in grain.

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The textural relations above described, and illustrated in the photomicrographs, do not sustain the conclusion[66] that only quartzites containing a fine-grained groundmass or cement can be used in the manufacture of high-grade silica refractories.

In Figs. 10, 11, 12, 13, [67] quartz grains in extinction positions are shown by black areas; quartz grains in other positions appear as lighter areas. The small black dots are mostly inclusions.

Study of Silica Brick

Description of Test Brick

The conditions under which the test brick were made were devised by the engineering department of the Harbison-Walker Refractories Co., with the purpose of studying the effects of variations in grind and temperature of burning, and of repeated burning, upon the physical properties and constitution of silica brick. Accordingly, three lots of 9-in. brick (approximately 9 by 4½ by 2½ in. in size) were made up. These lots will be hereinafter referred to as "regular", "medium", and "fine grind" respectively. The regular-grind specimens were made by the usual method for 9-in. brick; those of medium grind were given the grind used for silica shapes, and pounded into loose-side molds, as silica shapes are treated. The material used for the fine grind was screened through an 8-mesh hand screen, returned to the pan and reground long enough to add and mix lime thoroughly; the brick were pounded into loose-side molds. Screen analyses are shown in Table 13.

Table 13.—*Screen Analyses of Test Brick*

Grind	Screen Numbers in Meshes per Linear Inch					
	Held on					Through
	8, Per Cent.	16, Per Cent.	20, Per Cent.	30, Per Cent.	40, Per Cent.	40, Per Cent.
Regular	15.3	13.4	4.6	5.4	4.8	56.5
Medium or shape	10.2	13.1	2.2	5.0	4.7	64.8
Fine		14.3	4.1	6.5	4.1	71.0

A number of brick from each lot were burned under the ordinary conditions of manufacture at three positions in the kiln, designated for convenience as positions *A*, *B* and *C*. The brick were burned a number of times, being under fire approximately 10 days each time, and specimens were set aside for study after each burning.

[66] See under *Texture*.

[67] Figs. 10 to 12 inclusive are from specimens furnished by the Harbison-Walker Refractories Co. Fig. 13 is from specimen furnished by the Anaconda Copper Mining Co.

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The temperatures attained on the various burns as measured by Orton-Seger cones are shown in Table 14.

TABLE 14.—*Temperature Measured by Orton-Seger Cones*

Number of Burn	Position		
	A	*B*	*C*
1	Cone 15	Cone 13	Cone 14
2	Cone 15	Cone 13	Cone 14
3	Cone 15	Cone 15	Cone 15
4	Cone 13	Cone 14	Cone 14
5	Cone 15	Cone 15	Cone 15
6	Cone 16	Cone 14	Cone 14

The fusion points of these cones are as follows:

Cone	Orton Scale, Degrees Centigrade	Seger Revised Scale,* Degrees Centigrade
13	1,390	1,380
14	1,410	1,410
15	1,430	1,435
16	1,450	1,460

* Advertisement in 1914 Calendar of *Tonindustrie Zeitung*

The fusion points in silica brick kilns probably average 70° lower than the figures given by the manufacturers,[68] but can not be told with exactness, since cones give very erratic results when used in such kilns.[69]

Physical Tests

Density and Porosity.—The bulk and apparent specific gravities and per cent. of open pore space of certain brick of the first burn are shown in Table 15. The term "apparent" specific gravity, is used in the sense defined earlier in this paper under the heading, "Properties of Silica Brick." The composition of the brick (hereinafter discussed) suggests that the value of the true specific gravity, of which no determinations were made, probably lies between 2.40 and 2.45. The per cent. of open pore space was calculated from the two specific gravities (see discussion under heading just mentioned).

From the figures given it appears that the greatest porosity is shown by the coarse (regular) grind, and the least by the medium (shape) grind.

[68] Hoffman: *Tonindustrie Zeitung*, vol. 35, pp. 1099–1100 (1911).
S. Geijsbeek: Melting Points of Pyrometric Cones under Various Conditions, *Transactions of the American Ceramic Society*, vol. 14, 849–871 (1912).

[69] K. Seaver: *Op. cit.*, p. 133.

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The data are insufficient to justify any conclusions as to the effect upon porosity of variations in burning temperature.

TABLE 15.—*Tabulated Data: Density and Porosity*

		First Burn					
Mark	Grind	Burned to Cone	Bulk Specific Gravity	Apparent Specific Gravity	Open Pore Space, Per Cent. of Volume	Open Pore Space, Mean Per Cent. of Volume	Open Pore Space, Mean Per Cent. of Volume
83–A–1	Regular	15	1.678	2.259	25.7	26.1	
279–A–1	Regular	15	1.655	2.254	26.5		
69–C–1	Regular	14	1.637	2.229	28.5	28.5	
228–B–1	Regular	13	1.671	2.251	25.7	25.7	26.8
585–A–1	Medium	15	1.770	2.247	21.3	21.8	
601–A–1	Medium	15	1.748	2.249	22.3		
679–C–1	Medium	14	1.723	2.232	22.7	22.7	
443–B–1	Medium	13	1.839	2.277	19.2	19.2	21.2
1,148–A–1	Fine	15	1.757	2.272	22.7	22.5	
1,149–A–1	Fine	15	1.765	2.272	22.3		
854–C–1	Fine	14	1.725	2.242	23.2	23.2	
997–B–1	Fine	13	1.774	2.264	21.7	21.7	22.5

Cross-Breaking and Crushing Tests

In the transverse tests the brick were laid edgewise with a span of 6 in. and the load applied at midspan. Modulus of rupture was calculated from the formula $R = \frac{3\,Wl}{2\,bd^2}$, in which l is the distance between supports in inches, b the breadth and d the depth of the specimen in inches, and W the load in pounds at which failure occurred. Compression tests were made on half bricks resulting from the transverse tests. The bricks were bedded flatwise on blotting paper in the testing machine to secure a uniform bearing surface. The crushing strength was obtained by dividing the crushing load in pounds by the mean area in square inches of the two surfaces of the brick.

Unfortunately, but three brick of each lot were available for these tests, a number insufficient to permit the determination of the mean strengths with the proper degree of accuracy. The proposed specifications[70] of the American Society for Testing Materials direct that at least five specimens shall be used in transverse and compression tests upon building brick, and the same requirement should undoubtedly apply to the testing of refractory brick.

The results obtained are shown in Tables 16, 17, and 18.

[70] *Proceedings of the American Society for Testing Materials*, vol. 13, p. 284 (1913).

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TABLE 16.—*Sample Log and Data Sheet*
Cross-Breaking and Crushing Tests

Cross-Breaking Span, 6 In.						Crushing: Half Brick Crushed Flat					
Mark	Breadth of Brick, In.	Depth of Brick, In.	*W*	*R*	Mean *R*	Crushing Area, Square Inches			*W*	*C*	Mean *C*
						1	2	Mean			
83–A–1	2.48	4.45	4,970	910		19.80	17.58	18.69	51,900	2,780	
279–A–1	2.46	4.45	4,830	890	920	19.54	17.97	18.75	45,600	2,430	2,680
289–A–1	2.52	4.43	5,260	960		16.46	16.16	16.31	46,100	2,830	
585–A–1	2.66	4.73	7,010	1,060		21.75	21.33	21.54	101,000	4,690	
601–A–1	2.66	4.71	7,450	1,140	1,050	21.18	20.24	20.71	120,500	5,820	4,780
715–A–1	2.68	4.71	6,320	960		20.26	20.00	20.13	77,000	3,830	

W = load causing failure in pounds.
R = modulus of rupture, pounds per square inch.
C = compressive strength, pounds per square inch.

TABLE 17.—*Collected Data*
Cross-Breaking and Crushing Tests

First burn — Brick Tested Edgewise 6-in. Span

Mark	Grind	Burned to Cone	Cross-Breaking Load in Pounds	Modulus of Rupture			Crushing Strength, Brick Laid Flat		
				Modulus Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean	Crushing Strength, Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean
83–A–1	Regular	15	4,970	910			2,780		
279–A–1	Regular	15	4,820	890	920	3	2,430	2,680	6.4
289–A–1	Regular	15	5,260	960			2,830		
135–B–1	Regular	13	3,240	600			2,680		
228–B–1	Regular	13	3,630	670	690	10	2,390	2,730	8.5
327–B–1	Regular	13	4,280	800			3,120		
69–C–1	Regular	14	4,010	730			2,210		
102–C–1	Regular	14	4,140	750	770	5	2,050	2,100	3.3
156–C–1	Regular	14	4,520	820			2,040		
585–A–1	Medium	15	7,010	1,060			4,690		
601–A–1	Medium	15	7,450	1,140	1,050	6	5,820	4,780	14.0
715–A–1	Medium	15	6,320	960			3,830		
443–B–1	Medium	13	4,990	800			5,430		
445–B–1	Medium	13	4,310	690	910	24	3,890	4,680	11.0
476–B–1	Medium	13	7,760	1,230			4,710		
679–C–1	Medium	14	5,430	830			4,340		
680–C–1	Medium	14	5,610	860	920	11	4,930	4,820	6.7
744–C–1	Medium	14	7,050	1,070			5,180		
1,148–A–1	Fine	15	5,790	930			7,270		
1,149–A–1	Fine	15	9,450	1,500	1,170	19	4,360	5,880	17.0
1,151–A–1	Fine	15	6,770	1,070			6,010		
997–B–1	Fine	13	5,360	860			6,300		
998–B–1	Fine	13	6,180	990	910	6	5,280	5,840	6.3
999–B–1	Fine	13	5,560	890			5,940		
854–C–1	Fine	14	3,270	520			5,180		
934–C–1	Fine	14	2,840	450	790	52	5,040	5,250	3.4
1,053–C–1	Fine	14	9,010	1,410			5,520		

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TABLE 17.—*Collected Data.—Continued*

Second burn

Mark	Grind	Burned to Cone	Cross-Breaking Load in Pounds	Modulus of Rupture			Crushing Strength, Brick Laid Flat		
				Modulus, Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean	Crushing Strength, Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean
275–A–2	Regular	15–15	5,820	1,080			2,230		
288–A–2	Regular	15–15	5,160	980			2,750		
290–A–2	Regular	15–15	3,480	620	910	15	2,990	2,680	8.6
293–A–2	Regular	15–15	5,140	940			2,760		
138–B–2	Regular	13–13	5,620	1,040			2,770		
218–B–2	Regular	13–13	4,730	860	1,010	10	2,540	2,760	5.4
330–B–2	Regular	13–13	6,070	1,120			2,970		
10–C–2	Regular	14–14	5,420	980			2,690		
14–C–2	Regular	14–14	4,730	880	920	3	2,490	2,680	4.6
110–C–2	Regular	14–14	4,950	920			2,850		
567–A–2	Medium	15–15	10,440	1,600			5,780		
599–A–2	Medium	15–15	10,730	1,630	1,680	5	7,010	6,030	11.0
628–A–2	Medium	15–15	11,800	1,810			5,290		
467–B–2	Medium	13–13	9,350	1,460			7,030		
472–B–2	Medium	13–13	10,390	1,630	1,310	24	5,200	6,270	11.0
579–B–2	Medium	13–13	5,360	840			6,590		
572–C–2	Medium	14–14	10,340	1,570			5,600		
658–C–2	Medium	14–14	10,620	1,620	1,680	7	5,130	5,570	5.2
699–C–2	Medium	14–14	12,150	1,860			5,970		
824–A–2	Fine	15–15	4,440	680			7,870		
848–A–2	Fine	15–15	4,830	770			7,850		
850–A–2	Fine	15–15	12,450	1,950	1,270	43	6,700	6,890	14.0
970–A–2	Fine	15–15	10,550	1,670			5,150		
961–B–2	Fine	13–13	5,420	860			5,610		
1,019–B–2	Fine	13–13	6,770	1,080	1,070	13	7,390	6,290	12.0
1,042–B–2	Fine	13–13	7,820	1,280			5,870		
805–C–2	Fine	14–14	13,650	2,160			8,560		
836–C–2	Fine	14–14	8,050	1,260	1,590	24	7,150	7,830	6.2
1,012–C–2	Fine	14–14	8,730	1,360			7,780		

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TABLE 17.—*Collected Data.—Continued*

Third burn

Mark	Grind	Burned to Cone	Cross-Breaking, Load in Pounds	Modulus of Rupture			Crushing Strength, Brick Laid Flat.		
				Modulus, Pounds per Square Inch	Mean	Average Deviation, Per Cent. of Mean	Crushing Strength, Pounds Per Square Inch	Mean	Average Deviation, Per Cent. of Mean
39–A–3	Regular	15–15–15	9,260	1,680			3,320		
53–A–3	Regular	15–15–15	4,060	740			3,000		
144–A–3	Regular	15–15–15	5,770	1,040	1,200	25	2,940	3,110	4.5
249–A–3	Regular	15–15–15	7,330	1,340			3,170		
174–B–3	Regular	13–13–15	7,190	1,310			3,000		
202–B–3	Regular	13–13–15	5,460	990	1,010	20	2,590	2,830	5.7
280–B–3	Regular	13–13–15	3,890	720			2,910		
66–C–3	Regular	14–14–15	6,290	1,170			2,650		
153–C–3	Regular	14–14–15	6,380	1,160	1,110	7	3,490	2,990	11.0
179–C–3	Regular	14–14–15	5,470	990			2,840		
427–A–3	Medium	15–15–15	4,910	770			6,570		
428–A–3	Medium	15–15–15	6,550	1,030			6,520		
439–A–3	Medium	15–15–15	7,840	1,230	1,210	25	6,890	6,550	2.8
501–A–3	Medium	15–15–15	11,680	1,820			6,220		
431–B–3	Medium	13–13–15	3,040	470			4,860		
568–B–3	Medium	13–13–15	10,520	1,610	1,320	43	6,220	6,380	17.0
622–B–3	Medium	13–13–15	12,280	1,880			8,070		
513–C–3	Medium	14–14–15	8,280	1,270			4,620		
672–C–3	Medium	14–14–15	12,790	1,950	1,580	16	4,520	4,830	7.0
674–C–3	Medium	14–14–15	9,770	1,510			5,360		
1,028–A–3	Fine	15–15–15	5,780	910			7,390		
1,046–A–3	Fine	15–15–15	6,340	1,000			6,920		
1,057–A–3	Fine	15–15–15	6,900	1,100	1,240	28	8,000	7,060	9.0
1,066–A–3	Fine	15–15–15	12,130	1,930			5,920		
846–B–3	Fine	13–13–15	13,900	2,180			8,680		
882–B–3	Fine	13–13–15	10,510	1,650	1,730	17	8,610	8,120	9.0
906–B–3	Fine	13–13–15	8,470	1,350			7,010		
880–C–3	Fine	14–14–15	6,800	1,070			7,070		
974–C–3	Fine	14–14–15	3,300	520	680	38	6,240	6,600	11.0
1,074–C–3	Fine	14–14–15	2,790	440			5,860		

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TABLE 18.—*Summary*

Cross-Breaking and Crushing Tests

Burn	Burned to Cone	Modulus of Rupture, Brick Tested Edgewise, 6-in. Span						Crushing Strength, Brick Laid Flat					
		Regular Grind		Medium Grind		Fine Grind		Regular Grind		Medium Grind		Fine Grind	
		Mean	a.d.*	Mean	a.d.	Mean	a.d.	Mean	a.d.	Mean	a.d.	Mean	a.d.
1	15	920	3	1,050	6	1,170	19	2,680	6	4,780	14	5,880	17
	14	770	5	920	11	790	52	2,100	3	4,820	7	5,250	3
	13	690	10	910	24	910	6	2,730	8	4,680	11	5,840	6
2	15–15	910	15	1,680	5	1,270	43	2,680	9	6,030	11	6,890	14
	14–14	920	3	1,680	7	1,590	24	2,680	5	5,570	5	7,830	6
	13–13	1,010	10	1,310	24	1,070	13	2,760	5	6,270	11	6,290	12
3	15–15–15	1,200	25	1,210	25	1,240	28	3,110	5	6,550	3	7,060	9
	14–14–15	1,110	7	1,580	16	680	38	2,990	11	4,830	7	6,600	11
	13–13–15	1,010	20	1,320	43	1,730	17	2,830	6	6,380	17	8,120	9

* a.d. represents the average deviation in per cent. of mean.

Discussion of Cross-Breaking and Crushing Tests

Average Deviation and Index of Uniformity.—It is highly desirable not only that the brick should be mechanically strong, but also that different specimens of the same lot should be of uniform strength. For this reason, in Table 19, obviously the lot of brick of "regular grind, mark C–3," is a more satisfactory material, so far as cross-breaking is concerned, than the "fine grind, mark A–2," notwithstanding the fact that the mean modulus of rupture of the former is 1,110 lb. per square inch, of the latter, 1,270 lb. per square inch. Since, plainly, mean values in themselves are insufficient as a basis of comparison for the strength of such brick, the "average deviation" of the single observations from the mean (as developed in the principles of precision of measurements) has been calculated as a measure of uniformity of the different specimens, for each lot of brick tested.[71]

The deviation of single observations from the mean is due to two factors: (1) lack of precision of the measurements; and (2) lack of uniformity of different specimens of the material. In the series of tests herein described, variations due to the former factor amount to less than 2 per cent. and are practically negligible as compared with variations due to the latter. For this reason, the "average deviation" can be considered a true measure of uniformity. A very low "average deviation" is an indication of a high degree of

[71] Similarly, "average deviations" are given by the Mechanical Engineering Department of the Massachusetts Institute of Technology, in reporting results obtained in testing the strength of timber.

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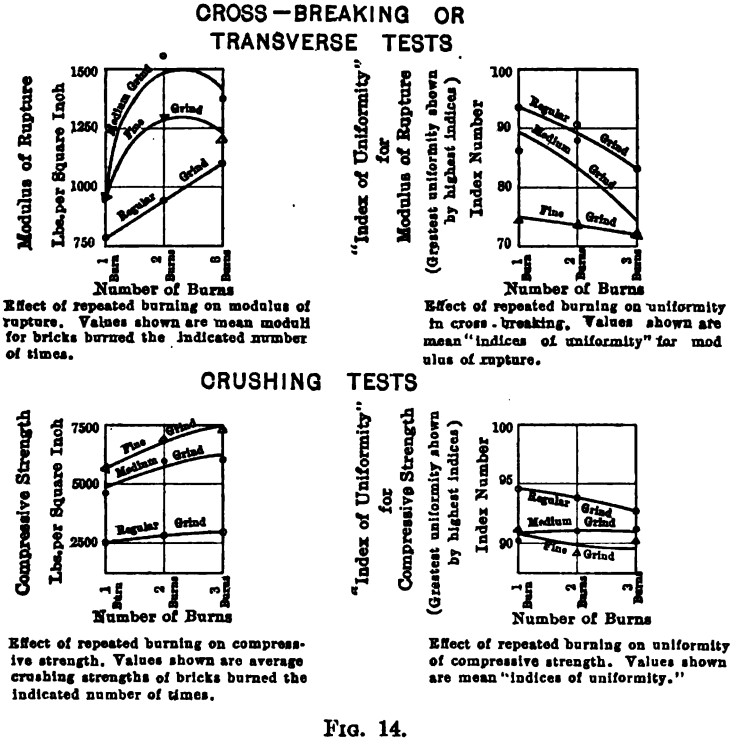
concordance in the values found for different specimens of the same material; a high "average deviation" indicates a lack of concordance.

The method of derivation of the "average deviation" for two specific cases is shown in Table 19.

TABLE 19

Grind	Mark	Modulus of Rupture	Mean Modulus	Deviation from Mean	Average Deviation or "a.d."	a.d. in Per Cent. of Mean
Fine.....	824–A–2	680		590		
Fine.....	848–A–2	770		500		
Fine.....	850–A–2	1,950	1,270	680	540	43.0
Fine.....	970–A–2	1,670		400		
Regular.....	66–C–3	1,170		60		
Regular.....	153–C–3	1,160	1,110	50	80	7.2
Regular.....	179–C–3	990		120		

In the graphical representation of the relative uniformity of different lots of specimens (Fig. 14) use has been made of an "index of uniformity,"



Effect of repeated burning on modulus of rupture. Values shown are mean moduli for bricks burned the indicated number of times.

Effect of repeated burning on uniformity in cross-breaking. Values shown are mean "indices of uniformity" for modulus of rupture.

Effect of repeated burning on compressive strength. Values shown are average crushing strengths of bricks burned the indicated number of times.

Effect of repeated burning on uniformity of compressive strength. Values shown are mean "indices of uniformity."

FIG. 14.

calculated from the average deviation (a.d.) thus: "Index of uniformity" = 100 − (a.d. in per cent. of mean). In the above table, the "index

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of uniformity" for modulus of rupture is 57 on this scale for the first lot, 92.8 for the second.

Effective Ultimate Strength.—In comparing the strength of different lots of brick, their relative uniformity must be considered. As shown above, the mean ultimate strength is not a satisfactory basis for comparison. Since the "average deviation" is a measure of uniformity, it seems logical to consider the value obtained by subtracting the average deviation from the mean strength, as a more practical comparative standard. This value may be called the "effective ultimate strength;" and to this, rather than to the mean, the factor of safety should be

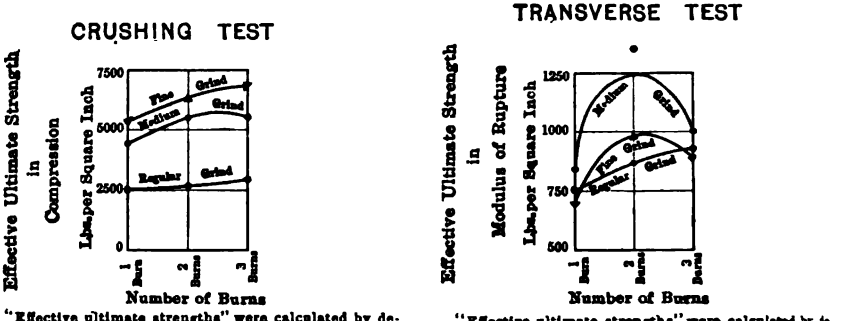

FIG. 15.

applied in calculating a safe working load for materials of which the load-carrying capacity is an important consideration. The effect of repeated burning upon the "effective ultimate strength," as so defined, is shown in Fig. 15.

Effect of Variation in Method of Manufacture.—The effects of the variables considered upon the strength of the test brick are represented graphically in Figs. 14 and 15.

The mean modulus of rupture of medium-grind brick is greatest; of regular-grind, least. The burning temperatures which produce the strongest brick in cross-breaking are as follows:

	First Burn	Second Burn
Regular-grind.....	Cone 15	Cone 15
Medium-grind.....	Cone 15	Cone 14 and 15
Fine-grind.....	Cone 15	Cone 14

Bricks of regular grind increase in cross-breaking strength after each burning; after the first burn, specimens burned at cone 15 are considerably stronger than those burned at cone 13 or 14, after the second burn this is no longer true. Modulus of rupture of medium- and fine-grind brick, in most cases, reaches a maximum on the second burn.

Crushing strength of regular-, medium- and fine-grind brick are to each other approximately as 41:84:100. There is an increase in strength after

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each burn; a slight increase for regular grind, a considerable increase for medium and fine grinds.

Uniformity of strength of a given lot of brick diminishes with repeated burning; slightly in compression, considerably in cross-breaking. The most uniform brick are those of regular grind; fine-grind brick are fairly uniform in crushing, very non-uniform in cross-breaking strength.

The figures indicate that the medium-grind brick is the most satisfactory in point of strength, the regular-grind least so. While the compressive strength of the medium-grind is somewhat lower than that of the fine-grind, its cross-breaking strength is greater and it is a much more uniform material. In cross-breaking, the fine-grind shows little if any advantage over the regular-grind, for while its mean strength is high, it lacks uniformity; in compression, it is far stronger than the regular-grind.

There is, in all cases, an advantage in burning the brick a second time; for the medium and fine grinds, a third burn is usually harmful, decreasing the cross-breaking strength considerably, and increasing the compressive strength very slightly. Brick of regular grind increase in strength on each burn up to the third, slightly in compression, considerably in cross-breaking.

The alternate expansions and contractions incurred in repeated burning may tend to cause the formation of minute cracks in the brick and to have an unfavorable effect upon their strength. For that reason it is probable that stronger brick would be produced by maintaining the maximum temperature a greater length of time during a single burn than by resorting to repeated burning.

Microscopic Study of Constitution of Brick [72]

Method in General

Quantitative determinations of the proportions of quartz, tridymite and cristobalite in specimens of the various lots of brick were made by the Rosiwal micrometric method.[73] The measurements were all made upon powdered material, the minerals being identified through comparison of indices of refraction by immersion in liquids of definite indices. The Zeiss mechanical stage, and a lens system giving a magnification of about 390 diameters, were used.

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[72] The microscopic work was performed under the direction of Dr. C. H. Warren, without whose instruction and assistance the writer would not have ventured to undertake a study of this character.

[73] A. Johannsen: *Manual of Petrographic Methods*, p. 291. McGraw-Hill, New York, 1914. Lincoln and Rietz: Determination of the Relative Volumes of the Components of Rocks by Mensuration Methods, *Economic Geology*, vol. 8, pp. 120–139 (1913).

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The indices of refraction of quartz are 1.544 to 1.533; cristobalite, 1.484 to 1.487; tridymite, 1.469 to 1.473;[74] calcium silicates, higher than those of quartz.[75] Separate portions of each powder were studied in two liquids whose indices of refraction were respectively 1.495 and 1.479. The higher-refracting quartz and compounds not identified were distinguished in the former liquid from the lower-refracting cristobalite and tridymite. A large part of the higher-refracting material had the refractive index and general properties of quartz.

The proportion of tridymite was determined by immersion of the powder in a liquid with refractive index of 1.479, which is between those of tridymite and cristobalite, but extremely close to them. On account of the latter fact, variations in room temperature caused some difficulty, since the index of the liquid was diminished 0.0005 to 0.0006 for each degree temperature rise. All material with an index lower than 1.479 was reported as tridymite, and the amount of cristobalite obtained by difference.

Sampling

A sample of about 300 grams from each lot of test specimens (100 grams from each brick of the lot) was crushed in a small Blake crusher and a disk grinder, to pieces approximately 1/16 in. in diameter. After cutting down to ±20 grams with a split shovel, the sample was reground until all passed through a 140-mesh sieve. For the microscopic study, that fraction was reserved (about 5 per cent. of the whole) which passed through a 240-mesh and was caught on a 260-mesh sieve. The size of these grains as determined microscopically varied from 0.05 to 0.06 mm. in diameter.

Identification

The principal optical property used in identifying the minerals was the index of refraction, by means of the Becke effect. The single grains were commonly made up of two, sometimes all three minerals, and the correct positions of the boundaries between tridymite and cristobalite could be told only by crossing the nicols and rotating the stage.

The more highly refracting material (chiefly quartz in the first burns) was easily recognized by its high relief in the liquid used, and by well-defined Becke lines. Tridymite was distinguished by its low index, low double refraction, and by the lath-shaped and wedge-shaped outlines of its crystals. Identification was confirmed by the facts that the lath-shaped sections had parallel extinction and negative elongation, and that

[74] See Table 2.

[75] Rankin and Wright: The Ternary System CaO–Al_2O_3–SiO_2, *American Journal of Science*, ser. 4, vol. 39, p. 74 (1915).

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the index of refraction of the faster ray, as determined by the immersion method in sodium light, was 1.469.[76]

The low relief, the index of refraction, and the extremely low double refraction, which could usually be detected only with the aid of the sensitive tint plate, served to identify cristobalite. In the liquid with index 1.479, the Becke lines were faint, but their directions of motion could be

Quartz shown in black; cristobalite dotted

FIG. 16.—SKETCHES OF GRAINS OF POWDERED BRICK. FROM BRICK OF REGULAR GRIND, BURNED ONCE AT CONE 14. × 218 APPROXIMATELY.

detected upon close observation so long as the room temperature was kept at the proper point.

The lack of homogeneity of the grains led to the introduction of certain unavoidable quantitative errors. Quartz could, as a rule, be recognized even on the interior of the grains by the clearness of the Becke lines and the high relief; yet numerous small particles of residual quartz

Quartz shown in black; cristobalite dotted
tridymite cross-hatched

FIG. 17.—SKETCHES OF GRAINS OF POWDERED BRICK. FROM BRICK OF REGULAR GRIND, BURNED FOUR TIMES AT CONE 14–14–15–14. × 218 APPROXIMATELY.

were often so mingled with the cristobalite that an accurate quantitative separation of the two was impossible. The double refraction of tridymite was the only means of determining with exactness the boundaries between tridymite and cristobalite, but in sections of certain orientation could not be used. The presence of numerous microscopic cracks caused anomalous effects, and in many cases interfered with the Becke tests.

The texture of representative grains is shown in Figs. 16 and 17.

Precision

A coarser powder than that used was found to give less accurate results. The very fine portion, which went through the 260-mesh sieve,

[76] See C. N. Fenner: Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, p. 351 (1913).

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was exceedingly difficult to study. Since the latter fraction comprised about 85 per cent. of the original powder, it is evident that any great variation in its composition from that of the study sample will very seriously affect the results. For that reason, check determinations were made in a few cases of the amount of quartz in both the fraction which had passed through the 260-mesh sieve, and that which had been caught upon it. Results so obtained averaged 2 per cent. lower in the former case than in the latter, a difference which lies within the error of observation.

Errors introduced into the tridymite determinations by discarding the very fine material are probably serious, yet it was not found possible to obtain concordant results with this portion of the powder. As shown in Fig. 19, tridymite forms most rapidly in the fine-grained groundmass of the brick, and comparatively slowly in the larger grains. A much larger proportion of the groundmass than of the grains will undoubtedly crush down in the grinding so as to pass through a 260-mesh sieve. For that reason, the amount of tridymite in the fraction of the powder reserved for study, is probably considerably less than in the brick itself.

TABLE 20.—*Results of Microscopic Study—Tabulated Data*
Constitution of Test Brick

Lot	Mark	Grind	Burned to Cone	Number of Burns	Constitution in Per Cent. by Volume		
					$n > 1.495$	$n < 1.479$	$n > 1.479$, < 1.495
					Quartz (+Silicates)	Tridymite	Cristobalite
A–1	277, 279	Regular	15	1	26	4	70
C–1	102, 156	Regular	14	1	25	4	71
B–1	135	Regular	13	1	40	1	59
A–1	715, 730	Medium	15	1	24	6	70
A–1	1,150, 1,151	Fine	15	1	26	4	70
C–1	934, 1,052	Fine	14	1	26	5	69
A–2	275, 288, 290, 293	Regular	15–15	2	19		
A–3	39, 53, 144, 249	Regular	15–15–15	3	16	19	65
C–3	66, 153, 179	Regular	14–14–15	3	14	20	64
B–3	174, 202, 280	Regular	13–13–15	3	18	10	
A–3	427, 428, 439, 501	Medium	15–15–15	3	13	17	70
A–3	1,028, 1,066, 1,057, 1,046	Fine	15–15–15	3	13.	17	70
C–4	12	Regular	14–14 15–14	4	12	30	58
C–6*	74	Regular	14–14–15 14–15–14	6	14	44	42
B–6*	326	Regular	13–13–15 14–15–14	6	12	45	43

* Analysis of No. 74 and No. 326 by V. Dolmage.

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The figures given as the proportions of quartz and tridymite in the test brick represent in each case results of measurements taken on 150 to 250 grains. As a check on the work, percentages were calculated for each 75 to 80 grains. The first two results usually agreed within 3 to 4 per cent.; if not, an additional 80 to 100 grains were studied. The figures so obtained are believed to be truly comparative as showing the direction and approximately the velocity of the transformations occurring, notwithstanding the constant errors due to the difficulties above-mentioned, and to the personal equation. The latter errors are believed to be small, since results obtained on the same powder by Mr. Victor Dolmage (who kindly assisted with many of the measurements) and by the writer, usually checked within 3 per cent.

Effect of Variations in Grind and Burning Temperature, and of Repeated Burning, upon Constitution of the Brick

The effect of repeated burning on the constitution of brick of regular grind, burned several times, is shown in Fig. 18. The analyses of this brick after repeated burnings are shown in Table 21.

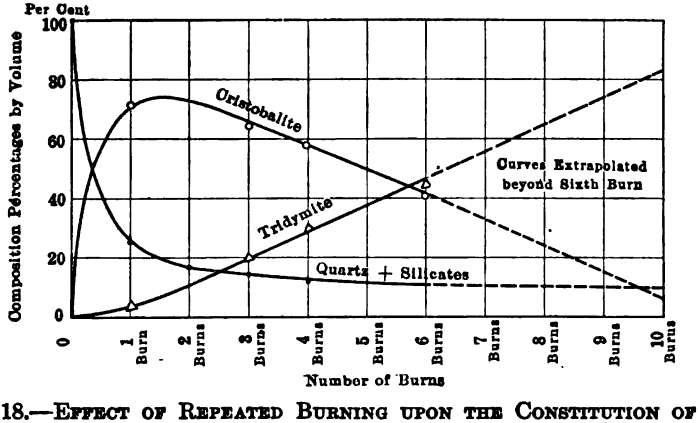

FIG. 18.—EFFECT OF REPEATED BURNING UPON THE CONSTITUTION OF SILICA BRICK. BRICK OF REGULAR GRIND. BURNING TEMPERATURE, CONE 14–14–15–14–15–14.

TABLE 21

Mineral	Volume, Per Cent.					
	Number of Burns					
	1	2	3	4	5	6
Quartz (+Silicates).....	25	....	14	12	....	14
Cristobalite.....	71	....	64	58	....	42
Tridymite.....	4	....	20	30	....	44

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It is evident that in the first burn the larger part of the quartz is transformed into cristobalite, and that on repeated burning the remaining quartz is very slowly transformed.[77] The cristobalite at first very slowly and later more rapidly, inverts to tridymite. These results are apparently in accordance with the conclusions of Grum-Grzimailo and Endell, that silica brick heated a sufficient length of time at the proper temperature go slowly but completely over into tridymite.

Time and temperature of burning appear to be the controlling factors. Bricks burned a single time at cones 14 and 15 show essentially the same amount of transformation; those burned at cone 13 show considerably less. The effect of variations in grind is comparatively slight.

Study of Thin Sections

Examination of thin sections cut from certain specimens of test brick show that the inversions proceed most rapidly in the groundmass, most slowly in the larger grains. This is illustrated in Fig. 19, a sketch made from thin section of brick of fine grind, burned three times at cone 15. The groundmass and the rims of the grains are made up of cristobalite and tridymite, with a little residual quartz; the interior of the grains consists of cristobalite and quartz, with a very little tridymite.

FIG. 19.—SKETCH MADE FROM THIN SECTION OF BRICK OF FINE GRIND, BURNED THREE TIMES AT CONE 15.

Photomicrographs of the thin sections are shown in Figs. 20 to 23 inclusive. The dark areas represent principally cristobalite, the lighter areas, quartz and tridymite. From a comparison of the first two figures with the second two, the increased amount of tridymite in the later burns is at once evident.

THE RELATION OF THE CONSTITUTION OF SILICA BRICK TO THE PHYSICAL PROPERTIES

The Effect of the Addition of Lime

It has already been shown that fineness of grind is not the controlling factor in the formation of tridymite, since that mineral evidently forms nearly as rapidly in the brick of coarse (or regular) grind, as in those of finer grinds. A seeming contradiction exists between this fact and the fact that the inversion to tridymite takes place more rapidly in the fine-grained groundmass of the brick than in the larger quartzite fragments (see Fig. 19).

The relative concentrations of lime in the groundmass of brick of

[77] Compare with results obtained by Seaver, shown in Table 12.

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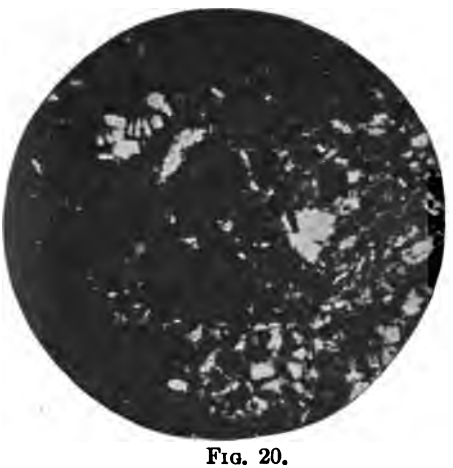

FIG. 20.

FIG. 21.

FIG. 20.—SILICA BRICK OF REGULAR GRIND, BURNED ONCE AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

The white areas represent residual quartz grains of irregular size and shape, surrounded by a dark field of cristobalite. Little, if any, tridymite is to be seen.

FIG. 21.—SILICA BRICK OF FINE GRIND, BURNED ONCE AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

The figure shows residual quartz grains surrounded by cristobalite. There are a few narrow, elongated white areas in the field, which may represent tridymite.

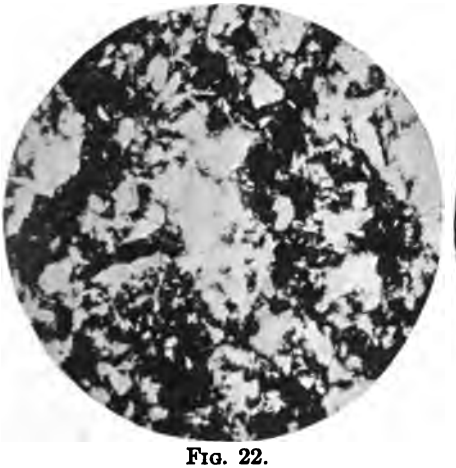

FIG. 22.

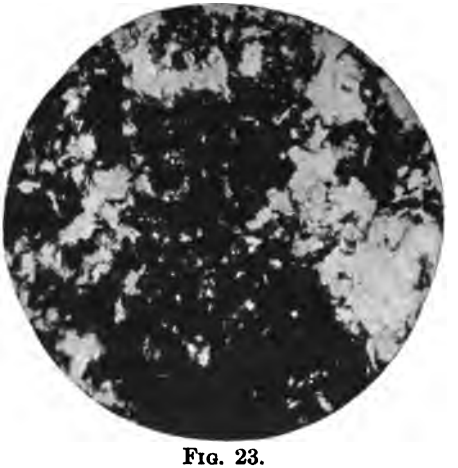

FIG. 23.

FIG. 22.—SILICA BRICK OF REGULAR GRIND, BURNED THREE TIMES AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

Tridymite and residual quartz are shown by the white areas; cristobalite by the dark areas.

FIG. 23.—SILICA BRICK OF REGULAR GRIND, BURNED THREE TIMES AT CONE 15 (WHITEHEAD). CROSSED NICOLS. × 120.

The larger white areas indicate places in which the conversion to tridymite is nearly complete. The smaller white areas in the darker portion of the field represent residual quartz and tridymite, surrounded by cristobalite (shown in black). Figs. 22 and 23 are taken from different parts of the same thin section.

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various grinds probably has some bearing in this connection. It will be recalled that in the manufacture of the brick, 2 per cent. of lime is added to ground quartzite containing about 2 per cent. of impurities (mainly Al_2O_3 and Fe_2O_3). Since all of this enters the groundmass, the proportion of lime therein is much higher than 2 per cent.

The greatest concentration of lime in the groundmass will occur in that brick with the least proportion of fines (*i.e.*, the regular grind) in which, therefore, conditions will be most favorable for the formation of flux compounds during the burning. The catalytic action of the flux compounds probably explains the observed inversion of cristobalite to tridymite, which as Fenner has shown, does not form except in the presence of a flux. According to this point of view, an increase in the amount of lime in the brick should accelerate the formation of tridymite.

Specific Volumes of the Silica Minerals

As an aid in interpreting the results of the microscopic study, use has been made of the specific volume curves of Fig. 24, based upon the following data:

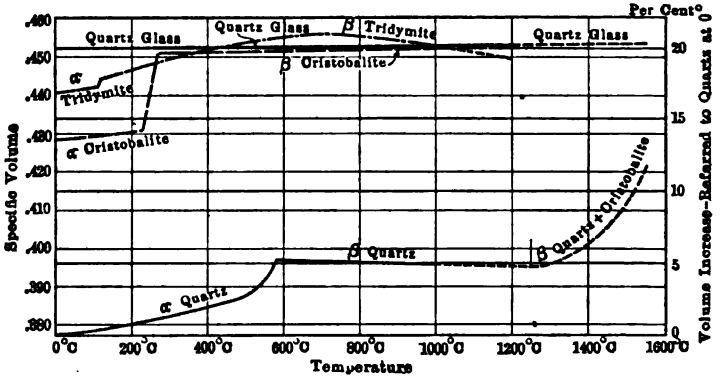

FIG. 24.—SPECIFIC VOLUMES OF THE SILICA MINERALS AND QUARTZ GLASS.

The quartz curve is a reproduction of that shown in Fig. 4, taken from the work of Day, Sosman and Hostetter.

The specific volumes at 20° of quartz glass, tridymite and cristobalite were calculated from their specific gravities, using the values shown below:[78]

	Specific Gravity
Quartz glass	2.210 (Dana, Endell)
Tridymite	2.270 (Fenner)
Cristobalite	2.333 (Fenner)

[78] See Table 3.

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The quartz-glass curve was obtained by computing the volume expansion from the linear expansion[79] as given by LeChatelier, and applying these figures to the specific volume at 20°.

The curve for cristobalite is based upon the work of Rieke and Endell, and of LeChatelier. Both show that the expansion of cristobalite from 0° to 230° is slight, and that the volume changes at 230° to 270° is considerable,[80] due to the α to β cristobalite inversion. Rieke and Endell find that the specific volume and coefficient of expansion of cristobalite above 300° are nearly the same as those of quartz glass; the specific volumes, as computed from the expansion data of LeChatelier[79] are considerably lower than those of quartz glass. The view of Rieke and Endell is taken in plotting the cristobalite curve of Fig. 24.

The tridymite curve was calculated from the thermal expansion as given by LeChatelier (Fig. 3). The specific volume of tridymite from 500° to 1,000° is seemingly a little greater than that of quartz glass. While this is probably incorrect, the difference between the specific volumes of the two materials is doubtless small.

The curves of Fig. 24 have been used hereinafter in discussing the probable effect of the changes in constitution upon certain properties of silica brick. It is believed that the inaccuracies of these curves are merely slight errors of degree; and that, while conclusions drawn from them can not be numerically exact, they will serve to indicate the general trend of the changes considered.

The Phenomena of Expansion

Permanent Expansion, Temporary or Thermal Expansion, and the Effect of Rapid Temperature Changes.—The permanent expansion of silica brick after the first burning (in which ±5 per cent. of tridymite and ± 70 per cent. of cristobalite are formed) amounts to 10 to 11 per cent. by volume. The gradual decrease of quartz and increase of tridymite caused by repeated burning, theoretically requires a slight additional expansion after each burn until equilibrium is reached, since the complete transformation of quartz to cristobalite is accompanied by a volume increase of 13.6 per cent.; quartz to tridymite, 16.8 per cent.

At 1,250° C. the specific volumes of cristobalite and tridymite are approximately 14 to 15 per cent. greater than that of quartz. Since the residual quartz of silica brick gradually goes over into the other forms of silica at furnace temperatures, an underburned brick will expand considerably after being placed in use.

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[79] Fig. 3.
[80] Fig. 3; also Fig. 5.

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Tridymite expands upon heating much less than does cristobalite. Its specific volume at 20° is 2.8 per cent. greater than that of cristobalite; at 1,000°, nearly the same as that of cristobalite. The gradual transformation of cristobalite into tridymite, shown to occur upon repeated burning of silica brick, probably takes place also in furnace linings. This inversion is accompanied by little, if any, change in volume at furnace temperature.

From the data of Fig. 24, specific volumes at different temperatures were computed for two materials assumed to consist of mixtures of quartz, tridymite, and cristobalite; expansion curves derived from these figures are shown in curves I and II, Fig. 25. Curve I is based upon an assumed composition of 25 per cent. quartz, 70 per cent. cristobalite, 5 per cent. tridymite; curve II on one of 12 per cent. quartz, 43 per cent. cristobalite,

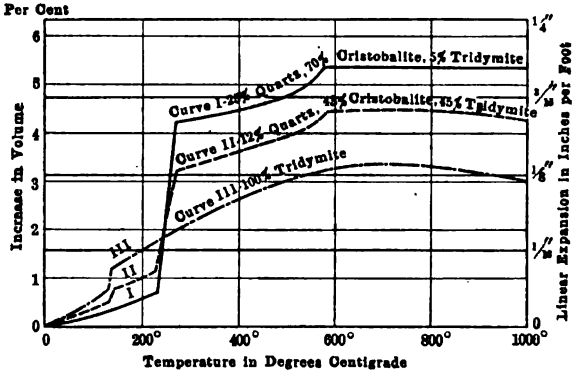

FIG. 25.—THEORETICAL EXPANSION CURVES FOR MIXTURES OF THE SILICA MINERALS.

Composition of Curve I corresponds closely to brick of regular grind *burned once* at cone 14.

Composition of Curve II corresponds closely to brick of regular grind *burned six times* at cone 14–14–15–14–15–14.

42 per cent. tridymite. These figures represent approximately the compositions of brick of regular grind, after one and after six burnings respectively. Curve III is taken from the expansion data of LeChatelier.[81]

Fig. 25 is intended merely to indicate the probable character of the expansion curve of silica brick, and of the effect upon the same of the observed changes in constitution. Obviously, the theoretical curves shown will not correspond precisely with those that would be obtained by actual measurements, being affected by inaccuracies in the data from which they were computed, and, in addition, modified by changes in

[81] Fig. 3.

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porosity of the brick, the action of flux compounds, and the lag in expansion due to the temperature gradient between the exterior and interior of the brick.

These curves indicate that the temporary or thermal expansion of silica brick is decreased by repeated burning, and that an all-tridymite brick would have an expansion amounting to about 60 per cent. of that of an ordinary commercial brick, as represented in Curve I.

It is well known that silica brick crack or spall badly if subjected to rapid temperature changes. The phenomenon is probably due chiefly to the rapid expansion at 230° to 270°, accompanying the α to β cristobalite inversion. This effect is diminished by repeated burning. The calculated volume increase at 230° to 270° amounts to 3.5 per cent. for brick burned once, 2.0 per cent. for brick burned six times. An all-tridymite brick should show very little tendency to crack upon rapid heating or cooling, since the only sharp break in the tridymite expansion curve, occurring at 130°, is attended by a volume increase of but 0.3 per cent.

For several reasons, therefore, a large proportion of tridymite in silica brick is desirable, provided the brick are not to be heated continuously above 1,470° C.

1. Tridymite has the least thermal or temporary expansion of the three silica minerals.

2. Tridymite has the greatest specific volume of the three minerals, and can not show any permanent expansion after repeated burning below its melting point.

3. A tridymite brick could probably be subjected to fairly rapid changes of temperature with little danger of cracking.

Comparison of curves I and II seems to show that, while repeated burning produces very desirable effects, the advantages derived up to the sixth burn are entirely disproportional to the additional expense involved, on account of the extreme slowness with which tridymite is formed. Moreover, if used continuously at a temperature exceeding 1,470°, a tridymite brick would slowly revert to cristobalite.

Since the time element appears to be the most important factor in the formation of tridymite, it appears that long-continued burning at the proper temperatures should be more satisfactory than repeated burning. The maximum temperature is maintained 40 hr.[82] in the burning of silica brick; maintaining this temperature for 2½ weeks should presumably give rise to the same changes in constitution as 10 separate burnings under the ordinary conditions of manufacture, and produce brick consisting almost wholly of tridymite (see Fig. 16).

[82] K. Seaver: *Op. cit.*, p. 136.

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Burning Temperature

In order to make a brick containing the maximum proportion of tridymite, the burning temperature should be that at which the inversion to that mineral proceeds most rapidly. It appears that this must occur below 1,470°, since tridymite is not known to form under any conditions above that point. While the inversion proceeds more rapidly at cones 14 and 15 than at cone 13, no conclusions can be drawn from the data herein given as to the most favorable temperature for burning.

Suggestions for Future Study

It seems desirable that the theoretical advantages of a high-tridymite brick should be investigated by comparing the thermal expansion of an ordinary commercial brick with that of a brick high in tridymite. The effect of rapid temperature changes might be compared by means of spalling tests.

If it should be proven that the production of a tridymite brick would be advantageous, the possibility of making such a brick on a commercial basis for certain purposes might well be considered. A small furnace, in which close temperature regulation is possible, should preferably be employed; and the results thereby obtained, if promising, should later be tried out on a larger scale.

The temperature at which the formation of tridymite proceeds most rapidly in silica brick, and the inversion-velocity at that temperature, are as yet undetermined. The possibility of accelerating the inversion by a slight increase in the amount of lime used or by the addition of some other catalyzing material, is worth considering.

It is not impossible that by long-continued heating at the proper temperature, with possibly a slight additional amount of lime, a tridymite brick might be produced on a commercial basis. The results of tests so far made, however, indicate that this is extremely doubtful.

SUMMARY

A brief review is given of the literature pertaining to the silica minerals and the silica refractories.

Microscopic study of the quartzites most important in the American refractories industry shows that they consist essentially of interlocking quartz crystals of random orientation, without a groundmass or cement. This is not in accordance with Wernicke's conclusion that no quartzites are suitable for the manufacture of high-grade refractories except those

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which consist of quartz grains in a cement of amorphous silica or cryptocrystalline quartz.

Physical tests and microscopic studies were made of silica brick manufactured under varying conditions of grind, and burned one to four times at slightly varying temperatures. The grinds considered were (1) "regular," or coarse, (2) "shape," or medium, and (3) fine; the burning temperatures ranged from cone 13 to cone 16.

In all cases, a second burning increases the strength of the brick; a third burning has a harmful effect upon the strength in some cases; in others, causes little change. Brick of shape grind are the strongest, those of regular grind the weakest. With a single burning, a temperature of cone 15 produces a stronger material than cone 13 or 14. Uniformity of strength of different specimens of a given lot diminishes upon repeated burning. The most uniform brick are those of regular grind; the least uniform, those of fine grind. Attention is called to the insufficiency of average values as a proper basis for the comparison of strength of non-uniform materials, and for practical purposes the use of certain derived values, termed the "index of uniformity" and the "effective ultimate strength" is suggested.

Upon repeated burning, the quartz, of which the brick is made, is transformed through cristobalite into tridymite. By microscopic methods, estimates were made of the degree of transformation in various bricks. The figures obtained, while not numerically exact, indicate the general trend of the changes considered. The powder studied from brick burned a single time under the ordinary conditions of manufacture consists by volume of approximately 25 per cent. quartz + silicates, 70 per cent. cristobalite and 5 per cent. tridymite; after a sixth burn the composition is approximately 12 per cent. quartz + silicates, 43 per cent. cristobalite, and 45 per cent. tridymite. The inversions proceed more rapidly in the groundmass of the brick than in the larger grains.

It is considered desirable that silica brick should contain a large proportion of tridymite because (1) tridymite has the least thermal expansion of the silica minerals; (2) it will show no permanent expansion after repeated burning; (3) a tridymite brick could probably be subjected to fairly rapid changes of temperature with little danger of injury.

The advantages in these respects theoretically derived up to the sixth burn are believed to be wholly disproportional to the additional expense involved.

SUPPLEMENTARY DATA

Spalling Tests

General Discussion.—It has been shown in the preceding pages that a relation probably exists between the amount of cristobalite in a silica

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brick, and its tendency to spall upon rapid change of temperature; and that since the percentage of cristobalite decreases on repeated burning, the spalling tendency should likewise diminish. It was desired to confirm this conclusion through experimental data.

In spalling tests as ordinarily conducted, brick are alternately heated and cooled rapidly a definite number of times between fixed temperature limits; they are usually cooled in the air, but sometimes by plunging into water. The loss of weight in per cent. is taken as a measure of the spalling. This method is not altogether satisfactory, since it does not indicate the internal condition of the brick, which may possibly weaken considerably in the test without much spalling, or lose surface spalls without much weakening; and if the hot test pieces are water-cooled, the formation of steam in the pores may disrupt the brick and lead to inaccurate conclusions.

For the purposes of this study it was decided to make use of a milder heat treatment than that usually employed; one sufficiently severe to cause internal cracks, yet not severe enough to cause spalls to chip off. For each lot of brick studied, the cross-breaking strength was determined upon a certain number of specimens which had been thus treated, and also upon an equal number which had not. The loss in mean strength, expressed in per cent., is considered to represent the "comparative spalling tendency."

Spalling Test in Detail.—The test was carried out as follows: The brick to be subjected to the heat treatment were placed in a gas-fired test kiln, 16 at a time, and heated at a uniform rate of 15° C. per hour (so as to cause no injury to the specimens during the heating) until 600° C. was attained. They were kept at that temperature for 3 hr. and at the end of that time were withdrawn and placed on edge on inverted steel pallets about 2 ft. apart, in a place protected from drafts, and allowed to cool in the air of the laboratory. They were not moved until the following day.

This treatment had the desired effect. No pieces spalled off, nor were the brick visibly shattered; yet when any two brick were struck together they gave a dead ring.

In Table 22 are shown the cross-breaking strengths before and after this heat treatment, of brick of regular grind burned a number of times varying from one to six. The burning temperatures are those shown in Table 14.

In Table 23, these values are summarized and the loss in strength, assumed to represent the comparative spalling tendency, is shown. Figs. 26 and 27 are graphic representations of the data given in Table 23.

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TABLE 22.—*Cross-Breaking Tests Made to Determine Spalling Tendency*

Brick Tested on Edge; Supports 6 In. Apart; Load Applied at Midspan

Cross-Breaking Strength of Brick *Not* Subjected to Heat Treatment						Cross-Breaking Strength of Brick Subjected to Heat Treatment					
No. of Burns	Mark	Lot	Cross-breaking, Load in Pounds	Modulus of Rupture		No. of Burns	Mark	Lot	Cross-breaking, Load in Pounds	Modulus of Rupture	
				Modulus, Pounds per Square Inch	Mean					Modulus, Pounds per Square Inch	Mean
1	83	A	4,970	910		1	268	A	1,730	310	
1	279	A	4,820	890		1	295	A	2,480	443	
1	289	A	5,260	960		1	299	A	2,290	412	
1	69	C	4,010	730	848	1	56	C	2,080	373	339
1	71	C	4,790	877		1	73	C	1,900	341	
1	102	C	4,140	750		1	97	C	780	138	
1	156	C	4,520	820		1	166	C	2,000	357	
2	275	A	5,820	1,080		2	18	A	3,430	605	
2	288	A	5,160	980		2	88	A	2,360	421	
2	290	A	3,480	620		2	125	A	3,120	564	
2	293	A	5,140	940		2	240	A	2,030	365	
2	138	B	5,620	1,040		2	196	B	2,440	434	
2	218	B	4,730	860	919	2	22	B	1,600	288	444
2	330	B	6,070	1,120		2	324	B	2,350	420	
2	335	B	3,260	606		2	351	B	2,360	425	
2	10	C	5,420	980		2	96	C	2,310	414	
2	14	C	4,730	880		2	108	C	2,300	419	
2	33	C	5,650	997		2	151	C	2,730	489	
2	110	C	4,950	920		2	318	C	2,740	490	
3	39	A	9,260	1,680		3	70	A	2,850	505	
3	53	A	4,060	740		3	123	A	2,900	518	
3	144	A	5,770	1,040		3	263	A	2,390	425	
3	249	A	7,330	1,340		3	303	A	2,620	468	
3	82	B	6,250	1,100		3	51	B	2,770	495	
3	174	B	7,190	1,310	1,120	3	208	B	3,230	572	489
3	202	B	5,460	990		3	267	B	2,800	494	
3	280	B	3,890	720		3	276	B	3,020	531	
3	66	C	6,290	1,170		3	32	C	3,040	538	
3	78	C	6,480	1,170		3	152	C	1,960	355	
3	153	C	6,380	1,160		3	155	C	2,730	491	
3	179	C	5,470	990		3	171	C	2,650	478	
4	41	A	5,160	947		4	149	A	3,000	547	
4	85	A	6,070	1,100		4	247	A	1,830	325	
4	91	A	3,900	691		4	305	A	2,800	500	
4	238	A	5,600	1,010		4	310	A	1,720	306	
4	188	B	5,170	923		4	89	B	3,040	549	
4	219	B	4,750	848	883	4	140	B	2,150	384	458
4	291	B	4,220	764		4	195	B	2,280	406	
4	329	B	5,480	990		4	350	B	2,050	366	
4	8	C	6,790	1,200		4	22	C	2,950	534	
4	170	C	4,420	789		4	77	C	2,580	471	
4	175	C	3,300	594		4	130	C	2,850	521	
4	313	C	4,150	737		4	148	C	3,230	586	
5	67	A	4,290	769		5	19	A	2,450	434	
5	104	A	4,650	830		5	76	A	2,270	407	
5	147	A	7,360	1,320		5	132	A	3,020	543	
5	158	A	5,730	1,040		5	173	A	2,900	516	
5	186	B	6,270	1,150		5	80	B	3,010	528	
5	215	B	4,920	886	1,050	5	216	B	3,350	601	558
5	224	B	6,000	1,060		5	308	B	3,800	691	
5	230	B	5,350	961		5	342	B	2,600	466	
5	257	C	5,950	1,090		5	122	C	2,820	510	
5	262	C	6,920	1,240		5	145	C	3,840	688	
5	306	C	5,950	1,080		5	165	C	3,410	611	
5	338	C	6,650	1,200		5	297	C	3,860	695	
6	4	A	5,620	1,010		6	87	A	3,570	638	
6	201	A	5,230	935		6	126	A	3,170	573	
6	266	A	4,780	850		6	157	A	3,250	572	
6	325	A	5,600	1,010		6	159	A	2,810	504	
6	136	B	5,580	1,010		6	30	B	2,610	468	
6	211	B	5,750	1,030	951	6	185	B	2,640	471	522
6	250	B	5,000	908		6	232	B	1,940	349	
6	255	B	3,680	668		6	320	B	2,750	479	
6	6	C	5,150	916		6	91	C	3,300	600	
6	13	C	3,980	704		6	94	C	2,990	549	
6	34	C	6,590	1,190		6	160	C	2,870	517	
6	150	C	6,560	1,180		6	317	C	3,010	549	

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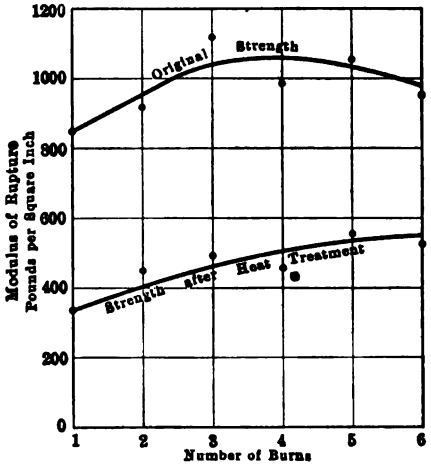

FIG. 26.—VARIATIONS IN THE CROSS-BREAKING STRENGTH OF BRICK OF REGULAR GRIND UPON REPEATED BURNING, AND THE EFFECT UPON THE STRENGTH OF A CERTAIN HEAT TREATMENT, CONSISTING OF SLOW HEATING TO 600° C. FOLLOWED BY COOLING IN THE AIR.

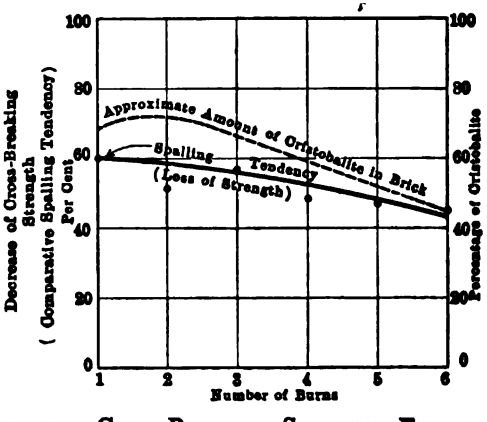

FIG. 27.—DECREASE IN CROSS-BREAKING STRENGTH EXPRESSED IN PER CENT. DUE TO HEAT TREATMENT DESCRIBED. THIS PERCENTAGE DECREASE IS ASSUMED TO REPRESENT THE COMPARATIVE SPALLING TENDENCY.

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TABLE 23.—*Summary of Cross-breaking Tests Showing Loss of Strength due to Heat Treatment*

Burn	Modulus of Rupture, Pounds per Square Inch		Loss of Strength due to Heat Treatment	
	Before Heat Treatment	After Heat Treatment	Pounds per Square Inch	Per Cent. (= Comparative Spalling Tendency)
1	848	339	509	60.0
2	919	444	475	51.8
3	1,120	489	631	56.2
4	883	458	425	48.1
5	1,050	558	492	46.9
6	951	552	429	45.1

Results.—The experimental data verify the conclusion that the spalling tendency of silica brick diminishes on repeated burning. In Fig. 27 there is a marked correspondence between the curve representing approximately the cristobalite content of the brick and that representing the spalling tendency. The latter, however, does not decrease as rapidly as the former. It is evident that, while reburning the brick diminishes

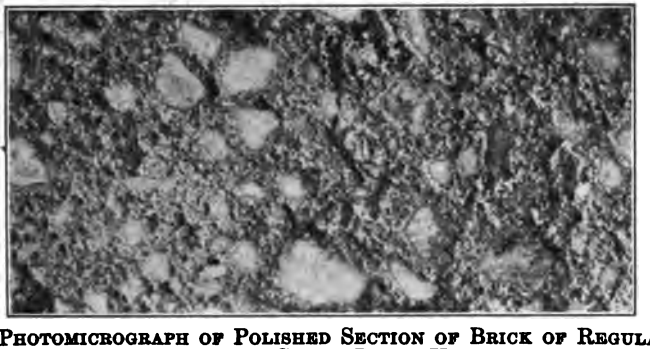

FIG. 28.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF REGULAR GRIND, BURNED ONCE IN SILICA BRICK KILN. × 4.

the spalling tendency, the change therein from 60 per cent. on a first burn to 45 per cent. after a sixth is too slight to be of any commercial importance.

A number of first-quality fire-clay brick, subjected to the same heat treatment, lost about 4.5 per cent. in strength. This result agrees with the well-known fact that the spalling tendency of clay brick, at the temperature of this test, is but a fraction of that of commercial silica brick.

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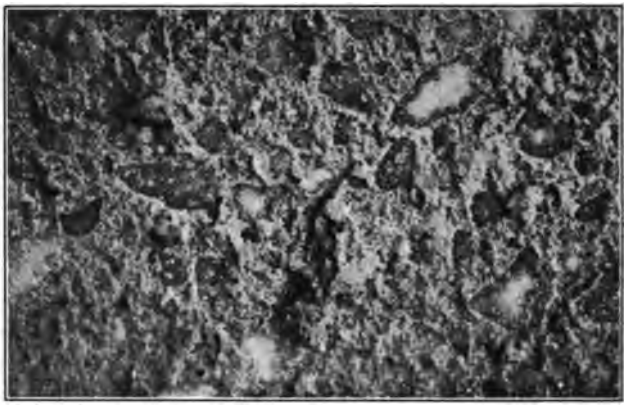

FIG. 29.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF REGULAR GRIND, BURNED SIX TIMES. × 4.

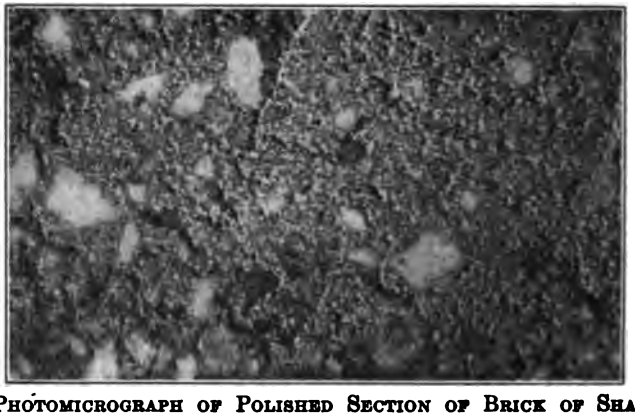

FIG. 30.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF SHAPE GRIND, BURNED ONCE. × 4.

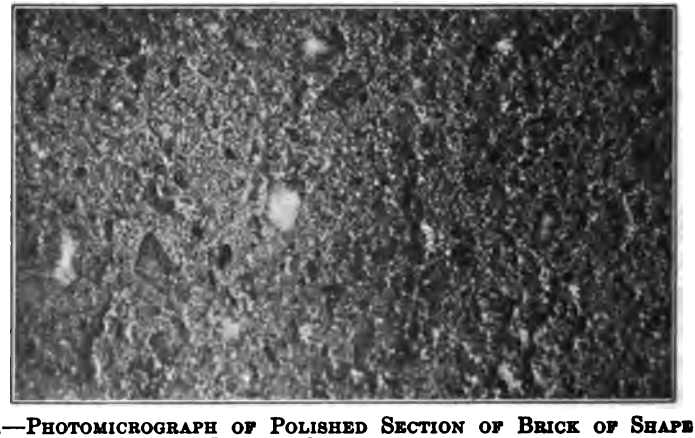

FIG. 31.—PHOTOMICROGRAPH OF POLISHED SECTION OF BRICK OF SHAPE GRIND, BURNED SIX TIMES. × 4.

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Changes in Texture of Brick Due to Burning

The photomicrographs shown in Figs. 28 to 31 bring out very clearly the change in the texture of the brick brought about by repeated burning. After a single burning the larger grains of quartzite appear nearly white; after six burnings the grains stand out much more distinctly in the pictures and have become darker and apparently more dense. A number of grains are seen in which the center is still white, while the margins are altered in the manner described.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Matte Granulation at Herculaneum, Mo.

BY S. PAUL LINDAU, B. S.,* HERCULANEUM, MO., AND HENRY B. SMITH,†
RIVERMINES, MO.

(New York Meeting, February, 1917)

THREE years ago it was decided by the management to granulate the matte that is produced in the smelter of the St. Joseph Lead Co. at Herculaneum, Mo., thereby doing away with a large amount of labor in handling the matte and in subsequent crushing. On Mar. 1, 1915, the granulator was put in operation and up to Jan. 1, 1916, 18,735 tons had been granulated. The four blast furnaces have a daily output of about 280 tons of pig lead and 100 tons of matte assaying about 11.0 per cent. Pb. It is expected that projected improvements in roasting methods will eliminate more sulphur from the charge, and cut down the matte-fall to about one-half of the present figure.

The blast furnaces are tapped into movable forehearths or settlers, 6 ft. by 4 ft. by 22 in., the slag overflowing into 27-cu. ft. slag cars which are hauled to the dump by electric locomotives. Previous to the adoption of granulation, the matte was tapped from the forehearths into small slag pots which were dumped after cooling. The matte cakes were broken with sledges, and transported to the crushing plant for fine crushing and screening. Here the matte was reduced to ¼-in. size before being fed to the roaster bins. A Wedge and a Holthoff mechanical furnace are used for preroasting before sintering on the Dwight and Lloyd machines. The matte is returned to the furnaces as an iron flux until the gradually accumulating copper content reaches about 7 per cent., after which it is put aside to be concentrated in a small copper blast furnace, the resultant high-grade matte being sold.

When the matte does not settle readily from the slag, two settling devices are often used, the second one being a large catch-pot, which gathers any matte that may overflow from the forehearth, the slag then overflowing from the catch-pot into the slag cars. When full of matte, the catch-pot is removed, allowed to cool, and the contents broken and crushed.

With the present plan of granulating, the matte is tapped into 6-cu. ft. iron ladles. These are transported to the granulating plant by a 5-ton

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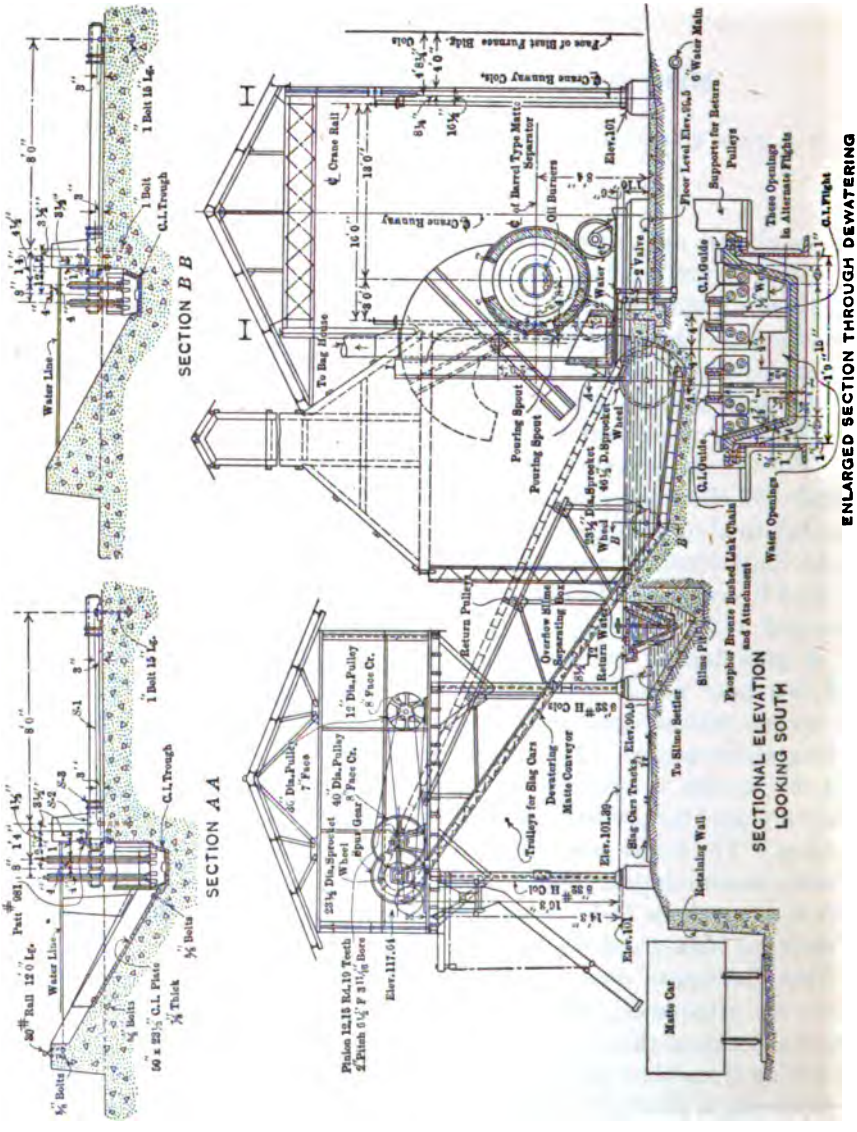
overhead electric traveling crane, the runway of which extends the full length of the blast-furnace building. A cylindrical container receives the matte from the ladles. This container is a specially designed Traylor



ENLARGED SECTION THROUGH DEWATERING CONVEYOR TROUGH

FIG. 1.—MATTE GRANULATING PLANT OF THE ST. JOSEPH LEAD CO., HERCULANEUM, MO.

copper converter minus the tuyères and with a 20-in. opening at one end for the flue and oil burners. It measures 77 by 120 in. inside, is lined with magnesite brick 9 in. thick and can be tilted by means of a 50-hp. motor. A steel flue 18 in. in diameter, leading from one end, conducts

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any fume to the bag house. Just under this flue are placed two oil burners for heating. Generally one is found sufficient to keep the matte in a molten state. The consumption of 18° to 20° Bé. crude oil is 250 gal. per 24 hr., or 2 to 2½ gal. of oil per ton of matte granulated. Air at 30 lb. pressure is supplied to the burners for atomizing the oil.

The granulation is accomplished by pouring the molten matte through two superimposed flat jets of water shooting horizontally into a concrete tank lined with cast-iron plates (Fig. 1). The stream of matte is disintegrated into small shot-like particles before reaching the body of water. The stream of molten matte is accurately directed on to the jets of water by a so-called pouring box, the spout of which is 6 in. above and 12 in. in front of the upper nozzle. This box is lined with common brick and the matte is poured directly into it by revolving the container. The pour holes are slots 2 in. wide by 8 in. high.

A dewatering drag conveyor removes the granulated particles of matte from the tank. The floor of the tank slopes down to the trough in which the conveyor operates, at an angle of about 30°. It is placed to one side, out of the line of the streams from the nozzles. It elevates the matte over the slag track and discharges into standard-gage railroad cars, which are weighed, and their contents sampled and emptied into the roaster bins.

In the end of the concrete tank opposite the nozzles, a V-shaped settling box takes the overflow water. This box prevents the loss of the coarser slime, while the finer material settles out in a series of settling tanks, one overflowing into the other. An excelsior filter finally clears the water before it enters the circulating pond. There are two series of settling tanks, one in use while the other is being cleaned out.

Operation and Construction

The matte in the cylindrical container is kept fluid by one or both of the two oil burners before mentioned. The burners are placed so that the flame shoots slightly upward in the container. An oxidizing flame is used, though there would be some practical advantages if a reducing flame could be employed. It was found impossible, however, to maintain the required temperature with a reducing flame. If the matte remains in the container too long, an oxidized scum is formed which interferes with the pouring.

The matte is discharged from the barrel container through an opening in the side, 2 by 8 in., provided with a spout delivering into the pouring box which rests on the concrete wall of the tank directly over the water nozzles. This pouring box is made of sheet iron and is lined with common firebrick; inside dimensions are 18 in. by 18 in. by 3 ft. The matte tends to chill and build up in the box, but a narrow passage about 3 in.

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wide is easily maintained along the path of the stream. This box in turn discharges through an opening 2 by 8 in., the matte being directed by a cast-iron spout so that it meets the horizontal jets of water at an angle of 70° to 80°. All the matte should be broken up and be prechilled before striking the main body of water. As a rule the stream of matte is broken up by striking the upper jet while the lower jet insures further cooling. It was found that if some of the matte missed the jets or if the rate of pouring was too rapid, thus preventing the thorough prechilling of the matte, some of the semimolten particles united again into large lumps, frequently causing explosions in the tank. Likewise, too rapid pouring will promote the formation of large granules up to 1 in. in diameter, which are detrimental to good roasting. Too slow pouring allows the matte to chill before leaving the pouring box, thereby causing the opening in the spout to gradually freeze. The right speed was obtained after a little experimentation. A stream of matte as large as 2½ in. in

FIG. 2.—GRANULATION TANK, SHOWING JETS OF WATER, POURING BOX AND DRAG CONVEYOR.

diameter where it strikes the jets has been granulated satisfactorily. A hot matte makes a better product for roasting, because it is more uniform and finer. A good product is that of which 75 per cent. passes a 1/16-in. screen. The average rate of pouring is 3.2 cu. ft. of matte per minute.

The water jets are delivered through rectangular nozzles, the openings being ⅜ by 3¾ in. with the nozzles 5 in. apart. The supply of water is 100 gal. per minute under a head of 40 ft. The granulating tank is constructed of concrete, 17 ft. long and 7½ ft. wide, the floor being plated with 1-in. cast-iron plates, at a 30° slope into the conveyor trough. This degree of slope has been found sufficient to cause the granulated particles to run into the trough. The drag conveyor is driven

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by a 10-hp. motor at a speed of 30 ft. per minute and runs in a sectional cast-iron trough 4 in. deep and 15 in. wide. A very small amount of water, about 5 per cent., passes over with the matte into the railroad cars, drainage being assisted by notching the conveyor flights alternately in the center and on the ends, thus allowing the entrained water to escape and flow back into the tank. The tail sprocket wheel and the idler sprocket wheel (Fig. 1) are both under water. The shafts of these wheels extend through stuffing boxes in the sides of the tank, to the bearings on the outside.

Some lead settles out of the matte while in the container, about 30 to 60 pigs of 65 lb. each, according to the condition of the furnaces, being poured out daily. In case lead goes over and is granulated with the matte, it manifests itself by a sputtering and popping on the surface of the water. After the matte has been poured down to the lead level, the operator turns the container backward and pours the lead into a ladle, whence it is molded into pigs.

To prevent metal losses, the fumes are caught by a swinging hood which fits over the charge opening and connects with the blast-furnace flue leading to the bag house. The hood may be swung back so as to uncover the charge opening, when the crane is ready to pour a ladle of matte into the barrel container.

This installation requires only two men per shift for operation, one to operate the container and one the crane. The forehearth tappers at the furnaces attach the crane hook to the ladles. Under the old method of handling the matte by hand-pots, the cost was 87 c. per ton, which included hand-breaking. Crushing and screening amounted to 56 c. per ton in addition, which brought the total cost to $1.43 per ton. Granulation costs only 75 c. per ton, which makes the total saving of $1.43 − 0.75 = $0.68 per ton of matte granulated.

Summary

Matte granulated per 24 hr., tons	100
Rate of pouring, cubic feet per minute	3.2
Oil consumed per 24 hr., gallons	250
Oil per ton of matte granulated, gallons	2.5
Water used per minute (under head of 40 ft.), gallons	100

Screen Analysis on Granulated Matte

Inch-Mesh	Per Cent.
On ¼	5
On 3/16	4
On ⅛	14
On 1/16	35
Through 1/16	42
	100

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Grain Growth Phenomena in Metals

BY ZAY JEFFRIES, B. S., MET. E.,* CLEVELAND, O.

(New York Meeting, February, 1917)

THE object of the present paper is to enlarge somewhat on the general principles advanced in my discussion[1] of Mathewson and Phillips' article on The Recrystallization of Cold-Worked Alpha Brass on Annealing.[2] It will also serve as an acknowledgment of Prof. Howe's[3] most important remarks on my contribution. In this paper the writer has adopted Prof. Howe's suggestion to substitute the term "germinative temperature" for "critical temperature for grain growth." Instead of being an exact or certain temperature it should be considered that the germinative temperature phenomenon exists throughout a small temperature range.

Factors Influencing Fast Growth Phenomena

The development of grains of macroscopic size at the germinative temperature should be considered the extreme condition of a general rule. When considered in this light it is not strange that the development of these large grains should be the exception and not the rule. The principal factors influencing the operation of the laws of fast growth temperature are as follows:

1. Rate of heating.

In the original paper,[1] the germinative temperature was defined as "the minimum temperature at which two adjacent grains can coalesce to form one larger grain, provided that this larger grain will have sufficiently increased its power of attack, to enable it to absorb adjacent grains which cannot coalesce with each other. Time is always to be understood as a factor governing the first stage of grain growth." If the time of heating is short, the germinative temperature is raised. The formation of large grains in metals may occur at relatively high temperatures when the rate of heating is rapid. It is to be understood that in such cases, grain growth

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[1] *Bulletin* No. 113, p. 987 (May, 1916).

[2] *Bulletin* No. 109, p. 1 (January, 1916).

[3] Prof. Howe's paper will be printed in December *Bulletin*.

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must be very rapid, the germinative grains having the power to absorb the inert grains before the latter can coalesce with one another. The rapidity of absorption of the inert grains by the germinative grains is at times almost beyond comprehension. In one instance, the author calculated that 37,500,000 individual grains were absorbed by one germinative grain in 10 sec. The grain growth took place at such a temperature that the inert grains would have coalesced freely with one another, had time been given.

A further increase in rate of heating will prevent the formation of coarse grains. Other influences of change of rate of heating were considered in the original paper.

Figs. 1 and 2 show visually the effect of rate on the formation of large grains. Fig. 1 represents a section of metal held at the fast growth temperature for 30 min. This is a sample of compressed metal heated with electric current in which the temperature gradient was the cause of the selective grain growth. The photograph is at a magnification of 5 diameters and the grains can readily be seen macroscopically. Fig. 2 represents a section of the same metal in which the temperature was

FIG. 1.

FIG. 2.

FIG. 1.—A SECTION OF METAL HELD AT THE GERMINATIVE TEMPERATURE FOR 30 MIN. × 5.

FIG. 2.—SECTION OF THE SAME METAL AS IN FIG. 1, IN WHICH THE TEMPERATURE WAS RAISED QUICKLY TO A POINT FAR ABOVE THE GERMINATIVE TEMPERATURE RANGE, AND HELD THERE FOR 30 MIN.

raised quickly to a point far above the germinative temperature range, and held at this temperature for 30 min. Some of the larger individual grains in Fig. 1 occupy an area equal to about 1,000 average grains in Fig. 2. (The lines in Fig. 2 are cracks which formed in the specimen during mounting.) The mean grain size of the fine-grained areas in Fig. 1 is very much less than the mean grain size in Fig. 2. The grain size at the center of Fig. 1 represents about the normal grain size just above the fast growth temperature. The grain size shown in Fig. 2 is approximately the equilibrium grain size for a temperature several hundred degrees above the fast growth temperature.

Fig. 3 (18 diameters) shows the large grains formed at the germinative temperature in the process of absorbing the equilibrium grains in the hotter portion of the sample.

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Fig. 4 (18 diameters) represents the same phenomenon when nearly all of the small grains have been absorbed.

2. *Resistance to grain growth* might be divided into two parts:

(*a*) The resistance to grain growth inherent in the pure predominating constituent; for example, copper in the pure state at a certain temperature has a certain inherent resistance to grain growth.

(*b*) The resistance to grain growth due to obstruction by foreign bodies; for example, copper oxide in copper or pearlite in low-carbon steel.

While it might seem anomalous, high resistance to grain growth due to the second cause greatly favors the formation of very coarse grains at the germinative temperature. As the resistance to grain growth increases in a given material the germinative temperature increases, and the time necessary for the formation of the large grains decreases rapidly. As an example of the above, a sample of metal containing 2 per cent. of non-metallic material required about 1½ hr. to form the large grains at a comparatively low germinative temperature. The same metal containing 10 per cent. of non-metallic material required but 2 or 3 min. to form the coarse grains. The coarse grains in the latter case were larger than those of the former.

In the sample containing 10 per cent. of non-metallics (by volume) the germinative temperature was only 100°C. below the fusion temperature. The addition of 12 per cent. non-metallics was sufficient to raise the germinative temperature above the fusion temperature and hence make it non-existent.

If it is considered that high resistance to grain growth favors the formation of coarse grains, then, naturally, low resistance to grain growth would defeat their formation. The explanation of this is found in the fact that as the resistance to grain growth increases, corresponding to a rise in the germinative temperature, the existing grains at the germinative temperature are very much smaller than the equilibrium grains for that temperature. Since the grains are so far out of equilibrium as regards size, when the germinative grains reach a size considerably larger than the existing grains, the latter are absorbed rapidly; first, because they are so small and, second, because the temperature is so high.

The mechanism of the fast growth phenomena operates in the reverse order for low resistance to grain growth. In the first place the germinative temperature is low and the existing grains at the fast growth temperature are more nearly in equilibrium in regard to size. This, coupled with the actual lower temperature, will materially reduce the velocity of grain growth as well as the size of the maximum grains at the fast growth temperature. In order that selective grain growth may take place at all, it is essential that the existing grains at the fast growth temperature should be smaller than the equilibrium grains for that temperature. As a general rule it might be stated that, other conditions being

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equal, the finer the existing grains at the germinative temperature, the greater will be the tendency toward selective grain growth.

3. *Influence of Grain Size Prior to Deformation.*—It is easily possible in pure metals either to help or defeat the selective grain growth at the germinative temperature. If very large grains have been formed by previous high temperature anneal, like normalizing of nearly pure iron such as Prof. Sauveur used in his experiments, it is probable that the grain fragments which formed in the cold-bent specimen after annealing were so large that the germinative temperature phenomena were largely masked. In addition to this, the germinative temperature would be so low that the velocity of grain growth would be decreased. Since the coalescence of adjacent grains in the grain growth region is a function of time, among other things, it is probable that grains not far removed from the centers of selective growth might coalesce with each other before the larger grains in their conquest would have time to absorb them.

While Prof. Sauveur observes that coarse grains did not form in nearly pure iron, it has been the writer's experience that there is nearly always a region in which the grains are larger than those on either side. Sometimes the difference is not great enough to be unmistakably observed in the ordinary microscopic examination, but grain size determinations usually show the difference. A good example of the grain-growth phenomena in a solid solution comparatively free from foreign bodies will be found in Mathewson and Phillips' micrographs.[4]

In a given pure metal with a given amount of cold deformation, the grain size before deformation will have a marked effect on the selective grain growth during annealing. If the initial grain size is large the grain fragments formed during annealing will be correspondingly large, and if the initial grain size is small the grain fragments will also be smaller than in the previous case. The latter condition is conducive to the formation of large grains at the germinative temperature. Instead of normalizing the pure iron, if Prof. Sauveur had given it an initial treatment so as to produce small grains he would have noticed a more marked recrystallization effect after cold deformation with subsequent annealing. On the other hand, if the grains are initially very large it is possible almost completely to mask the fast growth phenomenon.

4. *Temperature and Deformation Gradients.*—The formation of large grains at the germinative temperature is dependent upon a rather even grain size with a temperature gradient or an even temperature with a deformation gradient or any combination of the two. In compressed metals heated by electric current the grain size is usually quite uniform and the temperature gradient serves to start the formation of the large

[4] C. H. Mathewson and Arthur Phillips: Recrystallization of Cold-Worked Alpha Brass on Annealing, *Bulletin* No. 109, p. 1 (January, 1916).

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grains. The greater the temperature gradient, the greater will be the tendency toward selective grain growth.

If a metal is deformed in the cold in such a manner that the degree of deformation differs progressively from point to point—for example, in a bent metal bar (the neutral axis of the bar will have no deformation, but the edges perpendicular to the neutral axis will have a maximum deformation)—there will be a deformation gradient from the surface to the neutral axis. If a bar so bent be heated in a muffle it might be considered that there was no temperature gradient, but the deformation gradient, with temperature constant, would produce a recrystallization gradient in which the parts most severely deformed would recrystallize at the lower temperature. This difference in temperature of recrystallization furnishes the cause for the selective grain growth due to deformation gradients. Various combinations of temperature and deformation gradients might be encountered. For example, the condition might obtain in which the temperature gradient just counterbalances the effect of deformation gradient, and fast growth at the proper temperature would not occur. Local deformation gradients occur in any metal which has been cold plastically deformed, due to the variations in the directional properties and size of adjacent grains.[5]

If the total cold deformation of a piece of metal has been moderate such as would obtain after one or two passes through rolls, the local deformation gradients may give rise to a comparatively few germinative grains which during heating absorb the inert grains thus causing coarse crystallization. As the degree of cold deformation increases, the local deformation gradients will decrease. When the total average cold deformation exceeds about 30 or 40 per cent. reduction in area of the metal, the number of germinative grains formed during normal muffle heating may approximately equal the number of equilibrium grains for the maximum temperature reached during annealing, and thus the metal is said to be fine grained. The above adequately explains the cause of coarse crystallization during annealing of moderately cold deformed metals, and the formation of progressively finer grains with the same heat treatment, as the degree of cold deformation increases.

5. *Influence of Thickness of Sample.*—Stead and Carpenter[6] find that metal sheets which are less than about 0.012 in. in thickness are very much more susceptible to the formation of coarse grains than when the thickness is greater. Their experiments were conducted with electro-deposited iron and the coarse grains formed while cooling from gamma to non-gamma iron. The reasons for the formation of coarse grains are, first, a temperature gradient and, second, the thinness of the metal plates. These conditions will be recognized as ideal for the operation of the tempera-

[5] Mathewson and Phillips: *loc. cit.*

[6] J. E. Stead and H. C. H. Carpenter: The Crystallising Properties of Electro-Deposited Iron, *Journal of the Iron and Steel Institute*, vol. 88, p. 119 (1913).

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ture-gradient law, since the progressive change of gamma to non-gamma iron would produce very small grain fragments and even a slight temperature gradient would be sufficient to cause selective growth.

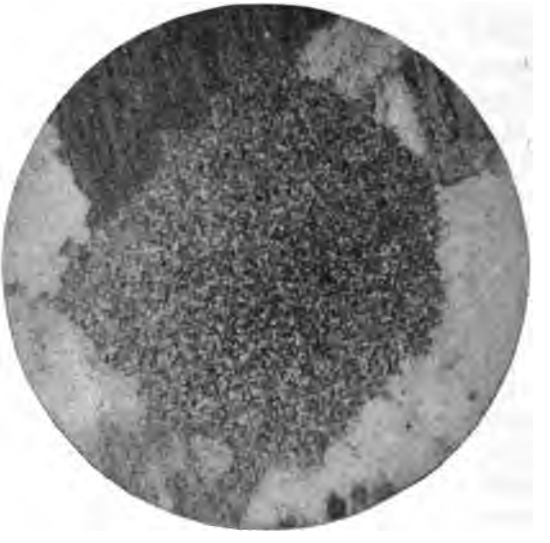

FIG. 3.—THE LARGE GRAINS FORMED AT THE GERMINATIVE TEMPERATURE IN THE PROCESS OF ABSORBING THE EQUILIBRIUM GRAINS IN THE HOTTER PORTION OF THE SAMPLE. × 18.

FIG. 4.—THE SAME AS IN FIG. 3 WHEN NEARLY ALL OF THE SMALL GRAINS HAVE BEEN ABSORBED. × 18.

The effect of thickness of section on grain growth is general in metals. The author has found that wires less than about 0.01 in. in diameter

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have two regions in which fast grain growth takes place. One occurs at the ordinary germinative temperature. The other occurs at most any temperature above the germinative temperature, the time varying inversely as the temperature. In the second fast growth range the large grains almost invariably first form near the surface of the sample. It is the writer's opinion that in such cases, where the ratio of the surface grains to the interior grains is large, the former, being unbalanced by having only one side in contact with adjacent grains, are readily absorbed by these interior adjacent grains. When the size of the surface grains due to this cause is enough greater than the equilibrium interior grains the latter are absorbed. The formation of large grains caused by small diameters of wires can be materially checked by the introduction of foreign bodies such as non-metallics. The practical effect of the introduction of non-metallics is to decrease the diameter of wire above which coarsening does not take place. In wires having a diameter of less than about 0.002 in. it is doubtful whether the formation of grains extending across the diameter of the wire can be prevented by any means, provided the wire is exposed for a long time at a sufficiently high temperature.

Fast Growth Phenomena in Low-Carbon Steel

What has preceded in this paper has considerable bearing on the formation of coarse grains in low-carbon steel. Referring to Prof. Sauveur's Fig. 27 and Chappell's Fig. 12,[7] there is a good reason why the coarse grains in the latter should be more columnar. In Sauveur's Fig. 27, as the large grains grow toward points of less cold deformation they encounter larger and larger grain fragments. This will be recognized as an adoption of Dr. Mathewson's ideas and nomenclature. The obstruction to grain growth offered by non-metallic globules is such that after the coalescence of the grains the globules do not change position but the grains grow around them. Assuming the same to be the case with pearlite as the obstructing medium (although the ferrite of the pearlite may be divorced, leaving only the cementite as the obstructing medium), the fragments which form in the interior of an original grain of ferrite would not have obstructing bodies at their interior boundaries. In the first stages of selective grain growth the grain fragments, being small, would be absorbed readily one at a time and when the obstructing masses of pearlite were encountered grain growth would easily take place around them. As the grain fragments increase in size their resistance to absorption becomes greater; hence, a point will be reached at which equilibrium exists. This is especially true when considering a definite time period.

On the other hand, as the large germinative grains grow at the expense

[7] C. Chappell: Recrystallization of Deformed Iron, *Journal of the Iron and Steel Institute*, vol. 89, pl. 48 (1914).

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of the small inert grains, the latter will have had more time to coalesce with one another as the time of sojourn in the germinative temperature range increases. It is well known that two adjacent grains which are not able to coalesce with each other in five minutes, might do so in five hours. Another factor which should not be lost sight of is that when a metal has been but slightly cold deformed, some of the adjacent grain fragments resulting may be in the same orientation and only be separated by a slip plane of amorphous metal. This amorphous material may be absorbed during heating by the crystalline metal, thus producing what is analogous to coalescence.

In Chappell's Fig. 12, however, the grain growth took place at a temperature well above the Acl point (870°C.) In very low carbon steels in which the mass of ferrite is much greater than that of the pearlite, there is considerable evidence that the pearlite does not change into the equilibrium percentage of austenite when heating above Acl. Quenching experiments on samples which have been previously normalized, invariably show less martensite in the very low carbon steels than would be expected from the equilibrium diagram. When these same steels, according to Osmond[8] are heated to very high temperatures—1200° to 1300° C.—larger percentages of martensite are formed. It is likely, therefore, that there is less austenite formed during the heating of these steels than would be expected from the equilibrium diagram. This is borne out by heating curves and by the occurrence of globules of free cementite. In fact, Chappell's own micrographs would lead one to this conclusion.

It is further probable that germinative grains which had reached conconsiderable size below the Acl point, would have greater power to divorce the ferrite from the pearlite than would smaller grains.* Consequently, while the pearlite or austenite might easily prevent the small grains from coalescing with one another, the large germinative grains might have sufficient power to absorb the inert grains. This is precisely what does happen in the case of a pure metal mixed with about 10 per cent. of its volume of a non-metallic material.

Prof. Howe's suggestion, that about 13 per cent. pearlite might easily offer sufficient mechanical obstruction to grain growth to completely defeat the selective action, is highly acceptable. Before leaving this phase of the subject, it would be interesting to consider the effect of the critical transformations in steel. If a piece of steel were heated by electric current to such a temperature that the inner portion were above the

*The writer has frequently measured the areas of pearlite in very low carbon steels, and almost invariably the percentage is less than would be expected from the combustion carbon determinations.

[8] *Trans.*, vol. 27, p. 879 (1897). A steel containing 10 per cent. carbon quenched from 1050° C. would consist of 60 per cent. free ferrite and 40 per cent. martensite.

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$A3$ point and the outer portion below the $A3$ point, a fast growth temperature condition should exist at the intersection of these two regions. Similarly, during cooling from above the $A3$ point if the inner portion were kept above the transformation point and outer portion below, a fast growth temperature condition should exist at the boundary. By the proper regulation of the heating rate or the cooling rate either large austenite grains or large non-gamma iron grains should be formed.

The existence of gases in metals cannot be neglected as a factor in the resistance to grain growth. It would seem, however, that if gas globules were responsible for the obstruction of grain growth, their presence could be determined by the aid of the microscope using high magnifications. That gases in solution or mechanically entrapped in metals may greatly affect the annealing temperature after cold deformation is shown quite conclusively by Rose[9] and Phelps.[10]

Effect of Cold Deformation on Uniformity of Orientation of Original Grains

Prof. Howe's statement that the uniformity of orientation after cold deformation is greater within the individual original grains than between adjacent grains, is no doubt generally true. It must be considered, however, that in cold-deformed metals there is sufficient difference in orientation within the original grains to be the chief cause of recrystallization on annealing. That we do not always see these differences in orientation may be due (1) to the fact that the diameter of individual grains in which we can study etching pits is usually not more than 0.0002 in. This distance is so small that variations in orientation might easily escape even a close observation. (2) To the fact that the sections (viz., longitudinal) which are usually examined in deformed metals are the most conducive to similar etching tints.

An experiment will demonstrate the points in question very clearly. Small wires 0.003 in. in diameter, were heat-treated in such a way that grains about ½ in. long were formed, each grain occupying the full section of the wire for that length. Pieces of this wire were bent sharply through an angle of about 90° in such a manner that the two ends of a single grain were not internally deformed. In other words, all of the deformation due to bending was confined to a small area near the center of an individual grain. It is obvious that the two ends of the grain in question now have decidedly different orientations. The exact angle of difference is known, namely, 90°. It is obvious that there is no sharp

[9] Thomas Kirke Rose: On the Annealing of Gold, *Journal of the Institute of Metals*, vol. 10, p. 150 (No. 2, 1913).

[10] John Phelps: The Effect of Hydrogen on the Annealing of Gold, *Journal of the Institute of Metals*, vol. 12, p. 125 (No. 2, 1914).

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line of demarcation at the bend. The change in orientation is progressive. When a sample like the above is examined under the microscope the etching tint is usually quite uniform. However, if the same sample be subjected to a twisting operation the orientation has been changed with respect to different axes, and the etching tint does not remain uniform. Fig. 5 (660 diameters) is a photomicrograph of a sample of wire so twisted. The differences in etching tint can be readily seen.

The recrystallization properties in the bent sample are approximately the same as those in the twisted sample. As further evidence of decided change in orientation within the initial grains, Stead and Carpenter

A SINGLE GRAIN, TWISTED COLD. SHOWS DIFFERENCES IN ETCHING TINT. × 660.

demonstrate this condition very nicely by means of different etching tints within the same initial grain and by drawings (The Crystallising Properties of Electro-Deposited Iron, *Journal of the Iron and Steel Institute*, vol. 88, Figs. 6*A* and 6*B*, Plate 12. Drawings on p. 147 (1913).

No explanation of the recrystallization of cold-deformed metals on annealing has been forthcoming which did not assume a change in orientation within the initial grains due to the deformation. Rosenhain[11] seeks to explain the formation of the new grains by assuming that the amorphous material forms nuclei for the new grains. The study of saturated liquid solutions in the presence of the solid solute teaches us

[11] *Internationale Zeitschrift für Metallographie*, vol. 5, p. 65 (1913).

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that the solute in the crystalline state, and not the solute or solvent in the amorphous (dissolved) state, is the controlling factor governing saturation equilibrium. As an example, if a solution of sodium chloride in water is saturated at a certain temperature with no crystalline sodium chloride present, the dissolved salt contains a certain amount of potential energy. This potential energy is not changed in quantity by the addition of a sodium chloride crystal to the solution, but some of the dissolved salt crystallizes and the degree of concentration in the solution is lowered. The force necessary to change this degree of concentration must be looked for in the crystal of sodium chloride. In the presence of amorphous and crystalline metal, therefore, the crystalline fragments and not the amorphous layers will be the controlling factors in recrystallization.

Beilby[12] has demonstrated the above experimentally. This experiment is so basic in its general relationship to recrystallization that it should be repeated here in his own words. "Experiments have been made on the building up of an entirely amorphous mass of gold. Gold was precipitated from a solution of the chloride as a spongy brown powder. The washed and dried powder was compressed in a die after the gases it had contained had been pumped off at a temperature of about 150°. The pressure was applied by a small hydraulic press. This pressure was so great that the gold plug made a depression in the bottom plate of the die, which was of tool steel. The pressure must have been 60 to 80 tons per square inch. The gold squirted very slightly through a hole in the bottom plate, and also between the plunger and the sides of the die. The squirted gold was very hard and rather brittle, quite unlike pure gold prepared in any other way. The specific gravity of the compressed plug was 19.05, or rather lower than the specific gravity of gold. Its microstructure was granular and slightly spongy. This amorphous gold was certainly very much more rigid than even the hardest of cold-worked gold; it was almost devoid of plasticity. Another plug of compressed gold was made from the same powder, but a tiny spiral of very fine soft wire was dropped into the die before it was packed with powder. When a very moderate pressure was reached the soft gold squirted quite freely past the plunger, leaving the plug of hard gold behind.

"The first compressed plug was heated to 350° and showed no development of crystallization. In the absence of crystalline nuclei the amorphous form is stable even at this temperature. The squirted metal from the second plug which did contain nuclei crystallized freely at 350°."

[12] G. T. Beilby: The Hard and Soft States of Metals, *Journal of the Institute of Metals*, vol. 6, pp. 36–37 (No. 2, 1911).

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Biographical Notice of Charles Kirchhoff

BY R. W. RAYMOND, NEW YORK, N. Y.

(New York Meeting, February, 1917)

CHARLES WILLIAM HENRY KIRCHHOFF was born March 28, 1853, at San Francisco, Cal., where his father, Charles Kirchhoff, was at that time employed in the consular service of his native country, Germany. A few years later, the family moved to Hoboken, N. J., in the public schools of which city the son received his preliminary education, proceeding later to Germany, where he was graduated in 1874 as mining engineer and metallurgist at the Prussian Royal Mining Academy of Clausthal, in the Harz. Upon his return to the United States, he became chemist of the Delaware Lead Refinery at Philadelphia, and retained that position for three years. But in 1877 he began what was to be an almost uninterrupted life-long association with David Williams, the publisher of *The Iron Age*, of New York, who established in that year, and continued for a brief period, a journal entitled the *Metallurgical Review*, on the editorial staff of which Mr. Kirchhoff received a place. The enterprise was doubtless an attempt to cover a wide metallurgical field outside of that which properly belonged to *The Iron Age*. But it was soon abandoned, and Mr. Williams wisely concentrated his energies upon the older journal, which, under his vigorous and skillful management, and the labors of the able editors and correspondents whom he selected, became one of the greatest institutions of its class in the world. From 1878 to 1881, Mr. Kirchhoff was assistant editor of *The Iron Age*. From 1881 to 1884, he was managing editor of the *Engineering and Mining Journal;* in 1884 he returned to *The Iron Age*, to become for five years its associate editor, and in 1889, on the retirement of James C. Bayles, editor-in-chief.

This list of dates and employments, without further comment, sufficiently indicates, at least to the eye of an expert, that Mr. Kirchhoff had found his congenial career in trade journalism. His qualifications for this profession were somewhat exceptional. He possessed the scientific training, the knowledge of foreign languages and literatures, and the power of making and keeping friends, which enabled him to get early notice of technical novelties; he had the taste for statistics which made him both industrious and intelligent in their collection and use; and to these traits he added a mastery of the meaning of such accumulated data, and a

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sane, critical judgment of the situations which they represented, as well as of the sources and the figures themselves, which made his opinion weighty concerning them. His estimates and prognostications, expressed in quiet, clear, forcible but unsensational style, were quoted by the non-technical press, when editorial utterances, more brilliant in rhetoric or pleasantry, passed unrepeated. In short, his mind was equipped for the long-distance as well as the near-by view of things; he could see both science and business, and his lens was achromatic at either focal distance. Such men surely achieve reputation and influence, and never lose what they have thus patiently won.

It was in 1883, while he was managing editor of the *Engineering and Mining Journal*, that Mr. Kirchhoff was engaged by the Director of the U. S. Geological Survey to collect and arrange annually the statistics of the production of lead, copper and zinc in this country. Having had, at an earlier period, some personal experience in work of that sort, I feel myself qualified to judge of its difficulties and to recognize success in the overcoming of them. It is not a simple matter of commanding the reports of all producers and compiling the results. It means the personal winning of the confidence of individual producing concerns, one after another; the intelligent reduction of their data to a common standard; the ingenious filling of gaps in the returns; the checking of statistics of production by those of transportation, sales and stocks on hand—in short, the editing, in a critical as well as a mechanical sense, of a mass of material more or less incomplete and heterogeneous. Such a labor, performed by a competent hand through a series of years, doubtless grows easier and easier because it educates the contributors while it makes them increasingly willing to do their part toward a result so creditable and useful. Mr. Kirchhoff's reports, continued for 23 years, constitute a lasting memorial of his own ability and of his influence upon the managers of great national industries.

But a man might possess all the admirable qualities above enumerated, and yet lack the indefinable gift of administration. This gift Mr. Kirchhoff exhibited when in 1904, upon the death of John S. King, business manager of *The Iron Age*, he became Vice-President and General Manager of the great David Williams Co. This responsibility, as well as that of his editorial position, he carried successfully until his retirement from active business at the close of 1909, when *The Iron Age* was sold to other parties. This was the occasion of a luncheon in his honor, given by his friends and associates at the Engineers' Club, Jan. 16, 1910, and made notable by the attendance of many distinguished engineers and captains of industry, including the venerable John Fritz, then 87 years old, who made the journey from Bethlehem for this purpose. Numerous letters and speeches (including the presentation, by Mr. Kirchhoff's associates in the David Williams Co., of a bronze statue) bore witness to the

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admiration and esteem with which he was universally regarded. There was general regret over his retirement from a position which he had so signally adorned—a regret from which, as I remember, I ventured to dissent, preferring to praise and congratulate him upon a rest well earned, and taken at a time when, with powers unimpaired, he could both enjoy and make fruitful of good to his fellows a life more wide and free. Alas! it was otherwise decreed.

Mr. Kirchhoff's activity and reputation were not confined to the United States. As editor of *The Iron Age*, he maintained relations with technical and commercial leaders abroad, and frequently visited Great Britain, Germany and France for the purpose of studying the developments of the iron and steel industries of those countries. A series of his articles in *The Iron Age*, republished in 1900 as a book, entitled "Notes on Some European Iron-Making Districts," was widely read. After his retirement, he reviewed in the columns of *The Iron Age* the proceedings of the International Metallurgical Congress, which he attended at Düsseldorf in the spring of 1910; and in the following October he read before the American Iron and Steel Institute in New York a notable paper on the recent progress of iron and steel metallurgy, as reflected in that Congress.

Mr. Kirchhoff was specially active and prominent in connection with the American Museum of Safety Devices and Industrial Hygiene, on the managing board of which he served for several years. His labors in this field were recognized by the government of France, which conferred upon him in 1908 a decoration as *Officier d'Académie*.

He was a member of the American Iron and Steel Institute, the Iron and Steel Institute of Great Britain, the Verein deutscher Eisenhüttenleute, the American Society of Mechanical Engineers, and honorary member of the Franklin Institute of Philadelphia.

He joined the American Institute of Mining Engineers in 1875, immediately after beginning his career as a metallurgical chemist. In 1884, he was strongly supported as a candidate for the office of Secretary, left vacant by the resignation of Dr. Brown. He and I were not only friends but associates at that time on the editorial staff of the *Engineering and Mining Journal;* and these relations were not at all disturbed by our amicable contest for the Institute secretaryship. No one who was present at the banquet held in connection with the Cincinnati meeting of February, 1884, has forgotten the graceful and humorous speech made by Mr. Kirchhoff from the standpoint of the defeated candidate. Thenceforward to the end of his life, he was my strong supporter, wise adviser, and loyal and beloved friend.

In 1887 and 1888, and again in 1892 and 1893, he served as a member of the Council; and in 1896 and 1897, as Vice-President; in 1898 and again in 1911 as President; from 1907 to 1912 as Director, and in 1913 as Past-President. His presidential address, at the New York meeting of Febru-

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ary, 1899, on "A Decade of Progress in Reducing Costs," is a classic, showing precisely what such a paper should be, and exhibiting those characteristics of careful and tireless inquiry, comprehensive grasp of details, wisdom in generalization and clearness of statement which I have already noted as the elements of his power and success.

CHARLES KIRCHHOFF.

But not less important—and far more laborious was his work in the memorable year 1890, when the Institute, with the aid of American engineers and ironmasters, entertained some 500 guests, belonging to the Iron and Steel Institute of Great Britain and the Association of German Ironmasters. Mr. Kirchhoff was Secretary of the American Reception Committee, of which Mr. Andrew Carnegie was Chairman. But his most arduous labors were performed in connection with Mr. W. P.

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Shinn of the Transportation Committee, for whom he personally conducted one of the two Pullman trains which were kept running through the month of October with the foreign guests as passengers.

For some years before his death, Mr. Kirchhoff fought a brave battle against a disease usually regarded as fatal. With patient and cheerful courage, he held his own for a while, and even seemed to be gaining ground. But it was trench-fighting, and gains were small and slow. An attack of the grip, reinforcing the older enemy, brought the end at last; and he died, July 22, 1916, at his summer home near Asbury Park, N. J. He leaves a widow, two sisters and a brother.

In closing this imperfect sketch of my dear friend, I can present no better summary of his character than that which is embodied in the Minute adopted by the Directors of the American Institute of Mining Engineers upon the tidings of his death, and I therefore repeat as my own this paragraph:

"Mr. Kirchhoff was one who secured, because he practiced, loyalty in friendship, justice and kindness in personal intercourse, openness and equity in business. Modest, but not timid; prudent, but progressive; gentle, but firm; full of chivalry and of common sense; indefatigably industrious and inexhaustibly patient, he won victories without making enemies, and crowned with a heroic death the record of a life unstained."

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Function of Alumina in Slags

BY CARL HENRICH, LINCOLNTON, GA.

(New York Meeting, February, 1917)

I HAVE read with particular interest that portion of the discussion by Anton Eilers referring to the high-lime (and also high-alumina) slags made by August Raht in 1881, while smelting the Horn Silver ores at Franklin, Utah. The two analyses of such slags, furnished by Mr. Raht to Mr. Eilers are:

Horn Silver Slags

	Slag I	Oxygen(*b*) percentage	Slag II	Oxygen(*b*) percentage	Average of I and II	
						Oxygen
SiO_2	33.9	18.1	35.0	18.7	34.50	18.40
FeO	26.8	6.0	25.4	5.7	26.10	5.80
CaO	26.1	7.5	24.9	7.1	25.50	7.30
BaO	3.2(*a*)	0.3	3.2	0.3	3.20	0.30
Al_2O_3	10.8	5.1	9.3	4.4	10.05	4.75
	100.8	37.0	97.8	36.2		

(*a*) Not determined in this analysis.
(*b*) Not given in Mr. Eilers' paper.

Oxygen ratio, Al as base.

I. O in acid: O in base::18.1:18.9::0.96:1
II. O in acid: O in base::18.7:17.5::1.06:1
Average: 18.4:18.15 = 1.01:1

Oxygen ratio, Al as acid.

I. O in acid: O in base::23.2:13.8::1.68:1
II. O in acid: O in base::23.1:13.5::1.76:1
Average: 23.15:13.4 = 1.73:1

By assigning to alumina the rôle of a base, Mr. Eilers deduces the formula $3FeO.2SiO_2 + 4CaO.2SiO_2$.

The nearest approach to a simple formula expressing the average composition of these slags, which I can figure, is

$$9RO.Al_2O_3.6SiO_2.$$

Allowing a ratio of 4FeO : 5CaO, we have for this formula:

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SiO_2 = 35.0 per cent.; Al_2O_3 = 10.0 per cent.; CaO = 27.0 per cent.; FeO = 28.0 per cent.; in round numbers. This would be a singulo silicate of CaO and FeO with about 34 per cent. or one-third of the RO bases replaced by Al_2O_3.

However, it has always appeared to me to be an absurdity to force a sesquioxide to replace FeO or CaO or any other RO, simply to make the analysis of some slag or mineral fit into some orthodox formula. That Al_2O_3 can be substituted for Fe_2O_3 or Cr_2O_3, just as MgO, CaO, BaO, FeO, and MnO can be substituted for each other in various chemical combinations, is well established. But I do not know of any natural silicate in which it has been shown conclusively that Al_2O_3 would take the place of RO, and I do not believe that alumina will do it in any artificially produced slag silicate.

If in the above slags we consider alumina as an acid, *i.e.*, the slags as mixtures of silicates and aluminates, we find that they satisfy, or closely approach the formula:

$$(9RO.5Al_2O_3) + 6(6RO.5SiO_2).$$

In short, assuming alumina as an acid, $2Al_2O_3$ would be the equivalent of $3SiO_2$, or 102.8 mass-units of alumina would be the equivalent of 90 mass-units of silica, and the oxygen ratio of the correct slag of the above type would be 5 : 3. The slag would be essentially a lime-iron 5 : 3 "silic-aluminate" (slightly acid "sesqui-silicaluminate").

Slag No. I corresponds very nearly to this degree with an oxygen ratio of 23.2 : 13.8. Slag No. II does not correspond so well, and it is also metallurgically slightly inferior to No. I as shown by the lead and silver contents. On the supposition of Al as basic, the increase in the silica of this slag should make it better metallurgically than No. I, which, however, is evidently not the case.

I am aware that many metallurgists are unwilling to assign to alumina the rôle of an acid or to consider it as a substitute for silica in the calculation of their slags. Except in iron blast-furnace practice, it is of no great importance whether we consider it as a base or an acid or do not consider it at all. As the insoluble residue of ores or limestone is frequently calculated as silica, it is automatically assigned to the rôle of an acid. But practical metallurgists have, as a rule, avoided the issue by following Kerl's advice and have kept the alumina of their slags as low as possible, so low, in fact, that usually it has been immaterial which way the alumina has been regarded. The metallurgist in a modern centrally situated blast-furnace plant can do this, but occasionally the alumina problem becomes a serious one to the metallurgist at an isolated plant, smelting, perhaps, the ore from only one mine, so that the question has its interest from a practical as well as from a scientific side.

In any slag the chief points of interest are:

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The formation and melting points, since these determine coke consumption and furnace temperature; its viscosity, which must be small, both to allow it to flow readily from the furnace and to permit a quick mechanical separation of it from the matte, speiss or metal; the chemical composition, since it must not dissolve large quantities of the valuable constituents of the ore, and must (as in iron smelting) occasionally remove detrimental ones.

In general, an increase of alumina or silica will raise the formation temperature of a slag, and ordinarily it will be made thereby more viscid at or near its formation temperature; but these qualities are likewise affected by the various bases entering the slag and their relative proportions.

As practical experience has ordinarily shown that scaffolds over the tuyères, and hard slag and metal taps resulted from high alumina, among the bases in unisilicate and sesquisilicate slags, alumina has acquired a bad reputation among lead and copper smelters.

I might suggest at this point that it is a well-known fact that suddenly cooled slags ("chilled") are more quickly attacked by hydrochloric acid than if slowly cooled. This suggests that the crystallized and crystalline minerals occurring in slowly cooled slags are not contained as such in the molten magma, but are only formed from the original magma during the cooling process.

Among the cases in which high alumina in the smelting charge became a practical problem were the Horn Silver ores smelted by Mr. Raht in 1881, already mentioned, the oxidized ores of the early days of the Copper Queen at Bisbee and the Detroit Copper Mining Co. at Morenci and my own experiences in smelting some ores of the Champion and United Copper Mines near Nelson, N. Z.

In those times, I suppose, we all considered alumina as a base when we considered it at all. I know that I did when confronted with an unusual amount of kaolinized porphyry in some of the Detroit Copper Mining Co.'s ores. I know, too, that the behavior of the slags calculated in this way was often a sore puzzle to me. For instance, I give below two slags made at Morenci about 1884. No. I was a good slag, keeping the furnace open and running freely, the other was a bad slag, inclined to form noses and scaffolds above the tuyères, thick, sticky and sure to bring the smelting campaign to an abrupt end if not promptly changed.

The oxygen ratio, considering alumina as a base, does not throw any light on the radically different nature and behavior of these slags. The only thing which at that time seemed feasible to me, was to follow Kerl's rules as far as possible, and keep the alumina low.

But when I went from Morenci to Nelson, N. Z., and there faced, unexpectedly, the problem of smelting the copper ores of one mine, with no available flux but magnesian limestone, I found myself "up against

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Detroit Copper Mining Co.'s Slags

	Slag I	Oxygen percentage	Slag II	Oxygen percentage
SiO_2	34.3	18.3	38.1	20.3
Al_2O_3	11.8	6.5	16.9	7.9
FeO	38.5	8.6	33.1	7.4
CaO	10.1	2.9	8.0	2.3
MgO	2.3	0.9	2.2	0.9
	97.0		98.3	

Oxygen ratio,

Al_2O_3 base:	18.3 : 17.9	20.3 : 18.5
Al_2O_3 acid:	23.8 : 12.4	28.2 : 10.6

it." The ore was mainly chalcopyrite, with some pyrite, in a gangue locally known as serpentine. The average composition of the country serpentine on both sides of the lode serpentine appeared to have the approximate formula $3MgO.Al_2O_3.3SiO_2$ or about 30 per cent. MgO, 25 per cent. Al_2O_3 and 45 per cent. SiO_2.

The lode serpentine, forming the gangue of the ore, approached more nearly the composition of $4MgO + 4Al_2O_3 + 5SiO_2$, or about 17.5 per cent. MgO, 43.5 per cent. Al_2O_3 and 39 per cent. SiO_2. One actual analysis of the gangue of roasted ore figures 36 per cent. SiO_2, 46 per cent. Al_2O_3 and 18 per cent. MgO_3. That silica-alumina ratio in the available ore of the smelting charge was a decided and puzzling novelty. Kerl's advice was evidently not applicable in this case. As I published, in 1886 or 1887, in the *Engineering and Mining Journal*, an account of my experiences in smelting that ore, it would be superfluous to repeat that account here. The following slag analyses and appended remarks will suffice:

	Slag 1	Oxygen percentage	2	O	3	O	4	O	5	O
SiO_2	36.4	19.4	34.6	18.4	39.1	21.3	21.8	11.6	18.4	9.8
Al_2O_3	27.8	13.1	26.4	12.4	22.9	10.8	31.8	14.7	20.8	9.8
FeO	14.2	3.1	32.2	7.2	28.6	6.4	30.6	6.8	35.9	7.9
CaO	1.5	0.4	....	....	....	....	....	.....	10.0	2.9
MgO	18.4	7.4	6.8	2.7	8.4	3.3	15.5	6.2	14.3	5.7

Oxygen ratio,

Al basic:	19.4:20.4;	18.4:22.3;	21.3:20.5;	11.6:27.7;	9.8:26.3;
	0.95	0.83	1.05	0.42	0.37
Al acid:	32.5:10.9;	30.6:9.9;	92.1:9.7;	26.8:13.0;	19.6:16.5;
	2.98	3.09	3.32	2.02	1.19

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	6	O	7	O	8	O	9	O
SiO_2	19.7	10.5	22.6	12.0	24.2	12.9	23.9	12.7
Al_2O_3	18.5	8.7	22.5	10.6	26.2	12.3	26.8	12.6
FeO	28.5	6.3	32.7	7.3	32.1	7.1	33.5	7.4
CaO	15.0	4.3	5.1	1.4	8.0	2.3	6.4	1.8
MgO	18.3	7.3	18.4	7.4	9.2	3.7	9.4	3.8

Oxygen ratio,

Al basic: 10.5:26.6; 0.40 — 12.0:27.8; 0.43 — 12.9:25.4; 0.51 — 12.7:26.6; 0.50

Al acid: 19.2:17.9; 1.07 — 22.6:15.2; 1.49 — 25.2:13.1; 1.92 — 25.3:13.0; 1.95

1. Slag of first trial run—very short run. Scaffolded and froze up. Furnace: 36-in. round water-jacketed (1884) Arizona copper furnace.

2. Slag from same run, when furnace began to scaffold.

3. Same run. From chilled scaffold above tuyères.

4. From second equally short run. Somewhat better roasted ore, more coke and somewhat higher blast pressure. Thick, gluey slag from slag-tap. Scaffolding.

5. From third, successful run, this time considering alumina as an acid, and counting 35 weight-units of alumina the equivalent of 30 units of silica. Blast pressure: 9 to 10 oz. First slag fall, changed to:

6. By addition of more limestone flux to the charge. This is a very free-running light slag, separating easily and clean from the 45 per cent. copper-matte. Thirty-six-inch furnace running at the rate of 45 tons smelting charge in 24 hr.

7. Slag from last 12 hr. of run, finishing roasted ore on hand, and with shortage of limestone flux, which could not be replenished in time. Furnace not running as free as on No. 6 slag, only at the rate of 27 tons in 24 hr. As available limestone flux diminished, Slag Nos. 8 and 9 ensued, which allowed to finish the campaign with increased fuel and

	Slag	Oxygen percentage
SiO_2	26.6	14.2
Al_2O_3	15.4	7.2
FeO	42.6	9.5
CaO	9.5	2.7
MgO	0.2	0.1

Al_2O_3 as a base; Oxygen ratio: 14.2:19.5. 0.73

Al_2O_3 as an acid; Oxygen ratio: 21.4:12.3. 1.74

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blast, using up all roasted ore on hand. If continued on this slag, however, the furnace would surely have scaffolded and frozen up.

An instructive slag, high in alumina, is the preceding typical Copper Queen slag of about 1884:

Considering the composition and behavior of all the slags given above it seems to me evident:

1. In slags containing larger quantities of alumina, the alumina should be considered as an acid, replacing silica, and not as a base.

2. The higher the percentage of alumina, the nearer the slag should approach a "singulo-silicaluminate," *i.e.*, the nearer the oxygen ratio of the bases should come to the combined oxygen of the silica and alumina.

3. An increase in magnesia calls for a higher percentage of bases, while absences of magnesia, and a pure limestone as a flux, will permit an approach to a "bi-silicaluminate" slag.

4. The safe way will be to start with a "sesqui-silicaluminate" as the type of slag to be produced:

$$x(4RO.3SiO_2) + y(2RO.Al_2O_3),$$

which is approximately the slag actually, successfully and involuntarily, I suppose, made in 1881 by August Raht, while smelting very aluminous and very limey ores from the Horn Silver Mine.

There has always been a bias among metallurgists in favor of assigning to alumina the rôle of base in the composition of the slags. This has been natural, as the textbooks on mineralogy assign to alumina in the composition of the silicate minerals the rôle of a base, with very few exceptions. Hence, not questioning the correctness of this authoritatively assigned basic function of alumina, expressed as such in all the orthodox formulas of these minerals, it seemed only proper and correct to consider Al_2O_3 as a base in the artificial silicates, the slags. When this view played the mischief with the running of the furnace, this was ascribed to the innate depravity of alumina. You could support a little of it, but you must not allow too much of it, or it would get the better of you. That was about the practical conception in lead and copper smelting, using low furnaces, cold blast, low pressure, and under the necessity of keeping the fuel cost down.

At least, this accurately describes my own view of the case in 1884, when confronted with kaolinized porphyry of the Detroit Copper Mining Co. But having had forced upon me the conviction of the acid character of Al_2O_3 in the artificial slag silicates, or at least in some of them, it has engendered some doubt whether alumina always has that function of a base in the natural silicate minerals assigned to it in the orthodox formula of silicates, a question with which I purpose dealing at some future time.

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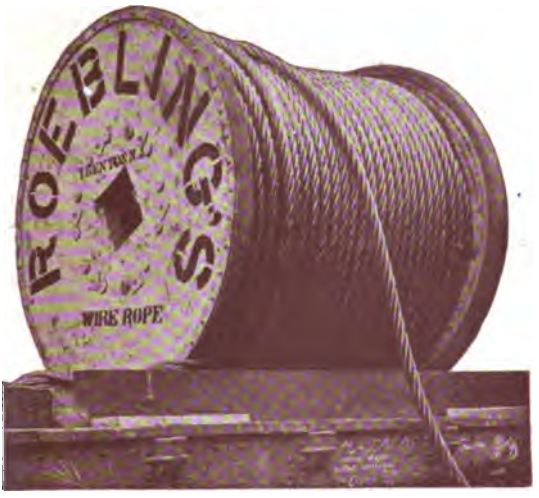

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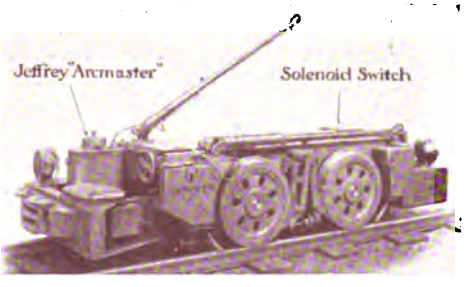

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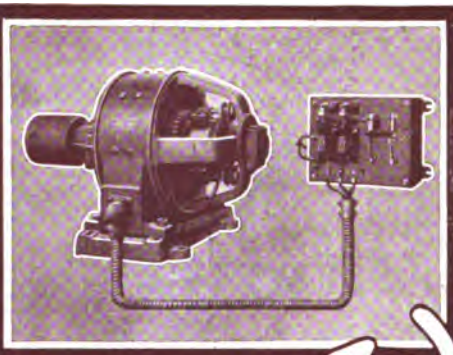

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DECEMBER, 1916

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Industrial Section . . . . . (Interpolated in Advertising Section)

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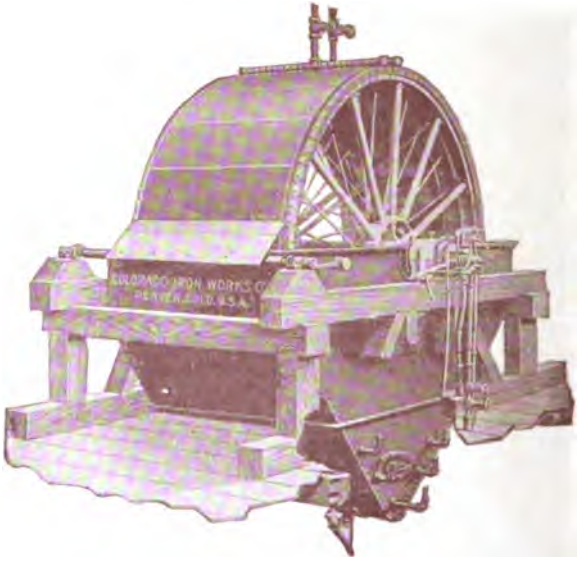

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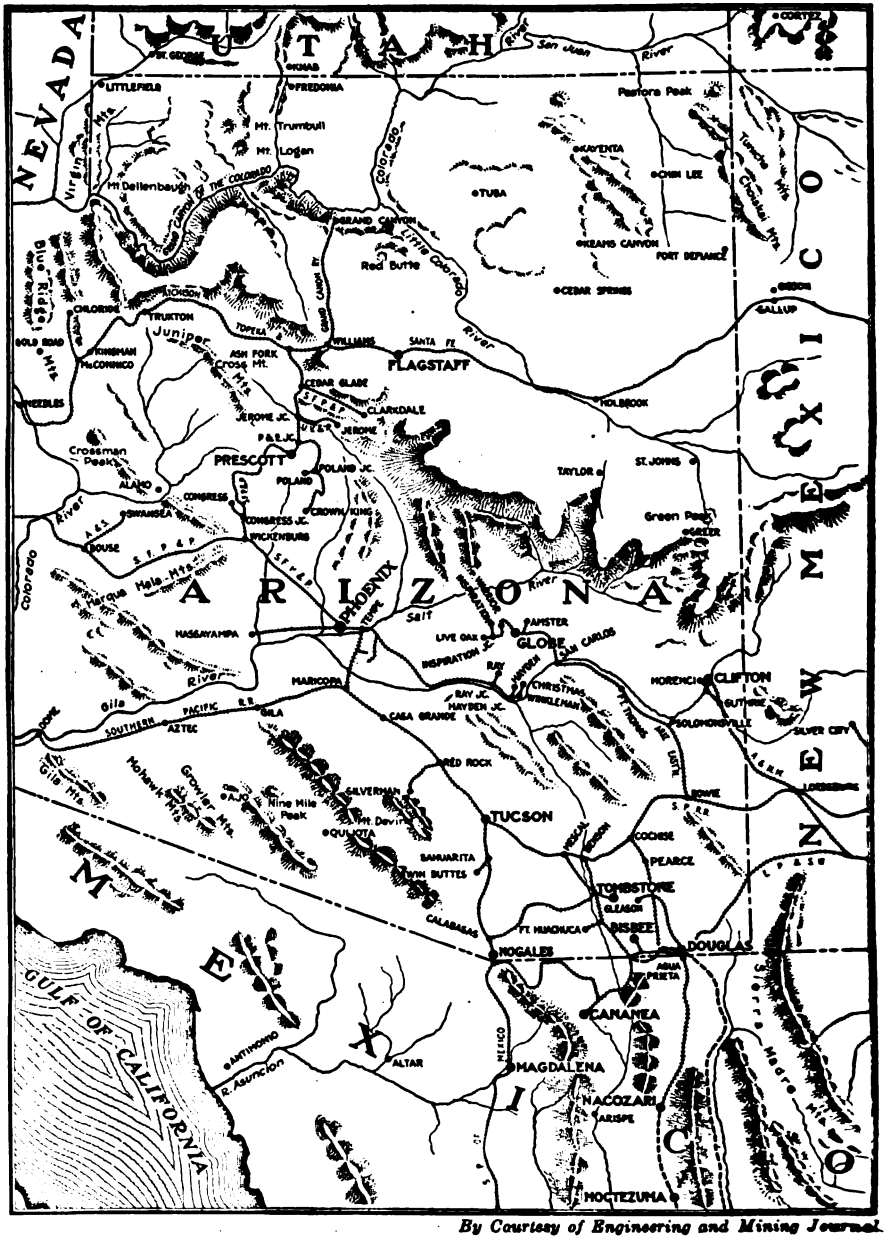

By Courtesy of Engineering and Mining Journal

MAP SHOWING POINTS IN ARIZONA VISITED DURING THE SEPTEMBER, 1916, MEETING.
(Santa Rita and Hurley, N. M., where are located the mine and mill of the Chino Copper Co., were also visited, but are not shown on the map.)

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1917 DUES

In accordance with the provision of the Constitution, notice is here given to all Members, Associates and Junior Members, that the dues of the year 1917 will be payable on Jan. 1, 1917, at the office of the Secretary. The dues for Members and Associates are $12 per year and the sum of $3 additional is asked of those Members and Associates who desire to have the three volumes bound in the Institute's standard half morocco binding.

The dues of Junior Members are $5 per year.

Volume 54 of the *Transactions* is now printed, and will be sent in January to all members whose dues for the year 1917 are paid.

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PROCEEDINGS OF THE ONE-HUNDRED AND THIRTEENTH MEETING, ARIZONA

Sept. 17 to 25 inclusive, 1916

GENERAL COMMITTEE

GERALD F. G. SHERMAN, *Chairman.*
ARTHUR NOTMAN, *Secretary.*

NORMAN CARMICHAEL,
W. L. CLARK,
B. BRITTON GOTTSBERGER,
JOHN C. GREENWAY,
W. G. McBRIDE,
FOREST RUTHERFORD.

General Committee on Transportation

WALTER DOUGLAS, *Chairman.*
CLEVELAND E. DODGE, *Secretary.*
ARTHUR S. DWIGHT,
JULIUS KRUTTSCHNITT, JR.
JOHN C. GREENWAY,

Committee at El Paso

W. D. GORDON, ROBERT McCART, JR., W. W. ROSE.

Committee at Santa Rita

JOHN M. SULLY, *Chairman.*

Committees at Douglas

Reception	*Train Movement*	*Technical Session*
H. A. CLARK,	J. O. AMBLER,	H. O. HAMMOND,
G. H. DOWELL,	ROBERT RAE.	FOREST RUTHERFORD.
A. V. DYE.		

Committee in the Globe-Miami Section

Local Committee

L. O. HOWARD, Globe, F. W. MACLENNAN, Miami, *Joint Chairmen.*

Entertainment of Ladies

MRS. B. B. GOTTSBERGER,
MRS. I. N. BARKDOLL,
MRS. L. O. HOWARD,
MRS. W. B. CRAMER.

Technical Sessions and Banquet

J. H. HENSLEY, JR.
G. N. BJORGE.

Transportation, Automobiles, Etc.

I. N. BARKDOLL, H. P. BOWEN, W. F. GEIGER.

Bureau of Information

ROBERT FRANKE.

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Such extraordinary developments in mining and metallurgy and so many new departures have been made in mining, concentration and smelting in the State of Arizona in recent years that extraordinary means had to be taken to cover the more important points of interest in the State. A special train to carry the party from point to point during the hours of darkness, so that daylight might be used to the utmost advantage was therefore provided. Even with this advantage, it was impossible to visit all the camps, and it was with great regret that the main party was obliged to omit such interesting places as the United Verde Copper Co. and the United Verde Extension Mining Co. at Jerome, the Ray Consolidated Copper Co. at Ray, the Clifton-Morenci District, and the new developments of the Gold Road-Oatman District.

Two special cars started from New York City carrying members of the party as far as Chicago. There the numbers were augmented by members from different points. The Chairman and representatives of the Chicago Local Section met the party and accompanied it on an interesting drive through Washington and Jackson Parks in automobiles. A banquet of about 75 persons at the La Salle Hotel followed.

Three special Pullman cars, the private car "Anaconda" belonging to Past-President Benjamin B. Thayer, and a special combination baggage and club car for the accommodation of the Institute party left Chicago on the Golden State Limited of the Chicago, Rock Island and Pacific Railroad, but some time before morning was made up as the Institute Special which then traveled as a separate unit through to the Grand Canyon, Ariz. Through the kindness of Mr. Carl Scholz, large geological maps showing the mineral resources of the Rock Island Lines between Chicago and El Paso were distributed to all members of the party and added greatly to the interest of the trip.

Additional members were picked up at Kansas City and other points and the special train arrived in El Paso at 2:40 p. m. on Sunday, Sept. 17. At this point parties from Montana, Utah, Colorado, California, and other places, joined the gathering, as well as more than 25 members from Arizona, including the Arizona Committee. Several El Paso members, under the leadership of their very efficient Committee, had arranged for a visit by automobiles to the National Guard camps—containing, it was said, about 50,000 militia from different States—to Fort Bliss, and to the El Paso Smelter of the American Smelting and Refining Co. Here the party had an opportunity to see a lead smelter which had been transformed to a copper smelter to meet commercial conditions and also a Peirce-Smith copper converter 13 ft. in diameter. Some of the members motored over to Old Mexico and collected souvenirs and saw a bull fight.

After a brief period for rest the members and guests were tendered by the El Paso members a most interesting Mexican supper at the Toltec Club, at which were present 179 persons and which was enlivened by the music of a Mexican band. The menu of this supper follows: Toronja; Enchiladas con Huevos; Tamales de Pollo; Tortillas; Frijoles Refritos; Nieve Napolitana; Quequi; Cafe. To each menu card was pinned a small Mexico sombrero made of horse hair.

The party leaving El Paso on Sunday evening numbered 150 persons, and the arrangements from that point were in the capable hands of the Arizona General Committee. Attractive, well printed, and well illus-

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trated general programs of the meeting were distributed which contained a map of Arizona showing the principal points of mining and metallurgical interest. In this booklet there were also some statistics and information regarding the mines of the Chino Copper Co. and of the Warren District and the smelters at Douglas. Throughout the entire trip the admirable arrangements of the General Committee and the Local Committees of the different districts were a frequent source of comment. Not only was every possible comfort and convenience provided, but carefully prepared data on each of the districts were given to the visitors upon arrival.

The map used as frontispiece in this *Bulletin* will be of interest to readers of the following pages.

Monday morning, Sept. 18, found the party at Santa Rita, N. Mex., where the members were awakened by a miners' salute consisting of a

FIG. 1.—THE OBSERVATION TRAIN WHICH CARRIED THE INSTITUTE PARTY AROUND THE CHINO COPPER CO. OPEN-CUT WORKINGS.

series of blasts at the mine of the Chino Copper Co. Upon detraining the party boarded an observation train elaborately decorated for the occasion, as shown in Fig. 1, and was taken around the open-cut workings of the Chino mine, estimated to contain about 90 million tons of porphyry ore averaging about 1.75 per cent. copper. This property is said to be the first in the Southwest to be mined by steam shovel, and, in the year 1915, produced an average of over 200,000 lb. of copper per day.

What can be described only as a fleet of automobiles, which had been gathered from mining districts within a radius of 30 miles, was then ready to take the members to Hanover, where are located the mine and mill of the Empire Zinc Co. These arrangements are typical of the thoroughness and thoughtfulness with which the Committees at all the different points had arranged to give their visitors the maximum opportunity to see the points of technical interest with the minimum of fatigue. Those who took this trip were especially interested

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in the Rowand-Wetherill magnetic separator treating an ore containing zinc and lead. Those of the party who did not go to Hanover visited the workings of the Chino mine and also the large preliminary crushing plant at the mine, but all foregathered at Santa Rita about noon to view the "Chino Bar for Ladies and Gentlemen" and the interesting relics of ancient Spanish mining, and to listen to the music provided by the excellent band of the Eleventh U. S. Cavalry. The tennis courts of the Chino Copper Co. had been covered by a large tent decorated with bunting and laid with tables for 200 guests. Here a splendid barbecue was enjoyed by the visitors. The party then had the option of going either by automobile or by the special train to Hurley, where is located the mill of the Chino company, giving the party its first view of flotation work on a large scale, as well as an opportunity to see tailing dams constructed to conserve both the waste water and the tailings

FIG. 2.—SCENE OF THE BARBECUE AT SANTA RITA, N. M.

which are to be subsequently treated in a flotation plant now being constructed.

A splendid dancing pavilion had been erected next to the tennis courts at Hurley, and after dinner many members of the party enjoyed dancing to the music of the Eleventh U. S. Cavalry band. The first day of the meeting closed leaving an impression in the minds of all that it would be very difficult to exceed the interest and pleasure that had been afforded.

Tuesday morning, Sept. 19, the Institute special train of 13 cars arrived at Douglas and was greeted by the Sixth U. S. Artillery band and another fleet of automobiles waiting to take the party to the Copper Queen smeltery. Some members took advantage of this opportunity and others went by the Institute special train, which was carried over the company's tracks and waited at the smeltery until the party had finished its visit there and then carried all the visitors to the plant of the

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Calumet and Arizona Co. All members of the party were particularly struck by the cleanliness of these smelters, as evidence of which may be mentioned that a buffet luncheon was served under the dust chambers of the Calumet and Arizona plant. After luncheon the party was carried by train to the Y. M. C. A. building, where a technical session was held on the subject of "Smelting." The ladies were entertained by an automobile ride and tea at the residence of Mrs. Forest Rutherford. In the evening the second technical session, on the subject of "Leaching," was held, and the ladies were entertained with dancing and a band concert at the Country Club. It is interesting to note that, in spite of the other attractions, the attendance at the technical sessions was invariably large, there being 175 persons at the opening session and 150 persons at the evening session.

FIG. 3.—SHOWING THE DUST BINS OF THE CALUMET AND ARIZONA COMPANY, DOUGLAS, ARIZ., UNDER WHICH A BUFFET LUNCHEON WAS SERVED.

The 30 miles to the Warren District were made at night and the special train arrived at Lowell in the morning, where the members were given a choice of four different trips, viz.: (1) Underground inspection of the Calumet and Arizona Mining Co.'s Junction shaft and thence to the Briggs mine and the Tintown vein; (2) Neptune tunnel of the Copper Queen Mine; (3) the Sacramento main shaft or (4) a surface trip, including the hoisting plant of the Sacramento shaft, the surface ore-loading plant, the central power house, the central tool-sharpening plant and the central sawmill of the Copper Queen, the Oliver power plant and the Junction power plant of the Calumet and Arizona, together with the underground pumping stations of the latter company, which handles the water from all the mines of the district.

In the afternoon a technical session on "Geology and Mining" was held in the main auditorium of the high school, there being about 300 persons present. The ladies were entertained at luncheon at the Copper Queen Hotel. After a brief period for rest, the members of the party

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were taken to the Warren District Country Club. Adjacent to this club is Camp John C. Greenway of the Twenty-second Regiment of U. S. Infantry and the District of Columbia militia. This regiment gave a dress parade in honor of the Institute party, affording a beautiful sight at the mouth of the Canyon overlooking the broad San Pedro Valley as the sun sank behind the distant mountains. This was followed by a banquet at the Country Club, attended by 280 persons. This banquet was a truly memorable occasion and reminded those members who were with the Institute party at the stop in Arizona in the year 1899 of the extraordinary entertainment afforded them on that occasion.* The food had to be brought by automobile from Bisbee, but was served in perfect condition notwithstanding that the number of persons greatly exceeded the expectations of the Local Committee and equally overtaxed the facilities of the Club.

Philip N. Moore, Vice-President of the Institute, acted as toastmaster and the following toasts were responded to by the speakers indicated:

"Welcome to Arizona," C. T. Knapp.
"Arizona, Old and New," L. D. Ricketts.
"Arizona Present," Walter Douglas.
"The American Mining Congress," Carl Scholz.
"Why We Are Here," Colonel Tillson.
"Ave et Vale," John Mason Ross.

FIG. 4.—THE INSPIRATION CONCENTRATOR WITH THE INTERNATIONAL SMELTING PLANT IN BACKGROUND.

The special train met the party at a near-by junction and to the other 13 cars was now added the private car "Nacozari" of Mr. Walter Douglas. The train proceeded by night to Globe, Ariz. Here, on Thurs-

*Described in *Trans.*, vol. 29, p. lxxxviii.

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day morning, Sept. 21, automobiles were in waiting to convey the party to the Old Dominion Copper Mining and Smelting Co's works, where members had an opportunity to see the basic converter which had already been described by L. O. Howard and in which 70,000,000 lb. of copper had been made without relining. In the afternoon a technical session was held at the Martin theater on the subject of "Concentration and Flotation," attended by about 200 persons, while the ladies were entertained at the Cobre Valle Country Club. The session was followed by a meeting of Secretaries of Local Sections, of whom nine were present. The Board of Directors met at dinner at the Old Dominion Hotel, and a technical session on the subject of "Mining" was held later at the Martin Theater, at which 220 persons were present.

On Friday morning, Sept. 22, the special train conveyed some of the party, and others went in automobiles, to the mine of the Inspiration Consolidated Copper Co. and the reduction works of the International

FIG. 5.—PLANT OF THE INTERNATIONAL SMELTING CO., MIAMI, ARIZ.

Smelting Co. This occasion was the greatest technical visit of the meeting. The plants have been so well described already in our *Transactions* that further comment is unnecessary here. To the admirable arrangements of the Local Committee is due the fact that ample opportunity was given to take in the many different points of interest in the very limited time available. An abundant supply of automobiles was available at all times to facilitate the handling of the party, while the special train took care of the main group. After the morning spent at the Inspiration and International plants, the visitors were taken to the mine and mill of the Miami Copper Co., where an excellent buffet luncheon was served at the Club House, followed by a visit to the plants. The company has good reason to be proud of the fine club house provided for its employees, and the Institute party has equally reason to congratulate itself upon having so comfortable a place for luncheon and for the technical session on "Fine Grinding," which occurred at 4:00 p. m. This session was attended by 160 persons who carried on so lively a discussion

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that the Chairman was obliged to cut it short in order that the members might take the special train for Globe, which left at 5:30 p. m.

While the men were inspecting the mine and mill and attending the technical session, the ladies were entertained at the bowling alleys of the club house and were then given an opportunity to swim in the crystal-like swimming pool.

At 7:30 p. m. more than 200 members and guests assembled in the auditorium of the Globe high school for the final banquet of the meeting. Sidney J. Jennings, First Vice-President of the Institute, acted as toastmaster and toasts were responded to as follows:

"Safety First," John E. Bacon.
"The Growth of the Globe District," L. D. Ricketts.
"Why the Young Engineer Should Join the Institute," E. P. Mathewson.
"The Utilization of Waste Products," F. G. Cottrell.

The meeting resolved itself into a spontaneous tribute to President Ricketts. His address as reported by the local paper follows:

"While copper and silver in the Globe District may have been known at an earlier date, the earliest actual records show that the Globe and Globe Ledge claims were located as silver mines in 1873. They were not worked, however, and in 1875 some rich silver mines were discovered, the Globe District was organized, and silver ores were milled and silver bullion shipped.

"The first copper was produced at Wheatfields from ore from the Hoosier claim, which was smelted in Mexican adobe furnaces in 1878. The campaign was but a short one, and some 40 tons of black copper pigs was the result. A little later on Mr. John Williams, Sr., worked the Carrie mine and smelter. His ore was a quartzite containing 6 per cent. copper and little or no iron. It is said that he purchased his flux from the big iron outcrop on the Globe claim, and that he made a profit. Incidentally it is said that the only impurity in the hematite was oxide of copper to the extent of about 25 per cent.

"This, it is said, was the discovery of the first important orebody in the district. This and other mining claims, consolidated into what is now known as the Old Dominion, were bought by Baltimore and Boston people and worked quite extensively considering the fact that the coke had to be brought in from Willcox, 140 miles distant, in wagons, and the black copper pigs had to be hauled back.

"My first visit to the camp was in 1890, when Mr. Douglas began the purchase of claims which later formed the United Globe Mines. At this time Professor Walker was superintendent of the Old Dominion mine. A little later some rich and very siliceous and aluminous ores were found in the region near what is now the Miami camp. The Black Warrior shipped rich chrysocolla ore to the Old Dominion smelter in 1895, and later the Keystone and Live Oak, which were located in about 1897, began to ship even more siliceous material to the same works.

"The railway from Bowie, several years in building, reached Globe in the latter part of 1898.

"In 1903 the Old Dominion Copper Mining & Smelting Co. of New Jersey was practically bankrupt. The Old Dominion Copper Mining & Smelting Co. of Maine was formed, and acquired most of the stock of the New Jersey company and all of the stock of the United Globe Mines. In exchange for the latter stock, and with treasury stock bought for cash, gentlemen associated with Phelps, Dodge & Co. took control. A more liberal policy was adopted, profits for the time being were put back into the mine and works, and in four years' time the stock rose from less than $5 to over $60 per share. Ever since that time this mine has been prosperous and profitable, and in your visit you noticed the splendid shape it is in and the great ability of the management.

"Returning to the new district near Miami, in 1903 and 1904, Mr. Coplen, in behalf of himself and others, bonded and purchased a few mining claims which became the beginning of the Inspiration group, and he opened up a small block of lean disseminated ore. In 1907, Mr. J. Parke Channing bonded the Miami group of claims, and developed for himself and his principals the great Miami mine.

"Shortly after, others purchased the Coplen property and many other claims and formed it into the Inspiration Copper Co. and started development on a considerable scale. Hovland and Smith also took over the Live Oak group of claims and like-

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wise began development, while parties affiliated with the Miami shareholders took over the Keystone group. The Miami company, after developing their mine, undertook the construction of their splendid plant, which up to that date excelled anything that had ever been built. As it was necessary for them to sell their concentrates they made a long-time contract with Cananea on very favorable terms. A little later, in 1911 as I recall it, the International Smelting & Refining Co. closed a long-time contract for Inspiration concentrates, and later on interests affiliated with this company purchased the Live Oak group and consolidated with the Inspiration Copper Co. into what is known as Inspiration Consolidated Copper Co. This consolidation occurred in March, 1912, and Mr. C. E. Mills accepted the management of the property.

"At first it was planned to use gravity concentration, and construction was begun on a building designed for that process. Late in 1912, however, Dr. Gregory of the Mineral Separations company advised us that his firm had had splendid results on our ore in their laboratory by flotation. It was therefore arranged that he was to send us a 50-ton machine to Inspiration. Dr. Gahl's most able paper tells the rest of the flotation story.

"After taking the two long-term contracts from Miami and Inspiration, it became evident to the companies interested that the high freight rates to Cananea made it

FIG. 6.—ROOSEVELT DAM FROM ABOVE.

desirable, and the International smelter now in operation at Miami was decided upon and built.

"In 10 short years the output of the Globe district has sprung from about 30,000,000 lb. of copper a year to about 230,000,000 lb. a year, and the works you have seen today have been put in operation.

"In closing this history, gentlemen, I cannot help telling an anecdote. Both Mills and I once worked for Phelps, Dodge & Co. for many years. For some reason many believe that I still work for them, though I left those good friends 10 years ago. Today I met a gentleman who knew us both in the old days and we were looking over the splendid work at Inspiration, when he said, 'What I can't understand is, how did you let Mills get away from you?' Gentlemen, I had not overlooked a bet. Referring to the puns of a newspaper cartoonist, I had early felt that Mr. Mills was indubitably to be the 'spir' in Inspiration.

"And now, gentlemen, our trip is wellnigh over. We have seen all of the black copper pigs we are going to see, and I feel sure you feel satisfied and rewarded for your long, arduous trip. In leaving Globe, therefore, I feel that it is in order and fitting that I, as your president, should tender our thanks to our hosts, the executive committee, the local committees, and the individual members of the Arizona Section, which covers, I understand, New Mexico and El Paso members."

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At an early hour on Saturday morning, Sept. 23, the Institute party started in automobiles over the famous Apache Trail, which leads for about 50 miles over the plains to the Roosevelt Dam, passing en route cliff dwellings which can be plainly seen from the road. From the Roosevelt Dam, which was an object of considerable interest to all the engineers, to the City of Phoenix, a distance of about 70 miles, the road travels along the gorge of the Salt River and then ascends to a mesa through heavily eroded gorges which form some of the most spectacular scenery in Arizona. The road itself is a real engineering feat, having been built by the Government for the purpose of carrying in supplies for the building of the Roosevelt Dam. As the party emerged from the mountains and came out into the Western Plain with the sun approaching the horizon, the scenery was indescribably beautiful.

Several hours were spent in Phoenix visiting with local members and partaking of a dinner which had been arranged for the visitors. The special train had been brought around by rail and late in the evening the party started for the Grand Canyon, where all arrived several hours late, but still very happy, early Sunday afternoon. The remainder of Sunday and Monday were spent visiting different points on the rim or on the trip down the Canyon. The party was now considerably diminished in number, various members having left for different points of the compass, so that only three special cars departed for the East on Monday night. Owing to a derailment to the west of Williams, the cars were held at that town until early Tuesday afternoon, but the waiting time was spent in an exciting game of baseball, and no one seemed to regret delaying the time for bringing to a close the 113th Meeting of the Institute.

The following members and guests registered at different points during the meeting, but the list is known to be incomplete as a good many attended different functions without registering:

L. M. ALLEN, Globe, Ariz.
R. S. ALLEN, Globe, Ariz.
MRS. R. S. ALLEN, Globe, Ariz.
J. OWEN AMBLER, Douglas, Ariz.
J. J. AMBROSE, Hayden, Ariz.
F. T. ANDERSON, El Paso, Tex.
C. E. ARNOLD, Miami, Ariz.
FRANK AYER, Miami, Ariz.
PERCY E. BARBOUR, New York, N. Y.
I. H. BARKDOLL, Globe, Ariz.
MRS. I. H. BARKDOLL, Globe, Ariz.
GEORGE D. BARRON, Rye, N. Y.
MRS. GEORGE D. BARRON, Rye, N. Y.
P. G. BECKETT, Globe, Ariz.
A. D. BEERS, New York, N. Y.
FRED BELY, Superior, Ariz.
ALLEN T. BIRD, Nogales, Ariz.
G. N. BJORGE, Globe, Ariz.
L. A. BLACKNER, Ray, Ariz.
F. C. BLICKENSDERFER, Morenci, Ariz.
A. L. BLOOMFIELD, Denver, Colo.
W. B. BOGGS, New York, N. Y.
MRS. M. BOOKMAN, St. Louis, Mo.
H. P. BOWEN, Miami, Ariz.
MRS. H. P. BOWEN, Miami, Ariz.
R. R. BOYD, Globe, Ariz.
M. L. BRADT, Saginaw, Mich.
S. D. BRIDGE, Comfort, Tex.
C. J. BRIGGS, El Paso, Tex.
J. F. BROWN, Goldfield, Nev.
W. C. BROWNING, Superior, Ariz.
D. W. BRUNTON, Denver, Colo.
H. A. BUEHLER, Rolla, Mo.
EDWARD E. BUGBEE, Boston, Mass.
W. BURNS, Morenci, Ariz.
A. B. CALHOUN, Globe, Ariz.
K. P. CAMPBELL, Sasco, Ariz.
NORMAN CARMICHAEL, Clifton, Ariz.
CHARLES A. CHASE, Denver, Colo.
WILL L. CLARK, Jerome, Ariz.
W. M. CLAYPOOL, Needles, Cal.
BEN H. CODY, Clifton, Ariz.
CARL H. COLE, Douglas, Ariz.
DAVID COLE, El Paso, Tex.
MRS. DAVID COLE, El Paso, Tex.
G. M. COLVOCORESSES, Humboldt, Ariz.
H. COOPER, El Paso, Tex.
F. G. COTTRELL, Washington, D. C.
W. B. CRAMER, Globe, Ariz.
ARTHUR CROWFOOT, Morenci, Ariz.
JOSEPH F. CULLEN, Midvale, Utah.
ARTHUR C. DAMAN, Denver, Colo.
S. H. DAVIS, Joplin, Mo.
E. G. DEANE, Miami, Ariz.
GEORGE C. DEWEY, Selby, Cal.
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H. S. DUNCAN, Globe, Ariz.
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M. ELSASSER, Los Angeles, Cal.
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PERCY LeR. FEARN, New York, N. Y.
LEON FEUCHÈRE, Warren, Ariz.
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F. N. FLYNN, Clifton, Ariz.
J. G. FLYNN, Miami, Ariz.
PAUL R. FORBES, New York, N. Y.
ROBERT FRANKE, Miami, Ariz.
W. E. GABY, Butte, Mont.
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W. I. GARMS, Hayden, Ariz.
W. F. GEIGER, Miami, Ariz.
R. J. GLENDINNING, Salt Lake City, Utah.
W. B. GOHRING, Warren, Ariz.
C. W. GOODALE, Butte, Mont.
WILLIAM D. GORDON, El Paso, Tex.
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W. S. HALL, Miami, Ariz.
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JAMES W. HAMBLETON, El Paso, Tex.
A. M. HAMILTON, Sasco, Ariz.
A. M. L. HAMILTON, Sasco, Ariz.
H. T. HAMILTON, Nacozari, Mex.
H. O. HAMMOND, Douglas, Ariz.
R. S. HANDY, Kellogg, Idaho.
A. B. HARDIE, Philadelphia, Pa.
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WALTER HARRIS, Globe, Ariz.
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JAS. L. HEAD, Warren, Ariz.
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MURRAY HENDRIX, Miami, Ariz.
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F. G. HILLS, Leadville, Colo.
F. W. HOAR, Globe, Ariz.
E. N. HOBART, Nogales, Ariz.
J. P. HODGSON, Bisbee, Ariz.
C. E. HOGUE, Globe, Ariz.
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MISS AMY S. JENNINGS, New York, N. Y.
MISS MARY A. JENNINGS, New York, N. Y.
FRANK E. JOHNSON, Salt Lake City, Utah.
WILLIAM STRICKLER JONES, Atlantic City, N. J.
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D. N. KAY, Ray, Ariz.
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R. W. KERNS, Warren, Ariz.
S. J. KIDDER, Mogollon, New Mex.
R. B. T. KILIANI, New York, N. Y.
K. L. KITHIL, Tucson, Ariz.
EDWARD H. KOENIG, Perth Amboy, N. J.
A. S. KONSELMAN, Cananea, Mex.
J. KRUTTSCHNITT, JR., Tucson, Ariz.
O. M. KUCHS, Tooele, Utah.
C. R. KUZELL, Anaconda, Mont.
JOHN LANGTON, New York, N. Y.
E. H. LAWS, Salida, Colo.
C. LEGRAND, Douglas, Ariz.
S. H. LEVISON, Hayden, Ariz.
W. W. LOGUE, Hayden, Ariz.
B. M. McATEE, Miami, Ariz.
ALAN F. McCORMICK, El Paso, Tex.
W. E. McCOURT, St. Louis, Mo.
GEO. T. McGEE, Helena, Mont.
A. G. McGREGOR, Bisbee, Ariz.
P. M. McHUGH, Denver, Colo.
R. McINTOSH, Lake Linden, Mich.
MRS. R. McINTOSH, Lake Linden, Mich.
WILL E. McKEE, Bisbee, Ariz.
WILLIAM T. MacDONALD, Hayden, Ariz.
F. W. MACLENNAN, Miami, Ariz.
J. F. MANNING, Holkol, Korea.
E. R. MARBLE, Hayden, Ariz.
EMORY M. MARSHALL, Globe, Ariz.
MRS. EMORY MARSHALL, Globe, Ariz.
E. P. MATHEWSON, Anaconda, Mont.
MRS. E. P. MATHEWSON, Anaconda Mont.
E. V. MATLACK, Webster Grove, Mo.
ELLWOOD V. MATLACK, JR., Webster Grove, Mo.
M. S. MAZANY, Miami, Ariz.
A. H. MEANS, Tucson, Ariz.
H. I. MERRICKS, Miami, Ariz.
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H. W. MORSE, Los Angeles, Cal.
McHENRY MOSIER, Bisbee, Ariz.
P. A. MOSMAN, New York, N. Y.
SEELEY W. MUDD, Los Angeles, Cal.
H. T. MURRAY, Hayden, Ariz.
R. T. MURRILL, Flat River, Mo.
HENRY W. NICHOLS, Chicago, Ill.
H. L. NORTON, Globe, Ariz.
ARTHUR NOTMAN, Bisbee, Ariz.
T. H. O'BRIEN, Dawson, N. Mex.
J. J. ORMSBEE, El Paso, Tex.
H. D. PALLISTER, El Paso, Tex.
J. H. PAYNE, New York, N. Y.
MRS. J. H. PAYNE, New York, N. Y.
E. F. PELTON, Tyrone, N. Mex.
BASIL PRESCOTT, El Paso, Tex.
R. J. PRITCHARD, Miami, Ariz.
WILLIAM J. QUIGLY, El Paso, Tex.
O. C. RALSTON, Salt Lake City, Utah.
F. L. RANSOME, Salt Lake City, Utah.
STUART L. RAWLINGS, San Francisco, Cal.
WILLIAM H. REA, Pittsburgh, Pa.
MRS. WM. H. REA, Pittsburgh, Pa.
MISS REA, Pittsburgh, Pa.
E. E. REYER, El Paso, Tex.
C. E. RHODES, Pasadena, Cal.
MISS MARION RICE, Schenectady, N. Y.
WALTER A. RICHELSEN, Cananea, Sonora, Mex.
L. D. RICKETTS, Warren, Ariz.
MRS. L. D. RICKETTS, Warren, Ariz.
EZRA B. RIDER, Bisbee, Ariz.
A. E. RING, Flat River, Mo.
J. F. ROBERTSON, Coniston, Ont.
BURR A. ROBINSON, New York, N. Y.
E. M. ROBINSON, So. Bethlehem, Pa.
CLYDE P. ROSS, Globe, Ariz.
E. W. ROUSE, Baltimore, Md.
G. H. RUGGLES, Miami, Ariz.
J. P. RUMINAPP, Miami, Ariz.
B. E. RUSSELL, Ray, Ariz.
F. RUTHERFORD, Douglas, Ariz.
E. M. SAWYER, Tyrone, N. M.
FRANCIS S. SCHIMERKA, Clifton, Ariz.
WALTER A. SCHMIDT, Los Angeles, Cal.
CARL SCHOLZ, Chicago, Ill.
C. D. SCHULTZ,
W. SCHUMACHER, El Paso, Tex.
MORTIMER A. SEARS, Santa Fe, New Mex.
GERALD SHERMAN, Bisbee, Ariz.
L. B. SHIPLEY, New York, N. Y.
ALEX SIBBALD, Globe, Ariz.
H. R. SIMPSON, Los Angeles, Cal.
K. M. SIMPSON, Los Angeles, Cal.
HOWARD D. SMITH, San Francisco, Cal.
E. G. SNEDAKER, Goldfield, Nev.
F. W. SOLOMON, Miami, Ariz.
P. G. SPILSBURY, New York, N. Y.
E. M. STEELE, Hayden, Ariz.
P. A. STEGER, Miami, Ariz.
PAUL STEIN, El Paso, Tex.
PAUL STERLING, Wilkes-Barre, Pa.
E. D. STEWART, El Paso, Tex.
BRADLEY STOUGHTON, New York, N. Y.
MRS. BRADLEY STOUGHTON, New York, N. Y.
ROGER STROBEL, East Helena, Mont.
WILMER C. SWARTLEY, Philadelphia, Pa.
W. S. SULTAN, Globe, Ariz.
JOHN C. TAYLOR, Denver, Colo.
KNOX TAYLOR, High Bridge, N. J.
ROSCOE TEATS, Tacoma, Wash.
B. B. THAYER, New York, N. Y.
O. J. TUSCHKA, Globe, Ariz.
MRS. O. J. TUSCHKA, Globe, Ariz.
G. D. VAN ARSDALE, New York, N. Y.
R. E. VINING, Perth Amboy, N. J.
JOHN D. WANVIG, Golconda, Ariz.
HARRY S. WARE, Anaconda, Mont.
J. H. WATKINS, Washington, D. C.
ARTHUR P. WATT, St. Francois, Mo.
A. J. WEINIG, Telluride, Colo.
HARRY V. WELCH, Los Angeles, Cal.
GEORGE H. WEST, Globe, Ariz.
J. R. WESTER, Morenci, Ariz.
CHARLES H. WHITE, Cambridge, Mass.
J. L. WHITE, Humboldt, Ariz.
H. E. WILLIAMS, Calumet, Mich.
MRS. H. E. WILLIAMS, Calumet, Mich.
J. S. WILLIAMS, Nacozari, Mex.
BAILEY WILLIS, Stanford Univ., Cal.
PHILIP D. WILSON, Warren, Ariz.
PHILIP WISEMAN, Los Angeles, Cal.
MRS. PHILIP WISEMAN, Los Angeles, Cal.
C. L. WOLFLE, El Paso, Tex.
J. R. WOODUL, Mexico.
WM. H. YEANDLE, JR.
R. B. YERXA, Miami, Ariz.
H. M. ZIESEMER, Bisbee, Ariz.

TECHNICAL SESSIONS

The full list of papers which were presented by their authors or by title at the meeting is as follows:

Mining

MINE ACCOUNTING FOR SMALL MINES. By James E. Chapman.

AUTOMATIC OPERATION OF MINE HOISTS AS EXEMPLIFIED BY THE NEW ELECTRIC HOISTS FOR THE INSPIRATION CONSOLIDATED COPPER CO. By H. Kenyon Burch and M. A. Whiting.

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COMPARATIVE FRICTION TEST OF TWO TYPES OF COAL MINE CARS. By P. B. Liebermann.
THE WATER PROBLEM AT THE OLD DOMINION MINE. By P. G. Beckett.
THE COMPOSITION OF THE ROCK GAS OF THE CRIPPLE CREEK MINING DISTRICT, COLORADO. By George A. Burrell and Alfred W. Gauger.
THE SOLUTION OF SOME HYDRAULIC MINING PROBLEMS ON RUBY CREEK, BRITISH COLUMBIA. By Chester F. Lee and T. M. Daulton.
THE RIFLING OF DIAMOND-DRILL CORES. By Walter R. Crane.
METHOD OF MINING TALC. By F. R. Hewitt.
STOPING IN THE CALUMET AND ARIZONA MINES, BISBEE, ARIZ. By Philip D. Wilson.
STOPING METHODS OF MIAMI COPPER CO. By David B. Scott.
COÖPERATIVE EFFORT IN MINING. By Joseph P. Hodgson.
DIESEL ENGINES VERSUS STEAM TURBINES FOR MINE POWER PLANTS. By Herbert Haas.
MODERN METHODS OF MINING AND VENTILATING THICK PITCHING BEDS. By H. M. Crankshaw.
MOTOR TRUCK OPERATION AT MAMMOTH COLLINS MINE, SHULTZ, ARIZ. By Wilbert G. McBride.
MINE FIRE METHODS EMPLOYED BY THE UNITED VERDE COPPER CO. By Robert E. Tally.
COST AND EXTRACTION IN THE SELECTION OF A MINING METHOD. By C. E. Arnold.
THE ILLUMINATING POWER OF SAFETY LAMPS. By W. M. Weigel.
POWER PLANT OF BURRO MOUNTAIN COPPER CO. By Charles Legrand.
ORE-DRAWING TESTS AND THE RESULTING MINING METHOD OF INSPIRATION CONSOLIDATED COPPER CO. By G. R. Lehman.
SHAFT SINKING THROUGH SOFT MATERIAL. By Edward A. Sayre.
THE BLOCK METHOD OF TOP SLICING OF THE MIAMI COPPER CO. By E. G. Deane.
THE ANTECEDENT MINERAL DISCOVERY REQUIREMENT. By E. D. Gardner.

Geology and Mineralogy

PETROGRAPHY OF THE MOUNT MORGAN MINE, QUEENSLAND. By W. E. Gaby.
GEOLOGY OF THE WARREN MINING DISTRICT. By Y. S. Bonillas, J. B. Tenney and Leon Feuchère.

Cyanidation

LABORATORY METHOD FOR DETERMINING THE CAPACITY OF SLIME-SETTLING TANKS. By H. S. Coe and G. H. Clevenger.
THE LIBERTY BELL METHODS OF PRECIPITATE REFINING. By A. J. Weinig.
MINING AND MILLING PRACTICE AT SANTA GERTRUDIS. By Hugh Rose.
CYANIDING CLAYEY ORE AT THE BUCKHORN GOLD MINE. By Paul R. Cook.

Miscellaneous

CALCULATIONS WITH REFERENCE TO THE USE OF CARBON IN MODERN AMERICAN BLAST FURNACES. By Henry Phelps Howland.
THE APPLICATION AND EARNING POWER OF CHEMISTRY IN THE COAL MINING INDUSTRY. By Edwin M. Chance.
THE SYSTEM TUNGSTEN-MOLYBDENUM. By Frank Alfred Fahrenwald.
COMPARISONS BETWEEN ELECTROLYTIC COPPER AND TWO VARIETIES OF ARSENICAL LAKE COPPER WITH RESPECT TO STRENGTH AND DUCTILITY IN COLD-WORKED AND ANNEALED TEST STRIPS. By C. H. Mathewson and E. M. Thalheimer.
TUNGSTEN AND MOLYBDENUM EQUILIBRIUM DIAGRAM AND SYSTEM OF CRYSTALLIZATION. By Zay Jeffries.

Flotation and Ore Dressing

FLOTATION CONCENTRATION AT ANACONDA, MONT. By Frederick Laist and Albert E. Wiggin.
THE FLOTATION OF MINERALS. By Robert J. Anderson.
AN EXPLANATION OF THE FLOTATION PROCESS. By A. F. Taggart and F. E. Beach.
A NEW FLOTATION OIL. By Maxwell Adams.

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A NEW SOURCE OF FLOTATIVE AGENTS. By G. H. Clevenger.
HISTORY OF THE FLOTATION PROCESS AT INSPIRATION. By Rudolf Gahl.
SOME MISCELLANEOUS WOOD OILS FOR FLOTATION. By R. C. Palmer, Glenn L. Allen and O. C. Ralston.
THE ADVENT OF FLOTATION IN THE CLIFTON-MORENCI DISTRICT, ARIZONA. By David Cole.
A COMBINED HYDRAULIC AND MECHANICAL CLASSIFIER. By M. G. F. Söhnlein.
COMPARATIVE TEST OF THE MARATHON, CHILEAN AND HARDINGE MILLS. By F. C. Blickensderfer.
MINE AND MILL PLANT OF THE INSPIRATION CONSOLIDATED COPPER CO. By H. Kenyon Burch.

Smelting, Etc.

THE DECOMPOSITION AND REDUCTION OF LEAD SULPHATE AT ELEVATED TEMPERATURES. By W. Mostowitsch. Edited by H. O. Hofman.
DETERMINATION OF DUST LOSSES AT THE COPPER QUEEN REDUCTION WORKS. By J. Moore Samuel.
AN INVESTIGATION INTO THE FLOWING TEMPERATURES OF COPPER MATTES AND OF COPPER-NICKEL MATTES. By G. A. Guess and F. E. Lathe.
FEATURES OF THE NEW COPPER SMELTING PLANTS IN ARIZONA. By A. G. McGregor.
SMELTING AT THE ARIZONA COPPER CO.'S WORKS. By F. N. Flynn.
THE BASIC LINED CONVERTER IN THE SOUTHWEST. By L. O. Howard.

Leaching

2,000-TON LEACHING PLANT AT ANACONDA. By Frederick Laist and Harold W. Aldrich.
POSSIBILITIES IN THE WET TREATMENT OF COPPER CONCENTRATES. By Lawrence Addicks.
LEACHING TESTS AT NEW CORNELIA. By H. W. Morse and H. A. Tobelmann.

Ore Deposits

GOLD AND SILVER DEPOSITS OF NORTH AND SOUTH AMERICA. By Waldemar Lindgren.
FUEL IN TURKEY. By Leon Dominian.
MANGANESE ORES OF RUSSIA, INDIA, BRAZIL AND CHILE. By E. C. Harder.
THE EMERALD DEPOSITS OF MUZO, COLOMBIA. By Joseph E. Pogue.
THE RADIO-ACTIVITY OF ALLANITE. By L. S. Pratt.
ZIRCON-BEARING PEGMATITES IN VIRGINIA. By Thomas L. Watson.
IRON PYRITES DEPOSITS IN SOUTHEASTERN ONTARIO, CANADA. By P. E. Hopkins.

Petroleum and Gas

PRINCIPLES OF NATURAL GAS LEASEHOLD VALUATION. By S. S. Wyer.
THE CALIFORNIA GASOLINE INDUSTRY. By W. R. Hamilton.
THE DIASTROPHIC THEORY. By Marcel R. Daly.
THE POSSIBILITY OF DEEP SAND OIL AND GAS IN THE APPALACHIAN GEO-SYNCLINE OF WEST VIRGINIA. By David B. Reger.

Instead of attempting to present all of these papers for discussion, the Board of Directors, upon the recommendation of the Committee on Papers and Publications, selected those papers which could be adequately presented by their authors or authors' representatives and which lent themselves particularly to discussion. Six technical sessions were then established and each was devoted to a particular subject with the understanding that the appropriate papers would first be read and discussed and following that general discussion of the subject would be welcomed. Opportunity was also given at every session for any person present to call for any paper on the list to be brought up for discussion.

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The first technical session was devoted to the subject of "Smelting" and was held at the Y. M. C. A. building, Douglas, Ariz., Tuesday afternoon, Sept. 19, at 2 o'clock.

President Ricketts announced that the 113th Meeting of the Institute was opened and called upon Walter Douglas to preside. The presiding officer then called upon John C. Greenway, who delivered a cordial Address of Welcome to Arizona. In response, President Ricketts announced that as he was both guest and host at the Arizona Meeting, he would call upon Past President Benjamin B. Thayer, who thereupon responded cordially to the Address of Welcome.

The following papers were then presented by their authors:

FEATURES OF THE NEW COPPER SMELTING PLANTS IN ARIZONA. By A. G. McGregor. Discussed by L. D. Ricketts, E. P. Mathewson.

SMELTER OPERATING NOTES FROM THE ARIZONA COPPER CO., LTD. By F. N. Flynn.

THE BASIC LINED CONVERTER IN THE SOUTHWEST. By L. O. Howard. Discussed by Walter Douglas, E. P. Mathewson, Kuno Doerr.

DETERMINATION OF DUST LOSSES AT THE COPPER QUEEN REDUCTION WORKS. By J. Moore Samuel. Discussed by Walter Douglas, E. P. Mathewson, S. J. Jennings, C. E. Arnold, A. G. McGregor, L. D. Ricketts.

The second technical session was on the subject of "Leaching" and was held at the Y. M. C. A. building at Douglas, Ariz., on Tuesday evening, Sept. 19, at 8 o'clock. H. W. Morse presided.

President Ricketts opened the meeting by reading the following telegram from Dr. Douglas, which was greeted with applause:

"Please convey my hearty greetings to the Institute and my regrets that my health prevents my presence with you in person. It is nearly 17 years since we met around the dinner table in Bisbee and lunched together in the mine. Though we cannot anticipate eternal life for our mines you will be glad to find that it is more productive than it was and is enjoying a vigorous old age and renewing its youth by new discoveries."

and the President was authorized to respond to this in the name of the members present, as follows:

"The members of the Institute at this meeting, some two hundred and fifty in number, by special resolution unite with me in thanking you for your most welcome message of good-will. We owe so much to you and it is good to feel we have your thoughts and good wishes. We thank you and send you our greetings and best wishes for you and yours."

The following papers were then presented:

LEACHING TESTS AT NEW CORNELIA. By H. W. Morse and H. A. Tobelmann. (Presented by H. W. Morse.) (Discussed by G. D. Van Arsdale, S. J. Jennings, F. S. Schimerka, C. G. Grabill, F. N. Flynn, written discussion by Lawrence Addicks.)

2,000-TON LEACHING PLANT AT ANACONDA. By Frederick Laist and Harold W. Aldrich. (PRESENTED BY E. P. MATHEWSON.) (Discussed by F. N. Flynn, E. P. Mathewson.)

POSSIBILITIES IN THE WET TREATMENT OF COPPER CONCENTRATES. By Lawrence Addicks. (Presented by the Secretary.) (Discussed by F. N. Flynn.

GENERAL SUBJECT OF LEACHING was discussed by F. S. Schimerka, B. B. Gottsberger, H. E. Williams, G. D. Van Arsdale, and E. P. Mathewson.

The third technical session was on the subject of "Mining and Geology" and was held in the auditorium of the high school at Bisbee, Ariz., on the afternoon of Wednesday, Sept. 20, at 2 o'clock. Gerald F. G. Sherman presided.

The following papers were presented by their authors:

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PETROGRAPHY OF THE MOUNT MORGAN MINE, QUEENSLAND. By Walter E. Gaby. (Discussed by L. C. Graton.)

GEOLOGY OF THE WARREN MINING DISTRICT. By Y. S. Bonillas, J. B. Tenney, Leon Feuchère. (Presented by Y. S. Bonillas.) (Discussed by I. B. Joralemon, F. L. Ransome, Gerald Sherman, L. C. Graton.)

STOPING IN THE CALUMET AND ARIZONA MINES, BISBEE, ARIZ. By Philip D. Wilson.

COÖPERATIVE EFFORT IN MINING. By Joseph P. Hodgson. (Discussed by J. A. Ede, Gerald F. G. Sherman, C. W. Goodale, Charles A. Mitke, C. E. Arnold, D. W. Brunton.)

The fourth technical session was on the subject of "Concentration and Flotation" and was held at the Martin Theater, Globe, Ariz., on Thursday afternoon, Sept. 21, at 2 o'clock. C. E. Mills presided.

The following papers were presented:

THE ADVENT OF FLOTATION IN THE CLIFTON-MORENCI DISTRICT, ARIZONA. By David Cole.

HISTORY OF THE FLOTATION PROCESS AT INSPIRATION. By Rudolf Gahl. (Discussed by F. S. Schimerka, H. W. Morse, R. S. Handy, C. A. Chase, E. P Mathewson, David Cole, L. D. Ricketts, G. H. Ruggles, F. G. Cottrell; written discussion by Frederick Laist and R. C. Canby.)

SOME MISCELLANEOUS WOOD OILS FOR FLOTATION. By R. C. Palmer, Glenn L. Allen and O. C. Ralston. (Presented by G. L. Allen.)

FLOTATION CONCENTRATION AT ANACONDA, MONT. By Frederick Laist and Albert E. Wiggin. (Presented by E. P. Mathewson.) (Discussed by O. C. Ralston, David Cole, E. P. Mathewson, Rudolf Gahl, Norman Carmichael, W. B. Cramer, C. W. Merrill, R. S. Handy, A. P. Watt, B. B. Gottsberger, F. S. Schimerka.)

The fifth technical session was on the subject of "Mining" and was held at the Martin Theater, Globe, Ariz., on the evening of Sept. 21, at 8 o'clock. Percy G. Beckett presided.

The following papers were presented:

THE BLOCK METHOD OF TOP SLICING OF THE MIAMI COPPER CO. By E. G. Deane. (Discussed by J. A. Ede, J. P. Hodgson.)

STOPING METHODS OF MIAMI COPPER CO. By David B. Scott. (Presented by the Secretary.)

MINE FIRE METHODS EMPLOYED BY THE UNITED VERDE COPPER CO. By Robert E. Tally. (Presented by the Secretary.) (Written discussion by C. L. Berrien.) (Discussed by C. W. Goodale, J. A. Ede, Gerald Sherman, J. P. Hodgson.)

ORE DRAWING TESTS AND THE RESULTING MINING METHOD OF INSPIRATION CONSOLIDATED COPPER CO. By George R. Lehman.

COST AND EXTRACTION IN THE SELECTION OF A MINING METHOD. By C. E. Arnold. (Discussed by L. D. Ricketts, F. W. Maclennan.)

After the papers scheduled for the meeting had been presented, Walter A. Schmidt gave a brief abstract of a paper prepared by himself, entitled "Results Obtained with the Cottrell Process at Smelter of International Smelting Co., Inspiration, Ariz. and the Factory of Riverside Portland Cement Co., Riverside, Cal.," illustrating his remarks with stereopticon slides, and ending with a demonstration of the process.

The sixth technical session was on the subject of "Fine Grinding" and was held at the Club House of the Miami Copper Co., Miami, Ariz., on the afternoon of Friday, Sept. 22, at 4 o'clock. B. B. Gottsberger presided.

The following papers were presented by their authors:

POWER PLANT OF BURRO MOUNTAIN COPPER CO. By Charles Legrand. (Discussed by B. B. Gottsberger, S. J. Kidder, and John Langton.)

COMPARATIVE TEST OF THE MARATHON, CHILEAN AND HARDINGE MILLS. By F. C. Blickensderfer. (Discussed by B. B. Gottsberger, R. B. Yerxa, A. P. Watt, R. B. T. Kiliani, F. S. Schimerka, S. J. Jennings, R. S. Handy, C. W. Merrill, Robert Franke, F. J. H. Merrill.)

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REPORT OF THE NOMINATING COMMITTEE

The Committee on Nominations begs to submit the following names as its nominees for the respective offices indicated:

For President, SIDNEY J. JENNINGS, New York.
For Vice-Presidents, WALTER H. ALDRIDGE, New York; MARK L. REQUA, San Francisco.
For Directors, J. E. JOHNSON, JR., New York; ALLEN H. ROGERS, Boston; HOWARD N. EAVENSON, West Virginia; ROBERT M. RAYMOND, New York; WILLET G. MILLER, Canada.

The nomination for President has attracted the attention of a large number of members, as is evidenced by the receipt of many hundreds of letters and post cards. Most of these favored either Sidney J. Jennings or Philip N. Moore, each of whom received many cordial endorsements. Such a wide interest on the part of the members is a very wholesome sign, but where there is so much activity it is natural to expect a certain amount of criticism. Some of the letters indicate that the authors feel that the nomination of President should be left entirely to the Committee on Nominations without any effort by the members at large to influence them. Others feel that the nominations are largely controlled by a small group and that the members at large can have little influence. As a result of the experience of the last few months, a majority of the Committee on Nominations begs leave to suggest that the method of making nominations be changed so that two candidates, at least, for the various offices be nominated, and suggests that the Directors appoint a committee to investigate this question thoroughly with a view to making definite recommendations. In accordance with its opinion that the best interests of the Institute would be served by a change in the Constitution, allowing the nomination of two candidates, and in view of the fact that this Committee has no authority to make more than a single nomination for each office, considering the high standing of both Mr. Jennings and Mr. Moore and the large number of members favoring each of these gentlemen, this Committee respectfully suggests the desirability of the immediate nomination of Philip N. Moore by a petition of 25 members in accordance with the present Constitution. This Committee regrets its inability to make two nominations.

Respectfully submitted,

S. W. MUDD, *Chairman.*

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, OCT. 27, 1916

The report of the Comittee on Nominations was received and ordered to take the statutory course. Thereupon the chairman was authorized to appoint a committee of three members to report at the next meeting of this Board regarding the suggestions made by the Nominating Committee.

The following vote of thanks was then passed to the members of the Arizona Committees and others:

"WHEREAS, the Board of Directors of the American Institute of Mining Engineers has a deep appreciation of the splendid success of the Arizona meeting and

WHEREAS, the Board is aware that this success was brought about by the generous and efficient efforts of the members of the Local Committees, therefore it is

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RESOLVED, that the thanks of the Institute be and hereby are cordially extended to each and every member of the Committees in El Paso, New Mexico and Arizona, and to such others as participated."

The sum of $250 was appropriated to the Southern California Section for the year 1917.

President L. D. Ricketts and Past President B. B. Thayer were appointed representatives of the Board at the meeting of the Montana Section on Nov. 10, 1916.

The Institute agreed to pay its share on certain improvements being made on the fifth floor of this building.

The exchange of publications was authorized with a Russian Magazine known as "The Ore Messenger" and with the Geological Survey of Canada.

The following Committee was appointed to confer with a committee of the Mining and Metallurgical Society of America regarding the best method for the revision of the mining laws: J. R. Finlay, D. C. Jackling, Hennen Jennings and C. F. Kelley.

JOHN FRITZ MEDAL TO BE PRESENTED TO DR. ELIHU THOMSON

The John Fritz Medal was awarded in January, 1916, to Dr. Elihu Thomson, "For Achievements in Electrical Inventions, in Electrical Engineering, and in Industrial Development, and in Scientific Research."

The medal will be presented to Dr. Thomson at a meeting to be held in Boston on Friday evening, Dec. 8, 1916, in the Central Lecture Hall of the new buildings of the Massachusetts Institute of Technology.

The program of the evening will include adresses by John J. Carty, Chairman of the Presentation Committee of the Board of Award, E. W. Rice, Jr., President, General Electric Co., and Dr. Richard C. Maclaurin, President, Massachusetts Institute of Technology. The presentation will be made by Dr. Charles Warren Hunt, Chairman of the Board at the time the award was made, and the ceremonies will conclude with the response of Dr. Thomson.

The John Fritz Medal is awarded from time to time for notable scientific or industrial achievement, and was provided for in a fund subscribed in memory of the great engineering pioneer, John Fritz. The award of the medal is made by a permanent board composed of four members from each of four American National Engineering Societies, namely, the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Mining Engineers and the American Institute of Electrical Engineers. The members of the 1916 Board are:

Representing the A. S. C. E.,	CHARLES WARREN HUNT, JOHN A. OCKERSON, GEORGE F. SWAIN, CHARLES D. MARX.
Representing the A. S. M. E.,	JOHN R. FREEMAN, AMBROSE SWASEY, JOHN A. BRASHEAR, FREDERICK R. HUTTON.
Representing the A. I. M. E.,	ALBERT SAUVEUR, E. GYBBON SPILSBURY, CHARLES F. RAND, CHRISTOPHER R. CORNING.

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Representing the A. I. E. E., RALPH D. MERSHON, C. O. MAILLOUX, PAUL M. LINCOLN, JOHN J. CARTY.

The members of the four national engineering societies and others interested, including ladies, are cordially invited to be present at the presentation ceremonies.

AMERICAN INDUSTRIAL COMMISSION TO FRANCE

Joseph G. Butler, Jr., who represented this Institute on the American Industrial Commission to France, has presented a report to the Chairman of the Commission regarding the steel industry of France in war time. In this report Mr. Butler calls attention to the fact that only a few steel works were visited and that he thinks it probable that there are many plants in France more up-to-date than these. He noticed especially in the plants visited a dearth of labor-saving devices and not very great attention to the safety of the employees. He also noticed that the whole of the iron and steel interests in France are being used at the present time by the Government for war purposes, and he was told on good authority that the production has doubled since May, 1915. When the war is over, it is thought that France can send large quantities of iron ore from Algeria to the United States and take in return American coal to France, an interchange which would doubtless be of mutual advantage. In Mr. Butler's opinion the shortage of coal in France may be overcome to some extent by the use of water-power and by electrical processes for steel manufacture, and he believes when the war ends France will be able to compete with other countries in exports of general iron and steel products, this trade being facilitated by the port improvements at Bordeaux and Marseilles. The French labor situation is much better than that in the United States from the standpoint of labor unions, but the wages are less and many women are now being employed to do men's work. This has a tendency to keep wages low, but is probably a war expedient only. The coöperation among manufacturers in France is at a high level and both local and national organizations exist for the discussion and voicing of trade problems. The national body which is a combination of the individual organizations is the Comite des Forges de France and it is an interesting fact that many of the members of this body are office holders under the Government, or heads of Government Committees which are organized to look after the very business in which the individuals are engaged.

Although the first rush of Germans into France deprived France of 80 per cent. of her pig iron and 70 per cent. of her steel productivity, it is important to note that, by the use of low-grade ores not previously exploited to any great extent, France has been able to increase the production about 100 per cent. in the past year. Mr. Butler was told that there is not a single idle plant formerly engaged in the fabrication of steel that is not now in full operation and that, while France is at present coöperating in every way with England to win the war, it is the intention of her industrial leaders to form a closer union with the British oprducers after the war is over.

At Le Creusot welfare work is highly efficient There are good hospital facilities, good dwellings, and everything is done to bring the workman in close and harmonious relations with his employer.

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PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at the Institute headquarters during the period Oct. 10, 1916 to Nov. 10, 1916.

Edwin C. Bloomfield, Montreal.
A. H. Collbran, Seoul, Korea.
Eugene Coste, Calgary, Canada.
Fred Crabtree, Pittsburgh, Pa.
P. St. John Dixon, London, England.
A. E. Drucker, London, England.
F. N. Flynn, Clifton, Ariz.
Justice Grugan, Edwards, N. Y.
Fred A. Hale, Jr., Goodsprings, Nev.
P. S. Haury, Seoul, Korea.
Roy E. Hoffman, Hannibal, Mo.
E. C. Harder, Cienfuegos, Cuba.
B. Jaeger, London, England.
Herbert H. Lauer, Philadelphia, Pa.
Frank C. Laurie, Los Angeles, Cal.
O. T. Lempriere, Melbourne, Australia.
W. C. Phalen, Washington, D. C.
William H. Rea, Pittsburgh, Pa.
Thor Warner, Kingman, Ariz.

Arthur P. Allen has resigned his position with the Calumet & Hecla and is now connected with the Highland Mining Co., Ashcroft, B. C., Canada.

H. Foster Bain, editor of the *Mining Magazine* of London, has resigned his position.

Frederick K. Brunton has resigned from the staff of the American Smelting & Refining Co., Garfield, Utah, to accept the position of assistant superintendent of the Consolidated Arizona Smelting Co., Humboldt, Ariz.

Will L. Clark has resigned as manager of the United Verde Copper Co., Jerome, Ariz.

J. V. N. Dorr, President of the Dorr company, has been awarded a John Scott Legacy Medal and Premium by the City of Philadelphia, upon the recommendation of the Franklin Institute. The award is "in recognition of his invention of hydrometallurgical apparatus, the Dorr classifier, and the Dorr thickener."

Edward B. Durham, now superintendent of construction for the Mammoth Copper Mining Co., Kennett, Cal., has just completed the dismantling and removal of the main line of the U. S. Mines' aerial tramway from Bingham, Utah, and its installation under new conditions at the Stowell mine. The work involved the moving of 100 tons of machinery, the redesigning of all the tramway structures, the framing and erecting of 150,000 ft. B.M. of lumber, the installing of the machinery and the putting of the line into operation.

F. N. Flynn, for the past seven years connected with the Arizona Copper Co., Ltd., at Clifton, Ariz., has resigned his position as Superintendent of the Smelting Department, effective Oct. 31, and has entered the employ of the Chile Exploration Co., Chuquicamata, Chile.

Roger H. Hatchett, who has been Mr. Flynn's assistant for six years past, has been appointed Acting Superintendent of the Smelting Department.

Myron E. Fuller, formerly connected with George W. Fuller, Consulting Engineer of New York City, has accepted a position on the engineering staff of the City of Philadelphia, Pa., in connection with proposed sewage-disposal works.

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W. Earl Greenough, for five years manager of the Marsh Mining Co., Coeur D' Alene District, Idaho, resigned his position with that company on Oct. 1, to engage in independent practice, and has opened offices in the Old National Bank Bldg., Spokane, Wash. He will be assisted by S. B. Davis, formerly of Wallace, Idaho.

E. H. Hamilton has been appointed Consulting Metallurgist for the Trail smelter of the Consolidated Mining & Smelting Co.

Edwin Higgins has resigned from the Bureau of Mines and California Industrial Accident Commission to open an office in San Francisco as Safety and Efficiency Engineer.

J. L. McAllen has resigned his position as superintendent of the Gold Bullion mine, Knik, Alaska, to engage in private practice in Portland, Ore.

F. E. Marcy, inventor of the Marcy Ball Mill, has established offices in the Atlas Building, Salt Lake City, Utah.

Richard A. Parker has been elected chairman of the Mining Bureau of the Denver Civic and Commercial Association.

Francis P. Sinn, of the New Jersey Zinc Co., Palmerton, Pa., has been elected one of the directors of the National Safety Council, which has created a new section having to do with safety in copper, lead and zinc smelting.

E. Gybbon Spilsbury has gone to Cuba on professional work and will be absent about three weeks.

Persifor G. Spilsbury, general manager of the Aguacate Mines, Costa Rica, has been transferred to New York, with offices at 55 Liberty Street.

Robert E. Tally, superintendent of mines for the United Verde Copper Co. for eight years, has been appointed assistant general manager, but will continue to act as superintendent.

POSITIONS VACANT

No. 157. Superintendent for roasting and magnetic separating plant working on zinc concentrates. Must have mechanical experience and a working knowledge of electricity. Salary will depend on experience and results obtained. Location Canada.

No. 155. Metallurgical chemist familiar with copper and steel for company just starting. Must have had technical education and practical experience with both metals. Salary moderate at beginning, but will be increased as business grows.

No. 159. An experienced, up-to-date coal-mining engineer to open up and manage soft-coal mines in Spitzbergen, preferably a Scandinavian capable of handling Norwegian laborers. Employment beginning January.

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No. 165. Wanted immediately, metallurgist capable designing constructing and superintending new concentrating and smelting plant. Proposed production ten thousand tons lead, four thousand tons zinc annually. Location, Caucasus. Advise terms when could come.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute of members or other persons introduced by members.)

Young mining engineer and geologist, member, technical graduate with four years' varied mining experience in the West and Southwest as miner, surveyor and engineer. Past 18 months, work in mine geology, mapping and examinations. Would prefer assisting consulting engineer or work with exploration company in foreign service. Unmarried. Age, 28. No. 331.

Mining engineer and executive. Nine years' experience in development and examinations. At present and for past four years manager of well-known and highly successful mining company. Desire similar position or affiliation with a New York company whose business relates to mining. No. 332.

New York connection with mining or brokerage concern or position as New York representative of outside company wanted by executive mining engineer with broad experience in examination and operation of mining properties. Just completing successful term as resident manager of prosperous mining and reduction works. No. 333.

Member, technical graduate, Mexican, at present holding position as geologist and consulting engineer for one of the largest producing mines in Mexico under English management, desires better position. Twenty-two years' successful experience, three and a half with present company. Excellent references. No. 334.

Member, graduate of the Columbia School of Mines. Drifting and shaft sinking a specialty. 7 years' practical experience in operating and developing work. Can open up new property or take entire charge of mine, skilled in up-to-date methods. At present employed, desire change. Able to handle organization of men. Excellent results. Will go anywhere. Moderate salary. References. No. 329.

Member, technical graduate, age 28, married, at present employed, desires permanent location, preferably with quarrying company in New England States. No. 335.

Member, technical graduate, age 27, 2½ years' experience in smelter and also experience concentrating, cyaniding, and assaying. Desires position on technical staff of smelter or as mill foreman. Can handle men and get results. Interview in Salt Lake. No. 336.

Member, technical graduate, age 37, married 13 years, practical experience in metal mines, 4 years as mine superintendent, desires position in the latter capacity. Have operated mines in United States, Canada and Mexico and am thoroughly experienced in organization and efficiency methods. Available on short notice. Best of references furnished. No. 337.

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LOCAL SECTION NEWS

CHICAGO SECTION

CHAS. H. MACDOWELL, *Chairman*,

LUTHER V. RICE, *Vice-Chairman*,

HENRY W. NICHOLS, *Secretary-Treasurer*,

GEO. P. HULST, ALEX. K. HAMILTON, FREDERICK T. SNYDER, HENRY P. HOWLAND.

The Field Meeting of the Chicago Section proved to be the most enjoyable meeting ever held by the Section. It took the form of a visit to the Northern Indiana Industrial District where the day was spent inspecting metallurgical plants in Indiana Harbor and East Chicago and in amusement at the Hammond Country Club. The weather was perfect and about 50 members and ladies attended the meeting.

An inspection of the plant of the Inland Steel Co. occupied the morning. The plant was decorated in honor of the occasion and the company provided attractive souvenirs. It was noticed here that many of the ladies displayed an interest in the inspection and an understanding of the processes that would not have shamed professional metallurgists. While the members of the Section seemed to be most interested in the by-product coke ovens and the by-product recovery plant, the ladies enjoyed especially watching the snake-like antics of the red-hot rods in the wire mill.

Leaving the steel plant, the Section enjoyed a ride through the district by automobile and arrived at the Hammond Country Club in time for lunch. In the afternoon some of the members inspected the plant of the International Lead Refining Co., at East Chicago, while the rest spent the time at the Country Club in golf, tennis, trap shooting, bridge and other amusements. During the dinner at the Hammond Country Club an informally organized male quartette led the singing in real college boy style. In the evening, E. E. Billow talked on the use of oil burners in metallurgical furnaces, Mr. MacDowell told of the present status of the Potash Industry and the Secretary told of some of the happenings at the Arizona Meeting.

HENRY W. NICHOLS, *Secretary*.

SAN FRANCISCO SECTION

T. A. RICKARD, *Chairman*,

W. H. SHOCKLEY, *Vice-Chairman*,

C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco,
E. A. HERSAM, H. W. YOUNG.

The regular monthly meeting of the Section was held on Oct. 10, 1916, at the University Club of San Francisco.

The report of the special committee appointed to investigate the action of the Directors of the Institute in demanding the withdrawal of

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certain portions of the paper prepared by W. H. Shockley for the International Engineering Congress was submitted, and the resolution submitted by this committee was adopted by unanimous vote and ordered sent on to the President and Board of Directors of the Institute.

The paper of the evening on "Flotative Agents" was read by Prof. G. H. Clevenger of Stanford University. The discussion was participated in by Prof. Hersam, Prof. Lawson, Messrs. Rickard, Shockley, Van Campen and others.

It was announced that the November meeting would be addressed by Prof. Lawson on the subject of "Ore Deposits."

C. E. GRUNSKY, JR., *Secretary.*

ST. LOUIS SECTION

C. J. ADAMI, *Chairman,*

F. W. DEWOLF, HERMAN GARLICHS, M. M. VALERIUS, *Vice-Chairmen,*

W. E. MCCOURT, *Secretary-Treasurer,* Washington Univ., St. Louis, Mo.

A. W. DICKINSON, Herrin, Ill., CHARLES T. ORR, Webb City, Mo.,
C. R. FORBES, Rolla, Mo., F. D. RASH, Earlington, Ky.,
ARTHUR THACHER, St. Louis, Mo.

The St. Louis Section held its usual fall field meeting on October 28. During the day about 30 members visited the properties of the New Staunton Coal Co., the Mount Olive and Staunton Coal Co., the Consolidated Coal Co., and the Superior Coal Co., in the vicinity of Livingston, Staunton, and Gillespie, Ill.

The evening meeting was preceded by a dinner at the Mercantile Club in St. Louis, at which 40 members and guests were present. The Chairman, Charles J. Adami, presided, and the following was the program:

Coal Resources of Southwest Illinois,	F. W. DeWolf.
Some Problems in Coal Operating,	T. T. Brewster.
Coal in the Southwest,	Eugene McAuliffe.
The Use of Electricity in Coal Mining,	S. W. Farnham.
The Institute Meeting,	Philip N. Moore.

AFFILIATED STUDENT SOCIETY NOTES

The October meeting of the **Mining and Geological Society, Lehigh University,** was held in the Eckley B. Coxe Mining Laboratory, and was called to order at 8:15 p. m., by President R. L. McCann.

After the reading of the minutes of the previous meeting, a short business session was held, for the purpose of electing a Vice-President to succeed G. S. Borden, who has left college. Nominees for the office were: H. J. Sloman, P. S. Justice, L. Sargeant.

Dr. F. F. Hintze, of the Geology Department, was elected a mem, ber of the Advisory Committee, which now consists of the President-

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Vice-President, and Secretary, with Dr. Hintze and Mr. H. P. Smith as faculty members. Dr. Hintze succeeds Mr. W. G. Matteson, who has left the Department of Geology to enter oil geology.

Professor Eckfeldt was then introduced and spoke on "The Arizona Meeting of the A. I. M. E." His talk was very interesting and instructive. He outlined the entire trip, relating many amusing and entertaining experiences. Professor Eckfeldt showed a number of slides, post cards, and photographs, and at the conclusion of his talk was extended a rising vote of thanks.

President McCann then urged all entering men to join the Society, and outlined the benefits to be derived from so doing.

The meeting then adjourned to Williams Hall to partake of "eats" and "smokes." Attendance about 45.

ALLAN E. DYNAN, *Secretary.*

The Mining Engineering and Geological Society of the State College of Washington has commenced its fall work with a vim. The meetings have been largely attended and two good papers have been presented by students.

We are hoping to have all prominent mining and metallurgical men who are out in this section talk to the society and would appreciate any information as to who is likely to be out here, so that we can extend an invitation to him to visit us.

The officers elected for this year are:

President, James G. Parmalee,
Vice-President, C. C. Camp,
Secretary-Treasurer, Aubrey C. Miller, and
Sergeant at Arms, Robert Moss.

Last year's activities were meetings every two weeks at which students, professors, and prominent mining men presented papers or gave informal talks; and initiations in the fall and spring. In the fall we meet the Civils in a game of football, and so far have been rather victorious as to the number of games won.

AUBREY C. MILLER, *Secretary.*

The Mining Society of the University of Washington has held its annual election of officers and has passed an amendment to its constitution.

The amendment is, that all members of the College of Mines shall be eligible to the Mining Society. The effect of this amendment has been to increase our membership to about 60 men and to interest the underclassmen in the activities of the Society and thus bring the College of Mines and its work before a much larger number of students.

We are holding regular meetings every two weeks. At the first meeting the following officers for the year 1916–17 were elected:

President, Jess Johnson,
Vice-President, E. Lee Tucker,
Secretary, Richard Luther,
Treasurer, Walter F. Brown,
Corresponding Secy., Henry G. Boulton.

RICHARD LUTHER, *Secretary.*

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At the first regular meeting of the **Missouri Mining Association of the Missouri School of Mines,** Mr. H. A. Buehler, State Geologist of Missouri, gave a very interesting and instructive talk on the Southwestern meeting of the A. I. M. E. We hope to have Mr. Buehler on our program again this year.

After Mr. Buehler's talk, the following officers of the Association were elected:

President, Martin F. Bowles,
Vice-President, R. O. Shriver,
Secretary, H. T. Herivel,
Treasurer, Robert Lyons.

H. T. HERIVEL, *Secretary.*

At the recent election of the **Mining Society of the School of Mines of the University of Pittsburgh** the following officers were elected:

President, J. S. Grumbling,
Vice-President, C. E. Thornhill,
Secretary and Treasurer, C. L. Armstrong,

The meetings of the Society are held the second and fourth Thursdays of each month.

C. L. ARMSTRONG, *Secretary.*

The **Geological and Mining Society** of **Stanford University** now has 23 active members. Meetings are held once a month with meetings in between when talks are given by visitors on especially interesting subjects.

The Society has made plans to publish the third volume of the Year Book of the Society and it will be out before the new year. It will contain a directory giving the name of every man who has graduated from or is now in the Department of Geology and Mining with professional record.

The officers for 1916–17 are:

President, V. L. King,
Secretary-Treasurer, Z. K. Melcon.

Z. K. MELCON, *Sec'y-Treas.*

The **Students' Organization of the Michigan College of Mines** has elected the following officers for the year 1916–17:

President, G. L. Adams,
1st Vice-President, D. E. Dunn,
2d Vice-President, E. L. Bemis,
3d Vice-President, M. W. Franey,
Secretary, J. W. Alt,
Treasurer, E. S. Listar.

We will arrange a program as soon as possible, and endeavor to have some of the copper country's prominent mining men address the student body.

GALE F. ADAMS, *Secretary for* 1915–16.

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LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

AMERICAN PETROLEUM INDUSTRY. Vols. 1-2. By R. F. Bacon and W. A. Hamor. New York, 1916.

COKE INDUSTRY OF NEW SOUTH WALES. (New South Wales Geological Survey. Mineral Resource No. 23.) Sydney, 1916.

HANDBOOK OF ROCK EXCAVATION, METHODS AND COST. By Halbert Powers Gillette. Illustrated. Clark Book Co., New York, 1916. Price $5 net. (Gift of Publishers.)

[NOTE.—The purpose of this book is to place in the hands of those interested in the economic handling of rock, in the most convenient form, the bulk of the material published 12 years ago under the title of "Rock Excavation, Methods and Cost," together with much more material since gathered by the author. Two companion volumes on "Earth Excavation" and "Tunnels and Shafts" are in the course of preparation by the same author. The large number of methods of excavating and transporting rock under different conditions and the tools and machines required for this work are fully described and illustrated. For each class of rock excavation, and for each type of tool or machine used, examples of actual costs are given. Where feasible, these are reduced to unit form, making possible a comparison of machines or methods and aiding in the estimation of the cost of rock work. The handbook size with flexible binding will commend itself to those who use the book. The scope

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of the book is further indicated by the chapter headings: Rocks and Their Properties; Methods and Cost of Hand Drilling; Drill Bits; Shape, Sharpening and Tempering; Machine Drills and Their Use; Cost of Machine Drilling; Steam, Compressed Air and other Power Plants; Cable Drills, Weld Drills, Augers and Cost Data; Core Drills; Explosives; Charging and Firing; Methods of Blasting; Loading and Transporting Rock; Quarrying Dimension Stone; Open Cut Excavation in Rubble Quarries, Pits and Mines, Railroad Rock Excavation and Boulder Blasting; Canal Excavation; French Works; Subaqueous Rock Excavations.—B. A. R.]

INTERNATIONAL ENGINEERING CONGRESS, 1915. Vol. VIII—Mining Engineering. San Francisco, 1915.

METHOD FOR MEASURING THE VISCOSITY OF BLAST FURNACE SLAG AT HIGH TEMPERATURES. (Tech. paper No. 157, U. S. Bureau of Mines.) Washington, 1916.

MINING OPERATIONS IN THE PROVINCE OF QUEBEC, 1915. Report on. Quebec, 1916.

THE ANALYSES OF COPPER AND ITS ORES AND ALLOYS. By George L. Heath. McGraw-Hill Book Co., New York, 1916. Price $3. (Gift of Publishers.)

[NOTE.—This volume contains an account of the principal methods employed by the largest refineries, foundries and custom sampling works for the control of operations and valuation of material. The chapters are divided into five groups following the logical sequence from the ore in the mine to the finished metallic product. Part 1 includes a description of sampling and crushing methods; reagents and standard solutions; and the analyses of ores, slags, matte, flue dust and furnace by-products for copper, lead, silver, gold and platinum. Part 2 covers the work of electrolytic refining. Part 3 deals with the electrolytic assay of refined copper and the determination by special methods of foreign metals and elements in copper. Part 4 is devoted to the methods of analysis of the principal commercial alloys of copper. In part 5 are treated the metallography of copper and brass and the electrical resistivity of copper. The technical chemist will find this book a valuable addition to his library; a volume which will reward his search for authoritative information in the field covered.—B. A. R.]

UNDERGROUND WASTES IN OIL AND GAS FIELDS AND METHODS OF PREVENTION. (Tech. paper No. 130, U. S. Bureau of Mines.) Washington, 1916.

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DIE GESTEINSBILDENDEN MINERALIEN. By Ernst Weinschenk. Freiburg, 1901.

GEOLOGY AND MINERAL RESOURCES OF THE YILGARN GOLDFIELD. Bull. No. 63, Western Australia. Geological Survey. Perth, 1915.

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JAPAN IMPERIAL GEOLOGICAL SURVEY. Geological Map. Division V. Tokyo, 1916.

JAPAN IMPERIAL MINERAL MAP, DIVISION V. Tokyo, 1916.

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STATISTICS OF THE MINERAL PRODUCTION OF ALABAMA FOR 1913. Bull. No. 15, Alabama. Geological Survey. University, 1914.

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RESCUE AND RECOVERY OPERATIONS IN MINES AFTER FIRES AND EXPLOSIONS. By J. W. Paul and H. M. Wolflin. Washington, 1916.

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Company Reports

BROKEN HILL PROPRIETARY COMPANY LIMITED. Report of 62d half-yearly ordinary general meeting, Aug. 1916. (Gift of Company.)

GIRARD ESTATE. Report of Mining Engineer and Agent, 1915. (Gift of James Archbald.)

General

APPLIED ELECTRICITY FOR PRACTICAL MEN. By A. J. Rowland. New York, 1916.

CONCISE TECHNICAL PHYSICS. New York, 1916.

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Gift of F. K. Gillespie

99 UNITED STATES TOPOGRAPHIC MAPS OF NEW YORK, NEW JERSEY AND PENNSYLVANIA.

Trade Catalogs

AMERICAN BLOWER COMPANY. Detroit, Mich. Sirocco Service.

BUFFALO FOUNDRY AND MACHINE COMPANY. Buffalo, N. Y. Chemical Apparatus. "Buflokast" and "Buflovak."

CHICAGO PNEUMATIC TOOL CO. Chicago-New York.
Bull. 34-K. Fuel oil driven compressors.
Bull. 34-2. Single compressors steam driven.
Bull. E-44. Duntley Electric sensitive drilling stand. 1916.

CROCKER NATIONAL FIRE PREVENTION ENGINEERING COMPANY. New York, N. Y. Prevention of Fire.

IVANHOE-REGENT WORKS OF THE GENERAL ELECTRIC COMPANY. Cleveland, O. The Hunchman. September, 1916.

KOEHRING MACHINE COMPANY. Milwaukee, Wis. Koehring Mixer. Aug.-Sept., 1916.

NITROGEN PRODUCTS COMPANY. Providence, R. I. Fixation of Atmospheric Nitrogen. 1916.

STROUD, E. H. & Co. Chicago, Ill.
Air separation pulverizer.
Air separation coal pulverizer.
Screen separation crushing, shredding, disintegrating and pulverizing mill.

TEXAS COMPANY. New York, N. Y. Lubrication. October, 1916.

UNDER-FEED STOKER COMPANY OF AMERICA. Chicago, Ill. Publicity Magazine. September-October, 1916.

UNITED SHOE MACHINERY ATHLETIC ASSOCIATION. Beverly, Mass. The Three Partners. September, 1916.

WESTINGHOUSE ELECTRIC & MANUFACTURING COMPANY. East Pittsburgh, Pa. Catalog 3-B. Westinghouse Instruments. July, 1916.

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MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Oct. 10, 1916 to Nov. 10, 1916.

ANCHOR, HANS C., Supt., Dome Extension Mines Co., Ltd., South Porcupine, Ont., Canada.
BABCOCK, EARLE J., Dean, College of Min. Engrg., State Univ. of N. Dak., Grand Forks, N. Dak.
BIDDISON, P. MCDONALD, Construction Engr., The Ohio Fuel Supply Co., 52 W. Gay St., Columbus, O.
BOISE, PAUL T., Asst. Mgr., Stimpson Equipment Co., Felt Bldg., Salt Lake City, Utah.
BOURNE, RALPH H., Vice-Pres..... Whiting Foundry Equipment Co., Harvey, Ill.
BROWN, EDWARD OTTO FORSTER, Min. Engr., 706–707 Salisbury House, London, E. C., England.
CALHOUN, ALLAN B., Min. Engr., Old Dominion Copper Min. & Smelt. Co., Box 535, Globe, Ariz.
CHAMBERS, ALLISON ROBERT, Asst. Mgr., Ore Mines & Quarries, Nova Scotia Steel & Coal Co., Ltd., P. O. Box 213, New Glasgow, Nova Scotia.
CHEN, TING CHI, Met. Engr., Blast Furnace Dept., Han-Yeh Ping Iron and Coal Co., Hanyang Iron and Steel Works, Hanyang, China.
CHU, HSING CHUNG, Min. Engr., Min. Dept., Provincial Financial Bureau, Tsinaufu, Shantung, China.
DALLY, ISAAC N., Min. Engr.....752 Newton St., Seattle, Wash.
DECAMP, W. VOL., Supt., Blue Bell Mine, Cons. Arizona Smelt. Co., Mayer, Ariz.
DIMMICK, SHELBY D., Min. Engr., Delaware, Lackawanna and Western R. R. Co., 1646 Capouse Ave., Scranton, Pa.
FERRAZ, JORGE BELMIRO DE ARAUJO, Asst. Geologist and Petrographist, Bureau of Geology and Mineralogy, Dept. of Agriculture, Rio de Janeiro, Brazil, S. Amer.
HARDIE, AUGUSTUS B., Const. Supt., The Process Engineering Co., 1209 Stephen Girard Bldg., Philadelphia, Pa.
HELMRICH, GEORGE L.....Braden Copper Co., Rancagua, Chile, South America.
HUBBARD, HARMON B., Supt., Open Hearths, Inland Steel Co., Indiana Harbor, Ind.
HURUM, FREDRIK J. O., Met. Engr.....Carbon Steel Co., Pittsburgh, Pa.
INOUYE, JUNZO, Met. Engr., Mitsubishi Iron Works, Kenjiho, Koshu, Kokaido, Korea.
KING, EDWARD CHARLES, Smelter Supt., Cons. Ariz. Smelt. Co., Humboldt, Ariz.
KINNEAR, JOHN CHARLES, Asst. Smelter Supt., Nevada Cons. Copper Co., McGill, Nev.
LUNA, GILBERTO, Cia. Minera Angustias, Dolores y Anexas, South America, Pozos, Gto., Mexico.
MCCOY, ALEXANDER WATTS, Geol.....Marland Oil Co., Ponca City, Okla.
MCGRATH, RAY P., District Mgr., Sullivan Machinery Co., 461 Market St., San Francisco, Cal.
MCLAUGHLIN, DONALD HAMILTON, Geol., Secondary Enrichment Investigation, 40 Conant Hall, Cambridge, Mass.
MARCY, FRANK E., Cons. Engr.....438 Atlas Bldg., Salt Lake City, Utah.
MARTIN, JACOB A., Mine Engr., Kennecott Copper Corpn., Latouche, Latouche Island, Alaska.
MEE, WHITNEY PLAYER, Chemist, Chino Copper Co., Box 361, Santa Rita, New Mexico.
MILES, JOHN HENRY, Genl. Supt., Alta Bert Gold Dredging Co., Trinity Center, Cal.
NEWHOUSE, EDGAR L., JR., Mgr., Garfield Chemical and Mfg. Corpn., Salt Lake City, Utah.
OSBORN, CHASE S., Explorer and Prospector.....Sault Ste. Marie, Mich.
PALMER, GUALTERIO E.....50 de Mayo No. 32—Room 1, Mexico City, Mex.
PATTISON, EVERETT H., Min. Engr.....626 Hutton Bldg., Spokane, Wash.
PEABODY, FRANCIS S., Prest., Peabody Coal Co., 332 S. Michigan Ave., Chicago, Ill.
PHELPS, WILLIAM BEMISS, Supt. Yago Alaska Gold M. & M. Co., Knik, Alaska.
QUIGLEY, WILLIAM J., Mgr., El Potosi Mining Co., 1206 Mills Bldg., El Paso, Tex.
RANSOM, RASTUS S., JR.....Box 293, Hartsdale, N. Y.
SMITH, ELLWOOD SPENCER, Asst. to the Genl. Mgr., Cons. Arizona Smelt. Co., Humboldt, Ariz.

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SMITH, JESSE D.,.....Mine Supt., Mineral Point Zinc Co., Galena, Ill.
STARAL, FRANK J., JR., Mgr., The Investors Co., Sheridan, Madison Co., Mont.
STROHBACH, WILLIAM CARL, Chemist, Ray Cons. Copper Co., Box 624, Hayden, Ariz.
WARREN, FLOYD ALPINE, SR., Supt., Blast Furnaces, Dwight-Lloyd Machine and Hearth Depts., St. Louis Smelt. & Ref. Co., Collinsville, Ill.
WERTHAN, SIDNEY E., Chem., Copper Queen Cons. Mining Co., P. O. Box 112, Bisbee, Ariz.
WEST, GEORGE HAZZARD, Smelter Foreman, Old Dominion Copper Min. & Smelt. Co., Box 2007, Globe, Ariz.
WESTCOTT, HENRY P., Engr., Natural Gas Construction Work, Metric Metal Works, Erie, Pa.
WILKIN, GARRATT S., Mining, Metal Mgr., Moscow Min. & Mill. Co., Moscow, Beaver Co., Utah.
WILLIS, CHARLES FRANCIS, Director, Arizona State Bureau of Mines, Univ. of Arizona, Tucson, Ariz.
WOODWARD, FRANK A.,.....Supt., Iron Cap Copper Co., Copper Hill, Ariz.
YOUNG, STUART M., Shift Boss, Min. Dept., U. S. Smelt. Co., Bingham Canyon, Utah.

Associate Members

ALLEN, HEMAN H., Sales Engr., Sullivan Machinery Co.; 704 Kearns Bldg., Salt Lake City, Utah.
RANDALL, Henry D.,..... Mgr., General Electric Co., Salt Lake City, Utah.
STEVENS, PAUL, Private Secretary to Genl. Mgr., Calumet & Arizona Mining Co., P. O. Box. 824, Warren, Ariz.

Junior Members

SANGER, ALAN BRIDGMAN, Student.....Mass. Inst. of Technology, Boston, Mass.
SWEET, ANDREW THOMAS, Instructor, Dept. of Metallurgy and Ore Dressing, Michigan College of Mines, Houghton, Mich.
WEBB, TORREY H.....616 W. 113th St., New York, N. Y.

Total Membership, Nov. 10, 1916.....5,848

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period Oct. 10, 1916, to Nov. 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Arthur P. Anderson, Miami, Ariz.
Proposed by C. E. Arnold, George R. Lehman, W. F. Palmer.

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Born 1890, Minneapolis, Minn. 1905–09, North Side High School, Minneapolis, Minn. 1909–14, Univ. of Minnesota, E. M. 1914–15, Construction Work, Great Northern R. R., Glacier National Park, Mont. 1915–16, Rodman, Raise Engr., Shift Boss, Inspiration Cons. Copper Co., Miami, Ariz.
Present position: Shift Boss, Inspiration Cons. Copper Co.

Lawrence Wright Anderson, Bingham, Utah.
Proposed by R. C. Gemmell, H. C. Goodrich, Ernest Gayford.
Born 1884, Salt Lake City, Utah. 1904–05, General engineering course, Univ. of Mich., Ann Arbor. 1905–06, General engineering course, Univ. of Columbia, New York City. 1906–07, Nevada Goldfield Reduction Co., Goldfield, Nev. 1907–09, Utah Copper Co., Magna concentrating mill, Garfield, Utah. 1909–11, Dragon Iron Min. Co., Silver City, Utah.
Present position: 1911 to date, Level foreman, Utah Copper Co.

Thomas B. H. Askin, Miami, Ariz.
Proposed by Rudolf Gahl, R. S. Allen, Guy H. Ruggles.
Born 1890, Newark, N. J. 1908–10, Sheffield Scientific School, Yale Univ. 1910–13, Michigan College of Mines, E. M., B. S. 1913–15, Statistician, Commonwealth Edison Co., Chicago, Ill.
Present position: 1915 to date, Ball Mill Expert, Inspiration Cons. Copper Co.

Clarence William Barnard, Globe, Ariz.
Proposed by H. H. Colley, W. B. Cramer, C. T. Emrich.
Born 1886, Maple Rapids, Mich. 1906, Grad., Durand High School, Durand, Mich. 1909, Grad., Michigan College of Mines, Houghton, Mich., B. Sc. and E. M. 1909, Employed as care taker, El Dorado Gold Dredge, Diamond City, Mont. 1909, Leaser on Gold Hill, Diamond City, Mont. 1909, Asst. Min. Engr., Penn Iron Min. Co., Vulcan, Mich. 1911, Furnace-man, Canadian Copper Co., Copper Cliff, Ont. 1911, Min. Engr., Asbestos and Asbestic Co., Asbestos, Prov. of Quebec.
Present position: Treas. and Resident Mgr., Arizona Asbestos Assn.

Charles Bentley, Rapid City, So. Dak.
Proposed by Welton J. Crook, A. S. Halley, Louis D. Huntoon.
Born 1877, Ontario, Canada. Prior to 1896, Graded School, Grindstone City, Mich. 1896–1909, Home study. 1909–11, So. Dakota State School of Mines as special student. 1896–1903, Sampler, Deadwood and Delaware Smelter, Deadwood, S. D. 1903–06, Asst. Assayer, Horseshoe Min. Co., Terry, S. D. 1906–09, Assayer and later Supt. of Cyanide Plant, Bonanza Mine, in Pis Pis District, Nicarauga, C. A. 1911–16, Head Assayer, Trojan Min. Co., Trojan, S. D.
Present position: In charge of Bureau of Mines, So. Dakota State School of Mines.

William C. Bochert, Rapid City, So. Dak.
Proposed by Welton J. Crook, Cleophos C. O'Harra, Louis D. Huntoon.
Born 1887, Grimma, Germany. Up to 1904, Public Schools, Dubuque, Iowa. 1909–10, Preparatory, So. Dakota School of Mines. 1910–14, College course, So. Dakota School of Mines, B. S. 1909–14, Underground work, Homestake Min. Co., Lead, So. Dak. 1914–16, Assayer and Millman, Trojan Min. Co., Trojan, So. Dak.
Present position: Prof. of Mining, So. Dakota School of Mines.

Floyd R. Brooks, Worthington, Ont., Canada.
Proposed by C. C. O'Harra, Welton J. Crook, Amil A. Anderson.
Born 1888, Rapid City, So. Dak. 1911, Grad., So. Dakota School of Mines, Rapid City, So. Dak. 1911, Engr., Inspiration Copper Co. 1911–13, Solution man, Trojan Min. Co., Trojan, So. Dak. 1913, Miner and Foreman, Inspiration Cons. Copper Co. 1913–14, Surface Boss, Mond Nickel Co., Ltd. 1914 to date: Shift Boss, underground, Mond Nickel Co., Ltd.
Present position: Mine Foreman, Mond Nickel Co., Ltd.

John Albert Burgess, Oatman, Ariz.
Proposed by Philip Wiseman, S. W. Mudd, Frank A. Keith.
Born 1876, St. John, N. B., Canada. 1902–06, Univ. of Cal., College of Mining, B. S. 1906–11, Chief Engr., and Geol., Tonopah Mining Co. of Nevada, Tonopah, Nev. 1911–16, Supt., Nevada Wonder Min. Co., Wonder, Nev.
Present position: Genl. Supt., United Eastern Min. Co.

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Arthur William Burgren, El Paso, Tex.
Proposed by W. M. Drury, W. J. Deavitt, R. F. Manahan.
Born 1880, San Francisco, Cal. 1886–96, Primary and Grammar Schools, San Francisco. 1904–05, Preparatory School, San Francisco. 1905–10, Stanford Univ., A. B. Degree Geology and Mining. 1896–02, Machinist with W. T. Garratt & Co., American Can Co., and Union Iron Works, San Francisco. 1902, Timberman with Oneida Gold Mining & Milling Co., Sutter Creek, Cal. 1903–04, Junior Engr., U. S. Army Transport "Thomas," Philippine Service. 1904–05, Student, Preparatory School, San Francisco. 1905–10, Student, Stanford University, Cal. 1910–11, Miner, Sampler, and Asst. Engr., with Montana, Tonopah and MacNamara Mining Co., Tonopah, Nev. 1911–12, Asst. to the Mgr., Santa Maria (Mexico) Mines Ltd., at Minillas, Zac., Mexico. 1912–14, Chief Engr., Dolores Mines, American Smelters Securities Co., Matehuala, S. L. P., Mexico. 1914–15, Millman, Flotation Mill of the Engel Copper Mining Co., Keddie, Cal. 1915, June and July examination work at Valdez, Alaska. 1915, To December 31st, Engr., Cinco Mines Co., Jalisco, Mexico.
Present position: January, 1916, to date, Chief Engr., Dolores Mines of the American Smelters Securities Co.

Albert Cameron Burrage, Boston, Mass.
Proposed by H. L. Smyth, Charles H. White, George S. Raymer.
Born 1859, Ashburton, Mass. 1883, Harvard College, A. B. 1899, Organizer and Director of Amalgamated Copper Co., and Santa Rita Copper Mining Co. 1900 to date, Associated as Director or President with a number of different mining and metallurgical companies. By profession a lawyer; since 1899, however, have given all my time to mining and metallurgical enterprises in various parts of the world.
Present position: Vice-President and Director of Chile Copper Co.

John Bruce Carlock, Los Angeles, Cal.
Proposed by Henry S. Drinker, Benjamin L. Miller, Howard Eckfeldt.
Born 1882, Carlock, Ill. 1896–97, Illinois State Normal Normal, Ill. 1897–99, Missouri Normal, Kirksville, Mo. 1899–1900, Elgin Academy, Elgin, Ill. 1900–01, Chicago Univ., Chicago, Ill. 1903–07, Lehigh Univ., So. Bethlehem, Pa., E. M. 1901–03, American Can Co., New York, Chicago, Kansas City. 1907–09, Blast Furnace Dept., Bethlehem Steel Co., So. Bethlehem, Pa. 1909, Edison Cement Co., New Village, N. J. 1910, Prospecting in Lower California. 1910–11, Supt., Eureka Min. Co., Magalia, Cal. 1912–13, Div. Engr., General Petroleum Co., Taft, Cal. 1913–14, Supt. of Refineries, General Petroleum Co., Los Angeles, Cal. 1914–15, Mgr., Pacific States Refineries, Oakland, Cal. 1915–16, Div. Mgr., Richfield Oil Co., San Francisco.
Present position: Mgr. of Refineries, Richfield Oil Co.

James Joseph Carrigan, Butte, Mont.
Proposed by C. W. Goodale, C. L. Berrien, J. Gillie.
Born 1886, Hancock, Mich. 1901–03, Attended St. Patrick's High School, Hancock, Mich. 1903–05, Attended Hancock High School, Hancock, Mich. 1905–08, Attended Michigan College of Mines, Houghton, Mich. 1908–09, Worked as miner in the St. Lawrence Mine of the Anaconda Copper Co. 1909–12, Worked in Geological Dept., of Anaconda Copper Co. 1912–15, Shift-boss in Badger State Mine of Anaconda Copper Co. 1915–16, Safety Engineer, Anaconda Copper Co.
Present position: Asst. Genl. Supt. of Mines, Anaconda Copper Co.

Perry L. Charles, Anaconda, Mont.
Proposed by Louis V. Bender, Harry S. Ware, Ralph Hayden.
Born 1892, Hiawiatha, Nebr. 1907–11, Broadway High School, Seattle, Wash. 1912–16, Univ. of Washington, Seattle, Wash., Chemical Engineering. 1911–12. United States Engineering Dept., River and Harbor Survey.
Present position: Chemist, Research Dept., Washoe Reduction Works, Anaconda Copper Min. Co.

Thomas Cowperthwaite, Warren, Ariz.
Proposed by L. D. Ricketts, John C. Greenway, Ira B. Joralemon.
Born 1876, Northumberland, England. 1905, Missouri School of Mines, B. Sc. 1907, Engrg. Dept., Calumet and Arizona Min. Co.
Present position: 1915 to date, Safety Inspector, Calumet and Arizona Min. Co. and allied companies.

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Earl B. Crane, Black Butte, Ore.
Proposed by W. B. Dennis, Robert M. Betts, Albert Burch.
Born 1878, Ottawa, Kans. 1896–98, Univ. of Idaho. 1898–1900, Mass. Inst. of Tech. 1896, Assayer, Josie Gold Min. Co., Rossland, B. C. 1901, Miner, Marrison Mine B. C. Copper Co., Greenwood, B. C. 1904, Office Man, Atlas Gold Min. Co., Concord, Idaho. 1906, Supt., Trethewey Silver Cobalt Min. Co., Cobalt, Ont. 1914, Engr., Cornucopia Mines Co., Cornucopia, Ore.
Present position: Supt., Black Butte Quicksilver Mine.

Charles Williamson Cromwell, Warren, Ariz.
Proposed by L. D. Ricketts, A. G. McGregor, Ira B. Joralemon.
Born 1881, Fort Smith, Ark. 1905, Univ. of Arkansas, B. C. 1905–06, Draftsman, Virginia Bridge & Iron Co., Roanoke, Va. 1906–07, Draftsman, McClintic-Marshall Const. Co., New York City. 1907–08, Draftsman, Southwestern Bridge Co., Joplin, Mo. 1908–12 Draftsman, Boston & Montana Cons. Copper & Silver Mining Co, Great Falls, Mont.
Present position: 1912 to date, Civil & Structural Engr., on metallurgical plants, Warren, Ariz.

George Corliss Crossley, Toms River, N. J.
Proposed by S. T. Nicholson, H. C. Boynton, C. G. Roebling.
Born 1881, Trenton, N. J. Attended State School, Trenton, N. J., and Drexel Institute, Philadelphia, Pa. After education, three years apprentice in machine shops, blacksmith shop, foundry and pattern shop of my father at Trenton, N. J. Two years outside erecting and prospecting. Since then operating own mines at Batchellerville, N. Y., and Cranberry Creek, N. Y. (feldspar and quartz); Saylorsburg, Pa., (kaolin); Wells Mills, N. Y. (fire clay); Crossley, Ocean Co., N. J. (high grade clays and silica sand). Did consulting work and prospecting occasionally since graduation from technical school. (Specialty, mining and milling of feldspar, quartz, kaolins and clays.)
Present position: Owner and operator, The Crossley Mines.

Michael Curley, Ajo, Ariz.
Proposed by J. Owen Ambler, John C. Greenway, L. D. Ricketts.
Born 1874, Ireland. High School education. Prior to date below in railroad work, Ticket Agent, Cashier. 1902, Accounting Dept., Oliver Iron Min. Co., Duluth, Minn. 1905, Chief Clerk, Camsteo District. 1908, Supt., Camsteo Mine, Coleraine Mine. 1910, Supt., Hill Mine, Marble, Minn. 1912, Supt., New Cornelia Copper Co., Ajo, Ariz.
Present position: Genl. Supt., New Cornelia Copper Co.

Joseph E. Curry, Warren, Ariz.
Proposed by Ira B. Joralemon, E. E. Whiteley, L. D. Ricketts.
Born 1870, Visalia, Cal. Twelve years as Accountant and Chief Clerk, Copper Queen Cons. Min. Co. Fourteen years as Accountant and Chief Clerk, Calumet and Arizona Min. and Superior and Pittsburgh Copper Co.
Present position: Chief Clerk, Calumet and Arizona Mining Co.

Lionel Hennan Duschak, Berkeley, Cal.
Proposed by F. G. Cottrell, Frank H. Probert, Benjamin B. Lawrence.
Born 1882, Buffalo, N. Y. 1899, Graduated Buffalo Central High School. 1904, University of Michigan, A. B. 1908, Princeton Univ., A. M., Ph. D. in Chemistry. 1905–09, Asst. and Instructor in Chemistry, Princeton University. 1909–13, Phys. Chemist, Corning Glass Works, Corning, N. Y.
Present position: 1913 to date, Chemical Engineer Metallurgical Div., U. S. Bureau of Mines, Berkeley, Cal.

Charles W. Eichrodt, Miami, Ariz.
Proposed by C. E. Arnold, George R. Lehman, W. F. Palmer.
Born 1889, Indianapolis, Ind. 1903–07, Manual Training High School, Indianapolis. 1907–09, Univ. of Michigan. 1909–11, Univ. of Indiana, A. B. 1911–14, Columbia Univ., E. M. 1914–15, Detroit Copper Co., Morenci, Ariz. 1915, Copper Queen Cons. Min. Co.
Present position: 1915 to date, Shift Boss, Inspiration Cons. Copper Co.

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Carl Sigmund Elayer, Copper Hill, Ariz.
Proposed by Robert R. Boyd, Walter Harris, W. B. Cramer.
Born 1890, Salem, Mo. 1908, Grad., East St. Louis High School, East St. Louis, Ill. 1911–12, Freshman year, Missouri School of Mines, Rolla, Mo. 1913, Sample Dept., Copper Queen Cons. Min. Co., Douglas, Ariz. 1913–15, Miner, Globe District, Globe, Ariz. 1915, Engr., Arizona Commercial Min. Co.
Present position: Mine Foreman, Arizona Commercial Min. Co.

T. Evans, Cananea, Son., Mex.
Proposed by L. D. Ricketts, George Kingdon, John C. Greenway.
Born 1871, Aberdeen, Mich. High School. 1889, Southern Ry., Memphis. 1892, Illinois Central R. R., Chicago. 1901, The Cananea Cons. Copper Co. 1910, Mgr., The Mine and Smelter Supply Co., Denver, Colo.
Present position: 1915 to date, Traffic Mgr. and Purchasing Agent, The Cananea Cons. Copper Co.

Charles Hurin Fay, Hanover, N. Mex.
Proposed by Arthur Thacher, Karl A. Strand, Cony T. Brown.
Born 1889, Wyoming, O. 1895–1903, Wyoming Public School, Wyoming, O. 1903–07, Wyoming High School, Wyoming, O. 1908–13, Colorado School of Mines, Golden, Colo., E. M. 1907–08, Clerk, The Philip Corey Co., Cincinnati, O. 1912, Chemist, San Luis Valley Sugar Co. 1913, Machine man, The Colorado Yule Marble Co., Marble, Colo. 1913, Mucker, The Smuggler Min. Co., Aspen, Co.
Present position: 1914 to date, Engr., Hanover Mines, The Empire Zinc Co.

Herbert Walter Felton, Nacozari, Son., Mex.
Proposed by J. S. Williams, H. T. Hamilton, C. I. Schultz.
Born 1884, Mazatlan, Sinaloa, Mex. 1898–1901, Grad., Hamilton Grammar School, San Francisco, Cal. 1901–04, Grad., Lowell High School, San Francisco, Cal. 1904–08, Grad., College of Mines, Univ. of Cal., Berkeley, Cal. 1908, Mucker, Tonopah Min. Co., Tonopah, Nev. Making Mine Model, West End Min. Co., Tonopah, Nev. 1909, Mine Sampler, Machine man, Machine Boss, Night Foreman, Butters Copala Mines Co., Copala, Sin., Mex. 1910–12, Shift Boss, Minas del Tajo Min. Co., Rosario, Sin., Mex. 1913, Mine Foreman, Mexican Mines Co., Bolanos, Jal., Mex. 1913 to date, Level Boss, Night Foreman, Division Foreman, Moctezuma Copper Co., Pilares de Nacozari, Son., Mex.
Present position: Division Engr. Moctezuma Copper Co.

Howard Hawthorne Fields, Alta, Utah.
Proposed by W. Fitch, R. C. Gemmell, H. C. Goodrich.
Born 1887, Minneapolis, Minn. 1906–07, Univ. of Minnesota, Civ. Engrg. 1910–11, Univ. of Minnesota, Min. Engrg. 1911–13, Michigan College of Mines. 1906, Engr., Washburn Coal Co., Williston, N. D. 1908–10, Engr. and Chief Engr., Coal Min. Dept., Cerro de Pasco Min. Co., Cerro de Pasco, Peru, So. Amer. 1911–12, summer vacations, Sampler and Asst. Engr., Utah Cons. Min. Co., Bingham, Utah. 1913, Engr., Utah Cons. Min. Co., Bingham, Utah. 1913–15, Asst. Field Engr. and Field Engr., General Dev. Co., Alaska. 1915–16, Genl. Mgr., Triangle Min. Co., Alta, Utah (Leasing).
Present position: Genl. Mgr., Triangle Mining Co.

Frederick H. Finnell, Iola, Kans.
Proposed by H. G. Hixon, J. P. Varian, G. M. Garlock.
Born 1879, Englevale, Kans. 1897, High School, Pittsburg, Kans. 1901–04, Chemistry, International Correspondence Schools, Scranton, Pa. 1914, Drawing, International Correspondence School, Scranton, Pa. Metallurgy, Mechanics and other studies at home. 1898–1902, Asst. in laboratory, Cherokee Lanyon Spelter Co., Pittsburg and Gas, Kans. 1903–06, Chemist, Prime Western Spelter Co., Gas and Iola, Kans. 1907, Chemist and Asst. Ore buyer, Morlorn and Schloss, Monterey, Mexico. 1908–12, Chemist, Prime Western Spelter Co., Gas and Iola, Kans.
Present position: Supt., Prime Western Spelter Co., Plant No. 1, Gas, Kans.

Duncan Davidson Forbes, Clifton, Ariz.
Proposed by Norman Carmichael, M. R. Hull, Arthur Crowfoot.
Born 1891, London, England. 1907–11, I had a high school education in London, England, followed by an engineering course at the University of London from which I graduated with an Honour B. Sc. degree in mechanical and civil engineering. 1907–

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12, Before entering the University I became a pupil of Sir A. F. Yarrow and had a three-year training in mechanical engineering in the shops and shipyard in Messrs. Yarrow & Co., Ltd., which was completed in 1912. 1912–14, Draftsman with Messrs. Yarrow & Co., Ltd., Glasgow, Scotland. 1914–16, Draftsman and Asst. Estimator with the Anaconda Copper Mining Co., Anaconda, Mont.

Present position: Estimator in the Engineering Dept., of the Arizona Copper Co., Clifton, Ariz.

Carroll A. Garner, Hazleton, Pa.

Proposed by R. V. Norris, H. W. Monts, Paul Sterling.

Born 1889, Ashland. 1910, Pennsylvania State College—School of Mines—B. S. in Mining. 1910–11, Miami Copper Co., Engrg. Dept., Miami, Ariz. 1911–12, Inspiration Cons. Copper Co., Miami, Ariz. 1912–13, Instructor in Mining, Pennsylvania State College.

Present position: 1913 to date, Div. Engr., Lehigh Valley Coal Co.

Herbert Spencer Gieser, Anaconda, Mont.

Proposed by Louis V. Bender, Harry S. Ware, Bayard S. Morrow.

Born 1884, Kansas City, Mo. 1906, Montana School of Mines, E. M. 1907, Asst. Surveyor, North Butte Min. Co., Butte, Mont. 1908, Cyanider, El Oro Min. and Ry. Co., Ltd., El Oro, Mex. 1909, Concentrator, Butte Reduction Works, Butte, Mont. 1909, Cyanider, Easton Pacific Mining Co., Virginia City, Mont. 1910, Laboratory, charge Sampling Mill, Cerro de Pasco Min. Co., La Fundicion, Peru. 1911, Foreman Concentrator, Braden Copper Co., Rancagua, Chile. 1911, Battery City Deep, Ltd., Johannesburg, Transvaal. 1911, Cyanider, Brakpan Mines, Ltd., Brakpan, Transvaal. 1912, Cyanider, Benoni Cons. Ltd., Benoni, Transvaal. 1912, Mine Sampler, Jupiter Gold Min., Ltd., Cleveland, Transvaal. 1912, Battery, East Rand Prop. Mines, Ltd., Boksburg, Transvaal. 1912–13, Mine Surveyor, Witwatersrand Deep, Ltd., Germiston, Transvaal. 1913–15, Experimental Chemist, charge sintering plant, Cerro de Pasco Min. Co., La Fundicion, Peru.

Present position: 1915 to date, Flotation operator, Anaconda Copper Min. Co.

George Henry Gilman, Claremont, N. H.

Proposed by Benjamin F. Tillson, R. M. Catlin, H. Comstock.

Born 1881, Richmond, Que. Academic in Canada, supplemented with private tuition by competent Engineers. 1899–1902, Special shop course in Engineering, Pattern making and Machine work, Canadian Rand Co., Sherbrooke, Que. 1902–04, Asst. Supt., Franklin Machine and Tool Co., Franklin, Pa. 1904–05, Master Mechanic, E. L. Smith Co., Barre, Vt. 1905–09, Engr., Hammer Rock Drills, Sullivan Machinery Co., Claremont, N. H. 1909–16, in charge design, development and manufacture Rock Drilling Machinery, Sullivan Machinery Co., Claremont, N. H.

Present position: Engineer in charge, Rock Drilling Dept., Sullivan Machinery Co.

Villeroy Gleason, Jr., Anaconda, Mont.

Proposed by Louis V. Bender, Harry S. Ware, Bayard S. Morrow.

Born 1888, Eureka, Kans. 1896–1903, Grade School, Kansas City, Mo. 1903–07, High School, Seattle, Wash. 1907–10, and 1911–12, College of Mines, Univ. of Wash. 1914–15, Student Assistant in Mining to Dean of College of Mines, Univ. of Wash. 1905–10, summers spent in mine, surveying, sampling and milling work necessary for degree. 1910–11, Underground Surveyor, Britannia, B. C 1912–13, Engineering work, Canadian Pacific R. R. 1913–14, Geologic Survey, with Milnor Roberts, Graham Island, B. C.

Present position: 1915 to date, Foreman in Concentrator, Washoe Reduction Works.

John Newton Goddard, Herculaneum, Mo.

Proposed by S. Paul Lindau, William Allen Smith, Franz Cazin.

Born 1876, Plainfield, N. J. 1898, Mass. Inst. of Technology, S. B. 1898, Chemist, Pueblo Smelt. & Ref. Co. 1899, Chemist, Monterey (Mexico) Plant, American Smelt. and Ref. Co.

Present position: 1905 to date, Supt. of Smelting, St. Joseph Lead Co.

Leslie Hall Goodwin, Boston, Mass.

Proposed by Allen H. Rogers, Edward E. Bugbee, Charles E. Locke.

Born 1886, Sanford, Me. 1906–08, English High School, Boston, Mass. 1908–12, Mass. Inst. of Technology, Boston, Mass., B. Sc. 1910, summer, Chainman, Oliver

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Iron Min. Co., Keewatin, Minn. 1911, summer, Underground miner, and Assayer, Utah Apex Min. Co., Bingham Canyon, Utah. 1912–14, Asst. Min. Engr., Quincy Min. Co., Hancock, Mich.

Present position: 1915 to date, Asst. Min. Engr., with Allen H. Rogers.

Clifford Gold Grant, Anaconda, Mont.

Proposed by Louis V. Bender, Harry S. Ware, C. R. Kuzell.

Born 1893, New Haven, Conn. 1910–13, Yale Univ., Sheffield Scientific School, Ph. B. 1913, Asst. Chemist, American Smelt. & Ref. Co., Durango, Colo. 1913, Testing Dept., International Smelt. & Ref. Co., Tooele, Utah. 1914, Mining, Silver King Mine, Park City, Utah. 1914, Asst. Chemist, Henry E. Wood Ore Testing Co., Denver, Colo. 1914–15, Min. Engr., Ottis Dev. and Power Co., Keystone, So. Dak. 1915–16, Asst. Testing Engr., Asst. Chemist, Anaconda Copper Min. Co.

Present position: Instrument man, Reverberatory Dept., Anaconda Copper Co.

Erasmus Haworth, Lawrence, Kans.

Proposed by H. A. Wheeler, Arthur Thacher, H. A. Buehler.

Born 1855, Warren Co., Iowa. 1881, Univ. of Kansas, B. S. 1888, Johns Hopkins Univ., Ph. D. 1883–92, Prof. of Science, Penn College, Iowa. 1892, Asst. Prof., Geology and Mineralogy, Univ. of Kansas. Promoted first to Associate Professor, then to full Prof. of Geology, Mineralogy and Mining. 1902–15, State Geol., resigned.

Present position: Head Prof., Geology and Mineralogy, Univ. of Kansas.

Clyde L. Holmberg, Jerome, Ariz.

Proposed by Robert H. Dickson, I. B. Joralemon, W. B. Gohring.

Born 1887, Duluth, Minn. 1902, Grammar School. 1906, High School. 1913, Graduated Michigan College of Mines, Sc. B., and M. E. 1908–09, Draftsman, Oliver Iron Mining Co., Duluth, Minn. 1909–10, Rodman, Oliver Iron Mining Co., Ishpeming, Mich. 1913–14, Surveyor Central Coal and Coke Co., Kansas City, Mo. 1914–15, Engineer, Arthur Iron Mining Co.

Present position: Engineer with United Verde Extension Mining Co.

Olof Hondrum, Cananea, Son., Mexico.

Proposed by M. J. Elsing, George Kingdon, L. D. Ricketts.

Born 1887, Norway. 1909, High School, Bemidji, Minn. 1913, School of Mines, Univ. of Minnesota, Minneapolis, Minn., E. M. 1913, Draftsman, Oliver Iron Min. Co., Eveleth, Minn. 1913–14, Engr., Cananea Cons. Copper Co., Cananea, Son., Mex. 1914–15, Assayer and Millman, Cerro de Plata Min. Co., Cerro de Plata, Son., Mexico.

Present position: 1915 to date, Chief Engr., Cananea Cons. Copper Co.

Albert Joseph Houle, Houghton, Mich.

Proposed by F. W. Sperr, F. W. McNair, W. E. Hopper.

Born, 1874, Ishpeming, Mich. 1878–89, Public Schools of Mich. 1889–93, Negaunee High School. 1893–96, Michigan Min. School, S. B., E. M. 1896–1907, Practice. 1896–99, Asst. in Min. Dept., Michigan College of Mines. 1899–1900, Surveyor, Copper Range Copper Co. 1900–01, Draftsman, Ellsworth Coal Co. 1901–02, Engr., United Globe Mines, Globe, Ariz. 1902–03, Construction, Encampment Smelt. Co., Wyo. 1903–06, Development work, Michigan Dev. Co., Pearce, Ariz. 1912–13, Smelt. work, Douglas, Ariz. 1907–16, Teaching.

Present position: Prof. of Metallurgy, Michigan College of Mines.

R. E. Howe, Anaconda, Mont.

Proposed by L. V. Bender, H. S. Ware, C. R. Kuzell.

Born 1878, Lodi, Ohio. 1895, Lodi High School. 1898–1900, Western Reserve Academy. 1904, Ohio State University, E. M. 1904–06, Assayer for Twentieth Century Mining & Power Co., and for Rabbit Foot Mining Co. 1907–16, With Anaconda Copper Mining Co., at Washoe Reduction Works as Testing Engineer, Chemist, Chief Chemist and Asst. Supt. of Converters.

Present position: Asst. Supt. of Converters, Washoe Reduction Works.

Hamilton H. Howry, New York, N. Y.

Proposed by R. M. Raymond, Robert Peele, Homer L. Carr.

Born 1893, Saginaw, Mich. 1909–11, East Denver High School, Denver, Colo. 1911–15, Columbia Univ., School of Mines, Met. E. 1915–16, U. S. Gold Corpn., Boulder, Colo.

Present position: Research work, Chile Exploration Co.

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Robert W. Hughes, Miami, Ariz.
Proposed by C. E. Arnold, H. I. Merricks, George R. Lehman.
Born 1891, Washington. 1906–10, High School, Mankato, Minn. 1915, Univ. of Wisconsin, B. S. 1913, nine months, underground, Anaconda Min. Co., Butte, Mont.
Present position: 1915 to date, Inspiration Cons. Copper Co.

Joseph Stewart Irwin, Tulsa, Okla.
Proposed by C. L. Severy, G. H. Cox, D. H. Radcliffe.
Born 1888, Louisiana, Mo. 1896–1907, Grammar School and High School, Louisiana, Mo. 1908–12, Missouri School of Mines, B. S. in Mine Engineering. 1913–16, Graduate work, Missouri School of Mines and Lehigh Univ. 1913–14, Instructor in Geology, Missouri School of Mines. 1915–16, Lehigh Univ., So. Bethlehem, Pa.
Present position: Geol., Carter Oil Co.

Ichiro Ito, Osaka, Japan.
Proposed by S. Karada, F. Fuketa, M. Yamashita.
Born 1888, Tokyo. 1913, Grad., Tokyo Imperial Univ., College of Engrg., Dept. of Met. 1913–16, Met. Engr., Ikinno Mine, Mitsubishi & Co.
Present position: Chief Engr., Mitsubishi Osaka Smeltering Works.

Garnett Alfred Joslin, Los Angeles, Cal.
Proposed by Burr A. Robinson, D. B. Myers, Charles E. Locke.
Born 1883, Fargo, So. Dak. 1902, Grad., High School, Los Angeles. 1904–09, Mass. Inst. of Technology, Civil and Min. Engrg., B. Sc. in mining. 1909–12, Asst. to Frank C. Loring, Toronto, in Cobalt and Porcupine mining districts. 1912, Goldfields American Dev. Co. of New York. 1912, Supt., stamp mill, Johnnie Min. & Mill. Co., Johnnie, Nev. 1912–13, Geol., Barber Asphalt Co., Venezuela. 1913–15, Consulting work, Los Angeles. 1915, Ray Cons. Copper Co., Ray, Ariz.
Present position: Consulting. Examination of mineral prospects and geological mapping of timber lands of Dierka Lumber & Coal Co. in Arkansas and Oklahoma.

Samuel Truman Kelsey, Jr., Thane, Alaska.
Proposed by Burton B. Nieding, Ralph L. Healy, B. L. Thane.
Born 1888, Eureka, Cal. 1902, Grad., Public School. Other education acquired by private study, practice and observation. 1910–12, Asst. to Examiner of Surveys, Field Division, U. S. General Land Office, Utah, Cal. and Alaska. 1913–14, Miner, sampler, assayer and surveyor, Kensington Min. Co., Kensington Mine, Kensington, Alaska.
Present position: 1915 to date, Mine Sampler and Associate Engr., Alaska Gold Mines Co., Perseverance Mine, Thane, Alaska. Four months as engineer on construction of steel tower, electric transmission line.

Donald Campbell Kemp, Rolla, Mo.
Proposed by H. A. Buehler, C. R. Forbes, Charles Y. Clayton.
Born 1889, Central City, Colo. 1915, Univ. of Colorado, B. A. 1915, Asst. Engr., Commonwealth M. & M. Co. 1916, Geol., Wichita Natural Gas and associated companies. 1916, summer, prospecting in Boulder Co., Colo.
Present position: Instructor in Geology and Mineralogy, School of Mines and Metallurgy.

Frank A. Kennedy, Madison, Wis.
Proposed by C. K. Leith, Bradley Stoughton, R. S. McCaffery.
Born 1879, Plainfield, Wis. 1906, Graduate University of Wisconsin, General Engineer. 1906–07, Oliver Iron Mining Co., as Asst. Engr. 1907–12, Chief Engr., The Shenango Furnace Co. 1912–16, present position, Asst. in Mining & Metallurgy, University of Wisconsin.

Rowland B. King, Ashcroft, B. C., Canada.
Proposed by Frederic Keffer, J. C. Haas, L. K. Armstrong.
Born 1888, Helena, Mont. 1903, Grammar School, Spokane, Wash. 1903–07, Apprentice Assayer, Spokane, Wash. 1906–13, Correspondence courses, International Correspondence School and American Soc. of Correspondence. 1913–16, Michigan College of Mines, Houghton, Mich., E. M. 1903–07, Assayer, The C. M. Fasset Co., Spokane, Wash. 1907, Chemist, The British Columbia Copper Co., Ltd., Greenwood, B. C. 1908, Assayer, Sampler, The Takilma Smelt. Co., Takilma, Ore.

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1908–09, Chemist, The British Columbia Copper Co., Greenwood, B. C. 1909, Assayer, Patrick Clark at Beowawe, Nev. 1910–13, Chief Assayer, The British Columbia Copper Co., Greenwood, B. C.
Present position: Genl. Supt., Highland Valley Min. & Dev. Co.

Bruno R. Koering, Detroit, Mich.
Proposed by E. H. Leslie, William H. Shearman, Thomas T. Read.
Born 1870, Germany. 1886–91, Univ. Halle Wittenburg and Freiburg, Germany, M. D. 1894–98, Miles Koering and Sharpe, Cons. Min. Engineers and Assayers, Hermovillo, Son., Mex. 1900, Min. Engr., Greene Cananea Co., Cananea, Son., Mex. 1901, Genl. Mgr., Manzanal Copper Co., Cananea, Son., Mex. 1902–12, operating my own mines and practising medicine in the State of Son., Mex. 1912–14, Perfecting my process and apparatus.
Present position: Pres. and Genl. Mgr., Koering Cyaniding Process Co.

Harry L. Kohlberg, Globe, Ariz.
Proposed by L. O. Howard, M. F. Mazany, W. F. Geiger.
Born 1884, Elberfeld, Germany. 1892–99, Ober-Realschule, Elberfeld. 1900, Night school: Drawing, English, etc. 1900–03, Grad., Konigliche Löbere Maschinenbauschule, Elberfeld. 1904–06, Draftsman, El Paso Foundry and Machine Co., El Paso, Tex. 1906–11, Draftsman, American Smelter Securities Co., Velardena, Mex. 1911–14, Draftsman, Repath and McGregor.
Present position: Mech. Engr., International Smelt. Co., Miami, Ariz.

Philip Kraft, Jr., So. Porcupine, Ont., Canada.
Proposed by C. D. Kaeding, Walter H. Aldridge, Frederick J. Pope.
Born 1890, New York City. 1910, Columbia College, B. S. 1912, Columbia School of Mines, E. M. 1914, Doctor of Engineering, Berlin.
Present position: Geol., Dome Mines Co., Ltd.

Max Kraut, Los Angeles, Cal.
Proposed by Francis J. Gallagher, S. W. Mudd, F. J. H. Merrill.
Born 1882, Berlin, Germany. 1900, General: Koenigstaedtischer, Real Gymnasium, Berlin. 1905, Engrg., Columbia School of Mines, E. M. 1905–06, Cerro de Pasco Min. Co., Peru. 1906–07, Minas de Cobre de Cutter Cove, Chile. 1907–10, Frank Klepetko, Peru, and U. S. 1911, Independent, Canada. 1912–13, Mina Mexico, Son., Mex. 1914–16, Phelps, Dodge & Co.
Present position: Southwestern Engineering Co.

Herbert Benjamin Kroeger, San Nicolas, Cuba.
Proposed by Robert P. Bowles, George F. Kunz, C. M. Weld.
Born 1888, Bridgeport, Conn. 1906, Graduate Bridgeport High School. 1906–07, Academic Dept., Columbia University. 1911, Graduate Michigan College of Mines with degree of B. Sc. and E. M. 1911–13, Chem. & Foreman with American Smelting & Refining Co., Perth Amboy, N. J. 1914, with Frank Klepetko, New York City.
Present position: 1914 to date, Chem., Canto Mining Co., San Nicolas, Oriente Prov., Cuba.

Gavin William Laurie, St. Louis, Mo.
Proposed by Arthur P. Watt, E. E. Dieffenbach, Edward Randolph.
Born 1890, St. Louis, Mo. 1904–08, High School, East St. Louis, Ill. 1908–10, Practical Engineering Schools, St. Louis, Mo. 1911–12, Union Electric Light Co., St. Louis, Mo. 1912, Merchants Power Co., Memphis, Tenn. 1913, Laclede Christy Play Product Co., St. Louis, Mo. 1914, St. Louis Smelt. & Ref. Co., St. Louis, Mo. 1915, Mine La Motte Co., Mine La Motte, Mo.
Present position: Plant Engr., Balbach Smelt. & Ref. Co., Newark, N. J.

Charles Shepard Lee, Douglas, Ariz.
Proposed by P. P. Butler, J. Moore Samuel, C. Legrand.
Born 1892, Socorro, New Mexico. 1914, Mass. Inst. of Technology, B. S. 1914–15, Half year post graduate at same school.
Present position: 1915 to date, Asst. to Testing Engr., Copper Queen Cons. Min. Co., Reduction Works.

Stuart Harrison Levison, Hayden, Ariz.
Proposed by R. B. Green, H. T. Murray, J. J. Ormsbee.
Born 1890, Atlanta, Ga. 1904–08, Manual Training High School, Brooklyn,

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N. Y. 1908–12, School of Mines, Columbia Univ., Met. E. 1912–14, American Smelt. & Ref. Co., Mexico. 1914–15, American Smelt. & Ref. Co., Hayden, Ariz.
Present position: Chief Chemist, American Smelt. & Ref. Co.

John Francis Linthicum, Bauxite, Ark.
Proposed by James E. Little, John R. Fordyce, Charles F. Rand.
Born 1884, Annapolis, Md. 1902, St. Johns College, Annapolis, Md., A. B. 1903–04, Asst. Chem., Maryland Steel Co., Blast Furnace and Coke Oven Laboratories. 1904–05, Chem., Maryland Steel Co., wheel works. Asst. Chem., Baltimore & Ohio R. R. 1906, Asst. Chem., New York Testing Laboratory, Long Island City, N. Y. 1907–13, Chief Chem., The Spanish-American Iron Co., Felton, Cuba.
Present position: 1913 to date, Chief Chem., and Mines Engr., The American Bauxite Co.

Octavius Louis Lumaghi, St. Francois, Mo.
Proposed by W. E. McCourt, W. H. Comins, A. S. Watt.
Born 1893, St. Louis, Mo. St. Louis University (preparatory, high school and college). 1913–14, Yale—Sheffield, Ph. B., M. E. 1915–16, Missouri School of Mines, B. Sc. in M. E.
Present position: Engineer, St. Louis Smelting & Refining Co.

Robert McIntosh, Lake Linden, Mich.
Proposed by C. H. Benedict, G. L. Heath, L. D. Ricketts.
Born 1880, Grinnell, Iowa. 1901, Graduated Grinnell College, Grinnell, Iowa, Ph. B. 1905, Graduated Cornell University, M. E. 1905–08, Drafting room and erecting shop Nordberg Mfg. Co., Milwaukee. 1908–11, Testing Engr., Tamarack, Osceola Consolidated, Ahmeek & Isle Royale Mining Companies, Calumet, Mich. 1911–15, Drafting room, Calumet & Hecla Mining Co., Calumet, Mich.
Present position: 1915 to date, Asst. Supt., Stamp Mills, Calumet & Hecla Mining Co.

Hugh J. Maguire, Anaconda, Mont.
Proposed by Louis V. Bender, Harry S. Ware, Pierce Barker.
Born 1882, Belfast, Ireland. 1890–98, Elementary education, Ireland. 1906–12, Preparatory and College course, Univ. of Idaho, Moscow, Idaho, B. S. in Min. Engrg. 1911–12, Dept. of Mining, under direction of Prof. R. S. McCaffery. 1903, Mucker, Blackhawk and Wyoming Mines, Federal Min. and Smelt. Co., Kellogg, Idaho. 1904–06, Millman, Sweeney Mill, Federal Min. and Smelt. Co., Kellogg, Idaho. 1907–08, summers, Millman, Mammoth Mill, Federal Min. and Smelt. Co., Wallace, Idaho. 1909–10, Millman, Sweeney Mill, Federal Min. and Smelt. Co., Wallace, Idaho. 1911, summer, Mill Foreman, Whitman Mining Co., Pearl, Idaho. 1912, Assayer, Danaher Min. Co., Hailey, Idaho. 1912, Testing Dept., Anaconda Copper Min. Co., Washoe Reduction Works, Anaconda, Mont.
Present position: Supt., Leaching Plant, Reduction Works, Anaconda Copper Min. Co.

Walter R. Malm, San Francisco, Cal.
Proposed by Charles W. Merrill, T. A. Rickard, W. H. Shockley.
Born 1893, San Francisco. 1908–12, Cogswell Polytechnical College, San Francisco, Cal. 1913, Mech. Engrg., Cornell Univ., Ithaca, N. Y. 1913–16, Min. and Met., Stanford Univ., Palo Alto, Cal. 1912–13, Assayer, Bookkeeper, Supt., East Eureka Min. Co., Sutter Creek, Cal. 1914, summer, Assayer, Mammuth Copper Co., Kennett, Cal. 1915, summer, Asst. Chemist, Noble Electric Steel Co., Heroult, Cal. 1915–16, Night Chemist, Pacific Coast Steel Co., South San Francisco, Cal. 1916, Chemist, Smith Emery & Co., San Francisco, Cal.
Present position: Chemist, Merrill Met. Co.

Edgar George Ross Manwaring, Kendall, Mont.
Proposed by G. T. McGee, C. W. Goodale, John Gillie.
Born 1883, Imlay City, Mich. 1901, Graduate Ann Arbor High School. 1901–03, University of Michigan in regular Mechanical Engineering Course. 1905, B. Sc. in Mining at University of Missouri, Rolla. 1905–06, Railroad work with Greene Cons. Copper Co., Cananea, Mex. 1906–07, Min. Engr., with San Pedro Copper Co., Cananea, Mex., until consolidation with Greene-Consolidated. 1907–08, Field Engr. for Calumet and Arizona, Courtland, Ariz. Made reports for Calumet and Arizona on their exploratory projects. 1908, Supt., Great Western Copper Co., at Courtland,

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Ariz. 1909, Six months with Barnes-King Development Co., Kendall, Mont. 1909–11, Min. Engr. for Kendall Gold Mining Co. 1911–12, Homesteading and surveying farm lands.
Present position: 1912 to date, with Barnes-King Development Co.

Earl Robert Marble, Hayden, Ariz.
Proposed by R. B. Green, J. J. Ormsbee, H. T. Murray.
Born 1890, Attleboro, Mass. 1895–1907, Public Schools, Attleboro, Mass. 1908–12, Chemical Engrg., Tufts College, B. S. 1912–14, Chief Chemist, Nevada Douglas Copper Co. 1914–16, Plant experience, American Smelt. & Ref. Co., Hayden, Ariz. Plant Chemist, Asst. to Master Mechanic, etc.
Present position: Furnaceman, American Smelt. & Ref. Co.

Donald Markle, Hazleton, Pa.
Proposed by R. V. Norris, Henry S. Drinker, A. Markle.
Born 1892, Hazleton, Pa. 1906–11, Hill School. 1911–14, Yale Scientific School, Ph. B.
Present position: 1914 to date, Lehigh Univ., School of Mines.

C. E. Meissner, Miami, Ariz.
Proposed by C. E. Arnold, George R. Lehman, W. F. Palmer.
Born 1890, Birmingham, Ala. 1905–09, Polytechnic Preparatory School. 1909–14, Grad., School of Mines, Columbia Univ., M. E. 1914–16, Rodman, Raise Engr., Shift Boss, Inspiration Cons. Copper Co.
Present position: Shift Boss, Inspiration Cons. Copper Co.

Samuel Melitzer, New York, N. Y.
Proposed by Robert Peele, Homer L. Carr, E. J. Hall.
Born 1888, New York City. 1905–09, Columbia College, B. S. 1909–11, Columbia School of Mines, E. M. 1911–12, Assaying, Kerr Lake Min. Co. 1912–14, Engrg. and operating, Ray Cons. Copper Co. 1914–16, Engrg. and operating, Braden Copper Co.
Present position: Assisting in opening a mine, Pinar del Rio, Cuba.

A. R. Minner, Independence, Colo.
Proposed by Robert M. Keeney, Irving T. Snyder, Justin H. Haynes.
Born 1872, Springfield, Mo. 1878–90, Public Schools at Augusta, Iowa. 1892, Business College, Stockton, Cal. 1902–08, International Correspondence School, Scranton, Pa. 1891–96, Millwright, Stockton, Cal., 1897–99, Millwright Mountain Lion Mill, Republic, Wash. 1900, Foreman millwright Republic G. M. Co., Republic, Wash. 1901–02, Const. Engr., U. S. R. & R. Co., Cañon City, Colo., 1902–03, Const. Engr., Utah Copper Co., Bingham, Utah. 1904, Const. Engr., Black Pearl Mill, Pearl, Idaho. 1905–06, Const. Engr., Golden Cycle Mill, Colorado Springs, Colo. 1907, Const. Engr., Strattons Ind., Victor, Colo. 1908, Const. Engr., Eagle Mill, Red Cliff, Colo. 1909, Const. Engr., Penn Mines Co., Argentine, Colo. 1910–11, Const. Engr., & Mill Supt., Strattons Ind., Victor, Colo. 1912–13, Const. Engr., & Mill Supt., Ozark M. & M. Co., Magdalena, N. M.
Present position: 1914 to date, Mill Supt., Vindicator Cons. G. M. Co.

Karl Irwin Mohler, Warren, Ariz.
Proposed by Ira B. Joralemon, E. E. Whiteley, Philip D. Wilson.
Born 1889, Freedom, Pa. 1895–1903, Public Schools, Freedom, Pa. 1903–07, Preparatory work, Beaver High School, Beaver, Pa. 1907–11, School of Mines, Univ. of Pittsburgh, Pittsburgh, Pa., E. M. 1910–11, Engrg. Dept., Cascade Coal & Coke Co., Tyler, Pa. 1911–13, Shift Foreman, Creighton Mine, Canadian Copper Co., Copper Cliff, Ont. 1913–14, Engrg. Dept., Cananea Cons. Copper Co., Cananea, Son., Mex.
Present position: 1914 to date, Engrg. Dept., Calumet and Arizona Min. Co.

Charles L. Montague, El Paso, Tex.
Proposed by David Cole, J. G. Bergman, H. L. Hollis.
Born 1880, Bandera, Tex. 1887–96, Public School, Bandera, Tex. 1900–01, Stenographer to Genl. Mgr., Material Agent, Mexican International Railroad. 1901–03, Stenographer, Material Agent and roadmaster, Cananea, Yaqui River & Pacific R.R. Co., Naco, Son., Mex. 1903–07, Secretary to Genl. Mgr., Cananea Cons. Copper Co., at Cananea, Mex. 1907–11, Private Secretary to W. C. Greene, in New

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York and Mexico. 1911–12, Treasurer, Dwight & Lloyd Metallurgical Co., New York. 1912–16, Cashier, Mercantile Banking Co., Cananea, Son., Mex. Also American Consular Agent.

Present position: Mgr., The Cusi Mining Co.

Henry Schuyler Montgomery, Warren, Ariz.

Proposed by A. G. McGregor, L. D. Ricketts, John C. Greenway.

Born 1882, Muskegon, Mich. 1900–03, Univ. of Michigan. 1903–05, Stanford Univ., Cal., B. S. in Chemistry. 1905–07, Draftsman, Llanos de Oro Min. & Mill. Co. 1907–08, Draftsman, Woodruff Construction Co., San Francisco. 1908–09, Min. Mach. Dept., Union Tool Co., Los Angeles. 1907–10, Engrg. Dept., Parkinson and Bergstrum, Architects, Los Angeles. 1910–15, Engr., MacKay Copper Process Co. 1915, Charge of Electrolytic Tank House, New Cornelia Copper Co.

Present position: Designer, New Cornelia Copper Co.

William A. Neill, Denver, Colo.

Proposed by P. M. McHugh, A. L. Blomfield, L. D. Ricketts.

Born 1880, Georgetown, Colo. 1886–96, Grammar School and East Denver High School. 1896–1900, Special night school in Mathematics and Mechanical Drawing. 1900–05, Special Instruction from Mr. F. E. Shepard, Prest., Denver Engineering Works. 1896–99, Apprentice in Mechanical shop of Denver Engineering Works. 1899–02, Mechanical Draftsman, Denver Engineering Works. 1902–05, Chief Draftsman and Asst. Supt., Denver Engineering Works. 1905–06, Asst. Chief Draftsman Allis-Chalmers Co., Mining Dept., Milwaukee, Wis. 1906–09, Chief Draftsman, Allis-Chalmers Co., Mining Dept., Milwaukee, Wis. 1909–16, Engineer in charge concentrating, cyanide and hoisting machinery Mining Dept., Allis-Chalmers Co.

Present position: Mechanical Engineer, The Dorr Co.

Henry William Nichols, Superior, Ariz.

Proposed by L. D. Ricketts, W. C. Browning, I. A. Ettlinger.

Born 1883, Goshen, Ind. 1889–97, Public Schools, Cook County, Ill. 1898–1902, Evanston Township High School, Evanston, Ill. 1903–09, Michigan College of Mines, B. S. and E. M. 1904, Construction Inspector, Champion Copper Co. 1906–08, Asst. Engr., United Verde Copper Co., Jerome, Ariz. 1910, Assayer, Nevada Cons. Steptoe Valley Smelter, McGill, Nev. 1912, Miner, Lion Hill Cons. Min. Co., Ophir, Utah. 1913, Leaser, Indian Queen Property, Newhouse, Utah.

Present position: 1913 to date, Concentrator Met., Magma Copper Co.

Philip Edward Nolan, Ludwig, Nev.

Proposed by E. W. Westervelt, Archie J. Orem, Erich J. Schrader.

Born 1887, Marinette, Wis. 1905, Grad., High School, Greens Bay, Wis. 1905–07, Ripon College, Ripon, Wis. 1910–13, Colorado School of Mines, Golden, Colo., E. M. 1907–08, Rodman and instrumentman, Colorado Fuel and Iron Co. 1908–10 and 1913–16, including summers of 1910–13, on general construction work as survey man, Junior Engr. and Asst. Engr., U. S. Reclamation Service.

Present position: Mine Engr., Nevada Douglas Cons. Copper Co.

J. Edward Ogden, New York, N. Y.

Proposed by A. R. Ledoux, Thomas Robins, J. A. Bensel.

Born 1870, Brooklyn, N. Y. 1890–93, Morton Iron Works, Brooklyn, Manufacturers, Gas Works Apparatus. 1893–1907, Merchant, purchase and sale contractors machinery and supplies. 1900–13, Manufacturer of bolts, forgings, etc., plant, Bayonne, N. J. 1903 to date: Prest., Star Expansion Bolt Co., Manufacturers Hardware Specialties. 1905 to date: Pres., J. Edward Ogden Co., Manufacturers and Contracting Engineers.

Present position: As above.

Tatsuro Otagawa, New York, N. Y.

Proposed by R. W. Raymond, George F. Kunz, Bradley Stoughton.

Born 1890, Tokio, Japan. Graduated 1913, Dept. of Mining, Tokio Imperial University. Studied at Technische Hochschule in Aachen, Germany, in 1914 and also at Royal School of Mines, London, and University of Leeds in 1914 to 1916.

Present position: Lecturer of Mining, Tohoku Imperial University, Japan.

James Caldwell Park, Florence, Colo.

Proposed by L. V. Emanuel, H. H. Utley, H. J. Seaman.

Born 1876, Cranford, N. J. 1899, Grad., Princeton Univ., C. E. 1899–1901, Freight Yard Construction, Lehigh Valley Ry. 1901–02, Engrg. Dept., Central R. R. of N. J. 1902–04, Asst. Engr., Construction on Cement Plants, Atlas Portland

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Cement Co., Northampton, Pa. 1904–05, Yard Construction, Asst. Engr., N. Y. C. & H. R. R., Watertown, N. Y. 1905–13, Engineering and contracting business.
Present position: 1913 to date, Supt., Florence Works, Standard Oil Co. (Indiana).

Harold Cleveland Plummer, Cananea, Son., Mex.
Proposed by M. J. Elsing, L. D. Ricketts, George Kingdon.
Born 1884, Cambridge, Mass. 1906, Mass. Inst. of Technology, B. S. 1905, Min. Engr., Adventure Cons. Copper Co., Greenland, Mich. 1906–09, Min. Engr., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz. 1909–10, Mine Supt. for Lewisohn Bros. in Homestead, Ore. 1910–12, Chief Engr., Tennessee Copper Co., Ducktown, Tenn. 1912–13, Efficiency Engr., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz. 1913–16, Chief Min. Engr., Cananea Cons. Copper Co., Cananea, Son., Mex.
Present position: Foreman, Sierra de Cobre Mine, Cananea Cons. Copper Co.

Philip James Arthur Plummer, Port Pirie, So. Aust.
Proposed by G. C. Riddell, C. M. Warner, J. Jobson.
Born 1881, South Aust. 1892, Matriculated Adelaide University. 1896, Associate Diploma Metallurgy, Mining, So. Australia School of Mines. 1896–99, Demonstrator, Metallurgy and Chemistry, S. A. School of Mines. 1899–1916, Assayer, Chemist and Metallurgist, Broken Hill Proprietary Co. and later the Associated Smelters Proprietary, Ltd., Port Pirie, So. Aust.
Present position: Met., Associated Smelters.

George Wallace Prince, Cananea, Son., Mexico.
Proposed by L. D. Ricketts, George Kingdon, C. E. Mills.
Born 1882, Pierpont, Ohio. 1894–98, High School, Pierpont, Ohio. 1898–99, Mt. Union College, Alliance, Ohio. 1899–1903, Ohio State University, Columbus, Ohio, mining engineering and special chemical course. 1904–09, Old Dominion Copper Mining & Smelting Co., Globe, Ariz., as Foreman night yard, Sampler, Asst. Chemist, and Assayer. 1910–16, Cananea Consolidated Copper Co., Reverberatory Furnace Foreman, Night Smelter Supt., and Asst. Smelter Supt.
Present position: Smelter Supt., Cananea Cons. Copper Co.

Warren C. Prosser, Silverton, Colo.
Proposed by L. V. Emanuel, George E. Collins, H. H. Utley.
Born 1886, New Jersey Heights, N. J. 1903–08, Colorado School of Mines, Golden. Mine and mill work, San Juan County, Colo. Among others Silver Lake property of Garfield Smelt. Co. For 10 years practicing mining engineering, Silverton, Colo.
Present position: Engr. in charge Intersection M. & M. Co., Highland Mary M. & M. Co.

Ernest Cope Ramsey, Columbus, Ohio.
Proposed by P. McD. Biddison, D. M. Armstead, H. C. Reeser.
Born 1882, Wattsville, Ohio. 1888, Grade school. 1891, Grade School at Columbus. 1897, High School at Columbus. 1910, Ohio State Univ., M. E. 1911, Freshman, Ohio State Univ. 1903, Asst. to Chief Engr., Columbus, Delaware & Marion Electric Railway. 1904, Steel car draftsman, The Kilbourne & Jacobs Mfg. Co., Columbus, Ohio. 1907, student, Ohio State Univ. 1910, Map draftsman, The United Fuel Gas Co., Charleston, W. Va. 1913, Agent, The Ohio Fuel Supply Co., Columbus, Ohio.
Present position: 1915 to date, Min. Engr., The Ohio Fuel Supply Co.

George William Rathjens, St. Paul, Minn.
Proposed by C. W. Tubby, Fred J. Leacey, Rukard Hurd.
Born 1881, Boston, Mass. 1908–10, University of Illinois, B. Sc. in Civil Engineering. 1900–03, Chief of party, Chicago Great Western Ry. M. of W. 1903–04, Instrumentman U. S. Engineers. Asst. Supt., Barber Asphalt Paving Co., Transitman Inter-urban Ry. 1904–06, Asst. Engr., Minneapolis and St. Louis Ry. 1906, Resident Engr., Tidewater & Deepwater Ry. 1906–07, Supt., Twin City Brick Co. 1907–08, Asst. Road Master, Great Northern Ry. 1909–10, Asst. Engr., Chicago, Milwaukee and St. Paul Ry.
Present position: 1911 to date, Supt., Twin City Brick Co., and Vice Prest. since March, 1915.

George Kingsbury Reeder, Klockmann, Idaho.
Proposed by J. C. Ralston, L. K. Armstrong, J. M. Porter.
Born 1887, McPherson, Kans. Grade Schools, Spokane. 1903–07, High School,

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Spokane. 1907–12, Washington State College, Pullman, Wash., B. S. in Min. Engrg. 1912–13, Prospecting and development work in Idaho, Washington and Montana. 1913, Miner, Schaunessey Hill Min. Co. 1914, Millman, Mad Ox Min. Co., Cal. 1914, Miner, Cornucopia Mines Co., Ore. 1915–16, Surveyor, assayer, engineer, Idaho Continental Min. Co.

Present position: Engr., Idaho Continental Co.

Herman H. Retter, Pilares de Nacozari, Son., Mex.

Proposed by J. S. Williams, H. T. Hamilton, C. I. Schultz.

Born 1870, Hanover, Germany. 1878–85, attended public school in Germany. 1904–07, studied under direction of International Correspondence School their mining course. 1908–09, Studied Spanish Course of same school. At present am studying the Alexander Hamilton Business Course. 1899–1907, Mucker, miner, trackman, pipeman, and the last 3½ years of that period as shift boss, Cons. Mercur Gold Mines Co., Mercur, Utah. 1907–11, Mine Foreman, Minas del Tajo, Rosario, Sinaloa, Mex. 1911–12, Shift boss, Division Foreman, Moctezuma Copper Co., Nacozari, Son., Mex.

Present position: 1912 to date, Asst. Mine Supt., Moctezuma Copper Co.

Ernest John Ristedt, Miami, Ariz.

Proposed by Emory M. Marshall, I. H. Barkdoll, Rudolf Gahl.

Born 1886, Niles, Ohio. Public and High School, Cleveland, O. 1905–09, Colorado School of Mines, E. M. 1909–10, with N. H. Brown, Min. Engr., Idaho Springs, Colo. 1910–11, ore concentration, with J. G. Roberts, Idaho Springs, Colo. 1911, Chemist, Miami Copper Co., Miami, Ariz. 1911–12, Engr. and Assayer, Princesa Min. Co., Cusihuiriachic, Chih., Mex. 1912–13, Testing Dept., Miami Copper Co. 1913–16, Assayer, Asst. Chemist, Chief Chemist, Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.

Present position: Inspiration Cons. Copper Co.

James Bertrand Rogers, Pikeville, Ky.

Proposed by E. C. Lee, P. N. Moore, R. H. Sweetser.

Born 1870, Mt. Gilead, O. 1890, Grad. Rayen High School, Youngstown, O. 1896, Grad. Ohio State Univ., Columbus, O., E. M. 1896–97, Civ. Engr. in railroad location and construction. 1898–1900, Chemist, U. S. Steel Corpn. 1901–06, Asst. Mgr., Salem Iron Co., Blast Furnace, Leetonia, O. 1906–13, Genl. Supt., Blast Furnace Dept., Ashland Iron & Min. Co., Ashland, Ky. 1913–15, Genl. Supt., Hamilton Otto Coke Co. and Miami Iron & Steel Co., Hamilton, O.

Present position: 1915 to date, Proprietor, Kewanee Coal Min. Co., Kewanee, Ky. Holder of letters patent for improved eyesight for blast furnace tuyères.

William H. Selden, Jr., Iron River, Mich.

Proposed by C. E. Lawrence, A. Formis, G. R. Waeber.

Born 1885, Portland, Conn. 1907–08, Exeter Academy. 1915–17, Special non-matriculated work in geology dept., Columbia University. 1908 to date: Worked in the Menominee district of Michigan, also the Marquette field, but largely in connection with our own properties where some geological experience has been acquired.

John Stephen Shedden, New York, N. Y.

Proposed by H. N. Spicer, Noel Cunningham, Clive S. Newcomb.

Born 1880, Browns Park, Colo. 1900, Univ. of Wyoming, B. S. 1904, Cornell Univ., M. E. 1904–06, Beaver Asbestos Co., Thelford Mines, Quebec. 1907–08, Bradley Engineering and Machinery Co., Spokane. 1908–09, United Iron Works, Spokane. 1909–12, Engineering work for self, Spokane and North Yakema, Wash. 1913–14, Jause Bros. Boomer and Hughes, Calgary, Alta.

Present position: Sales Engr., The Mine and Smelter Supply Co.

Palmer Godden Snedecor, Globe, Ariz.

Proposed by P. G. Beckett, I. H. Barkdoll, W. B. Cramer.

Born 1891, Birmingham, Ala. 1896–1903, Public School, Birmingham, Ala. 1903–06, University High School, Tuscaloosa, Ala. 1906–10, Univ. of Alabama, B. S. 1911–12, Univ. of Alabama, C. E. 1910–11, Asst. Engr. and Inspector, Mt. Hood Ry. and Power Co., Portland, Ore. 1912–13, Draftsman, Tennessee Coal and Iron Co., Birmingham, Ala. 1915–16, Asst. Engr., Duquesne Min. and Red. Co., Duquesne, Ariz. Was shift boss in flotation mill for 3 months during 1916.

Present position: Stope Engr., Old Dominion Copper Min. and Smelt. Co.

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Pedro Leopoldo Sol, Comodoro, Rivadavia, Rep. of Argentina.
Proposed by R. S. Haseltine, Ralph Arnold, W. R. Hamilton.
Born 1880, Concepcion del Uruguay, Rep. of Argentina. 1905, Diploma, Ingenieur des Arts et Manufactures de l'Ecole Centrale de Paris. 1906–07, Mining Inspector, Direccion General de Mines, Geologia e Hidrologia. 1911–16, Mgr. of the oil fields of Comodoro, Rivadavia, Rep. of Argentina.
Present position: Genl. Mgr. of the oil field of Comodoro, Rivadavia.

Fletcher B. Speed, Jr., Charlottesville, Va.
Proposed by J. S. Grasty, J. W. Richards, Howard Eckfeldt.
Born 1892, Baltimore, Md. 1909, Preparatory School, Baltimore, Md. 1913, Lehigh Univ., E. M. 1913, Engrg. Dept., Lehigh Coal and Navigation Co. 1914–15, Commercial Engrg. Dept., Chesapeake and Potomac Tel. Co., Baltimore, Md. 1916, Engr., Virginia Lead and Zinc Corpn. 1916, Associated with J. S. Grasty, in management and development of mineral properties.
Present position: Managing Engr., Virginia Barytes Co., etc.

Louis W. Stedman, Paragon, Idaho.
Proposed by L. K. Armstrong, Charles H. Goodsell, George A. Laird.
Born 1861, Dixton, Ill. 1881, Common High School. 1882, Cornell College, Mt. Vernon. Thirty-three years' experience in mining.
Present position: Mgr., Paragon Cons. Min. Co.

Eugene W. Steele, Hayden, Ariz.
Proposed by J. J. Ormsbee, R. B. Green, H. T. Murray.
Born 1889, Denver, Colo. 1911, Colorado College, B. Sc. in Civ. Engrg. 1911, Mill operator, Ray Cons. Copper Co., Hayden, Ariz. 1912, Construction Engr., American Smelt. & Ref. Co., Hayden, Ariz.
Present position: 1912 to date, Construction Engr., American Smelt. & Ref. Co.

Paul Alfred Steger, Miami, Ariz.
Proposed by C. E. Arnold, George R. Lehman, C. E. Mills.
Born 1885, Lichtensteig, Switzerland. 1904–07, Swiss Federal Polytechicum, Zurich. 1907–09, Travel in Europe. 1909–13, Office work, Surveyor, Foreman, Supt., Cia. Ma. La Parrena, Sierra Mojada, Coah., Mex. 1913–14, Visit to Europe. 1914–16, Asst. to Dr. Gahl, Inspiration Mill.
Present position: Met. Experimental work, Inspiraton Cons. Copper Co.

Paul Stein, El Paso, Tex.
Proposed by C. W. Badgley, Kuno Doerr, L. D. Ricketts.
Born 1890, Germany. 1907–12, New Mexico School of Mines, B. S. and M. E. 1912, Asst. Engr., Socorro Mines Co., Mogollon, New Mex.
Present position: 1912 to date, Chemist, El Paso Smelt. Works, of the Cons. Kansas City Smelt. & Ref. Co.

Henry J. Sterk, Anaconda, Mont.
Proposed by James K. Murphy, Louis V. Bender, Harry S. Ware.
Born 1891, L'Anse, Mich. 1910, L'Anse High School. 1915, Michigan College of Mines, B. Sc. in M. E. 1914, Ahmeek Mining Co. 1915, Keweenaw County Highway Commission.
Present position: 1916 to date, Anaconda Copper Mining Co.

Charles E. Stuart, El Paso, Tex.
Proposed by William J. Deavitt, R. F. Manahan, S. F. Shaw.
Born 1886, Princeton, Ind. 1902, San Diego High School, Cal. 1902–07, Univ. of Cal., B. S. 1907, Tonopah Min. Co. 1908–10, Montana Tonopah Min. Co. 1910–15, Engr., American Smelt. & Ref. Co. and American Smelters Securities Co.
Present position: Supt., Bonanza Unit, American Smelters Securities Co.

W. P. Sykes, Cleveland, O.
Proposed by Charles H. Fulton, J. Burns Read, Zay Jeffries.
Born 1893, Cleveland, O. 1916, Met. Engrg., Case School of Applied Science, B. S. 1916, American Steel Foundries, Alliance, O.
Present position: Heaterhelper, Coke Plant, The River Furnace Co.

Thomas Taylor, Clarkdale, Ariz.
Proposed by L. D. Ricketts, Robert E. Tally, C. E. Mills.
Born 1865, Swansea, So. Wales. A few years of public school education. Practical smelterman of 30 years' experience. Supt., United Verde Smelter, for about

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20 years. Previous work as foreman, boss and regular smelter work in Arizona, Montana, New Jersey, Pennsylvania, and Wales.
Present position: Smelter Supt., United Verde Copper Co.

Arthur T. Thomson, Salida, Colo.
Proposed by Fred Johnston, Barry Hogarty, E. H. Laws.
Born 1881, Princeton, Ill. 1905, Grad., Colorado School of Mines, M. E. 1905–07, Assayer and Chemist, Pueblo Plant, American Smelt. & Ref. Co. 1907–09, Chemist, Garfield Smelter, American S. S. Co. 1909–16, Chemist and Asst. Supt., O. & C. S. & R. Co.'s plant, Salida, Colo.
Present position: Asst. Supt., O. & C. S. & R. Co.

Andrew Glasgow Van Eman, Boise, Idaho.
Proposed by Walter Hovey Hill, Frank E. Johnesse, F. C. Brown.
Born 1882, Washington, Iowa. 1899, Great Falls High School, Great Falls, Mont. 1906, special work, Univ. of Wash., Seattle, Wash. 1899–1903, Chemist, Boston and Montana Cons. Copper and Silver Min. Co., Great Falls, Mont. 1904–05, Chemist, Harry Lewis' Custom Assay Office, Butte, Mont. and Cataract Copper Co., Basin, Mont. 1906–07, Asst. Chemist, United Verde Copper Co., Jerome, Ariz. 1908–11, Chief Chemist, Nevada Cons. Copper Co., Steptoe Valley Plant, McGill, Nev. 1911–16, Proprietor, Custom Laboratory, Boise, Idaho.
Present position: Proprietor, A. G. Van Eman, Assayer and Chemist.

Arthur Clifford Veatch, Whitehall, London, England.
Proposed by David T. Day, Alfred J. Brooks, George Otis Smith.
Born 1878, Evansville, Ind. Indiana Univ., Cornell Univ., Wisconsin Univ. 1898–1900, Asst. State Geol., Louisiana. 1900–01, Geol., Houston Oil Co. 1902–10, Field Asst., Asst. Geol. and Geol., U. S. Geol. Survey. 1908–10, Chairman, Old Land Classification Board and Coal Land Classification Board, etc., U. S. Geol. Survey. 1910–11, Geol., General Asphalt Co.
Present position: 1912 to date, Chief Geol., S. Pearson Sons, Ltd.

Clifton E. Visel, Pilares, Son., Mex.
Proposed by J. S. Williams, H. T. Hamilton, Casper I. Schultz.
Born 1887, Highwood, Conn. 1905, Grad., New Haven High School, New Haven, Conn. 1908, Grad., Sheffield Scientific School, Yale Univ., Ph. B. 1908–09, Mucker, Cresson Cons. Gold M. & S. Co. 1909–10, Inspector of steel construction for Mr. H. Kenyon Burch at Miami Concentrator, Miami, Ariz. Worked as operator in mill afterward for nine months. 1910, Timberman helper, Federal M. & S. Co., Mullan, Idaho. 1911, Timberman and jigger boss, Miami Copper Co's. Mine. 1912–15, worked as engineer in engineering dept., and later in smelter on converters, Old Dominion Min. & Smelt. Co., Globe, Ariz.
Present position: 1915 to date, Engr. and Chief Engr., Moctezuma Copper Co.

Constantin Wagner, Coquimbo, Chile, So. Amer.
Proposed by C. M. Loeb, E. A. C. Smith, Samuel J. Gormly.
Born 1879, Germany. German Gymnasium. Metallgesellschaft, Frankfort on M., Germany. Ore Trading Co., Ltd., Santiago, Chile.
Present position: Genl. Mgr., Fundicion Guayacan.

Guy Porter Watson, Rancagua, Chile.
Proposed by L. E. Grant, B. T. Colley, W. J. Turner.
Born 1886, Greeley, Colo. 1910, Colorado School of Mines, Met. Engr. 1912, Colorado School of Mines, M. E. 1908, Summer work in Cripple Creek, Colo. 1907–09, Summer work as instrument man with construction company in Colorado, New Mexico and Wyoming. 1910–11, Smelter research dept., Steptoe Valley Smelting & Mining Co., McGill, Nev. 1911–12, Chemist, Steptoe Valley Smelting & Mining Co., McGill, Nev. 1912–14, Control Chemist, Steptoe Valley Smelting & Mining Co., McGill, Nev.
Present position: 1914 to date, Met., Braden Copper Co.

William H. Webster, Douglas, Ariz.
Proposed by Forest Rutherford, G. H. Dowell, P. P. Butler.
Born 1884, Truxton, N. Y. 1906, Amherst College, B. A. 1906–07, Weighmaster, Copper Queen Smelter. 1907–08, Underground timekeeper, Copper Queen Mines. 1908–12, Met. Statistician, Copper Queen Smelter. 1912–16, Shift Supt., Copper Queen Smelter.
Present position: Asst. to Genl. Mgr., Copper Queen Cons. Min. Co.

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Harry V. Welch, Los Angeles, Cal.
Proposed by H. W. Morse, Walter A. Schmidt, L. D. Ricketts.
Born 1882, Greeley, Colo. 1901, Grad., Colorado State Normal School, B. Ped. 1906, Univ. of Colorado, S. B. 1911, Univ. of Cal., M. S. V. 1913–14, student, winter semester, Univ. of Berlin (Germany). 1914, summer semester, Univ. of Gottingen (Germany). 1914–15, Grad., School year, Univ. of Princeton. 1907–09, Chemist, Paraffine Paint Co., San Francisco. 1909–11, Asst. Dept. Chemistry, Univ. of Cal., Berkeley, Cal. 1911, Chemist, Anaconda Smelter Commission, Anaconda and Great Falls, Mont. 1912–13, Asst. Physical Chemist, U. S. Bureau of Mines, San Francisco, Cal. 1913–15, Met., Anaconda Smelter Commission, Anaconda, Mont.
Present position: 1915 to date, Chemist, Western Precipitation Co.

Walter Henry Wellman,
Proposed by W. M. Claypool, S. J. Jennings, F. F. Colcord.
Born 1873, Cincinnati, Ohio. High School Training. Accounting and Metallurgical and Mining Accounting. Cashier of Needles Mining & Smelting Co.
Present position: Supt., Needles Mining & Smelting Co.

Max Albert Whiting, Schenectady, N. Y.
Proposed by David B. Rushmore, W. R. Whitney, K. A. Pauly.
Born 1884, Watertown, Wis. Prepared for college in high school at Watertown, Wis. 1904, Univ. of Wisconsin, B. S. in Electrical Engrg. 1905–06, Testing Dept., General Electric Co. Since 1906, in Power and Mining Engineering Dept., General Electric Co. 1906–07, Research on lightning protection, General Electric Co. 1907–12, Estimates, investigations, designs, tests on installations and general consulting on electrical applications, principally in steel plants and ore docks. 1912 to present, consulting work as above, also similar consulting on mine hoisting and other electrical applications in mining. 1914–15, Designed and put in operation the automatic hoists for Inspiration Cons. Copper Co. 1915–16, Published paper, American Institute of Mining Engineers.
Present position: Working as above under general direction of D. B. Rushmore and K. A. Pauly. Electrical Engr., Power and Min. Engrg. Dept., General Electric Co.

George Kay Williams, Anaconda, Mont.
Proposed by Louis V. Bender, Harry S. Ware, C. R. Kuzell.
Born 1872, London, England. 1893, Finsbury Technical, London, England. 1894, Anaconda Copper Min. Co. 1898, Butte Reduction Works. 1899, Montana Ore Purchasing Co. 1901, North West Smelt. & Ref. Co. 1903, British Columbia Copper Co. 1907, Cons. Practice, Vancouver, B. C.
Present position: 1914 to date, Supt. Construction, Anaconda Copper Min. Co.

George Leslie Williams, Hanover, N. Mex.
Proposed by Arthur Thacher, J. A. Van Mater, Karl A. Strand.
Born 1890, Sardinia, N. Y. Grammar and High School. 1914, Univ. of Michigan, Ann Arbor, Mich., B. M. E. 1914–15, Testing and erecting Diesel Oil Engines, Gas Engines and Pumps, The International Steam Pump Co.; Snow Steam Pump Plant, Buffalo, N. Y.
Present position: Mech. Engr., The Empire Zinc Co. in the Southwest, in charge of power plants and equipment.

Kenneth Williams, Douglas, Ariz.
Proposed by Forest Rutherford, P. P. Butler, G. H. Dowell.
Born 1883, Corry, Pa. 1903–07, Univ. of Pennsylvania, B. Sc. in Chemistry. 1907–08, Chemistry Instructor, Univ. of Utah. 1908–09, Chemist, Tintic Smelting Co. 1909–12, Chemistry Instructor, Univ. of Utah. 1912–15, Chemist, Morococha Mining Co., Morococha, Peru. 1915–16, Blast Furnace Foreman, Cerro de Pasco Copper Co., La Fundicion, Peru.
Present position: Shift Supt., Copper Queen Smelting Co.

E. C. Wood, Philadelphia, Pa.
Proposed by Louis D. Huntoon, Benjamin B. Lawrence, H. F. Lefevre.
Born 1888. Public Schools, Kansas City, Mo. 1909, Sheffield Scientific School, Yale Univ., Mining course. 1909–10, Miner and Foreman, Ray Northern Copper Co., Ray, Ariz. 1910–11, Construction, Bingham and Garfield R. R., Bingham Canyon, Utah. 1911–12, Miner, Millman, Asst. Engr., Bingham-New Haven Min.

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Co., Bingham Canyon, Utah. 1913–14, Engr. on examination and construction, Chile Copper Co., Chuquicamata, Chile. 1914–15, Engr. and Mine Supt., Costa Rica Union Min. Co., Mivamor, Costa Rica. 1915–16, Asst. Engr., construction, E. I. duPont de Nemours Powder Co., Parlin, N. J.

Present position: Asst. to Mr. Louis D. Huntoon on mine examination work.

Don Carroll Woodward, Chile, So. Amer.
Proposed by Mark R. Lamb, W. R. Ingalls, Richard H. Vail.
Born 1884, Kenton, O. 1908, Grad., So. Dakota School of Mines.
Present position: Mgr., Sociedad de Minas de Cobre de San Bartolo.

Orel Edward Young, Salt Lake City, Utah.
Proposed by Oliver C. Ralston, Robert S. Lewis, C. A. Wright.
Born 1895, Clinton, Mich. 1902–07, Grade Schools, Clinton, Mich. 1907–11, High School, Clinton, Mich. 1912–16, Case School of Applied Science, Cleveland, O., B. S.
Present position: Research Fellow, Univ. of Utah, Dept. Met. Research.

Associate Members

John R. Gunn, West Plains, Mo.
Proposed by Edward M. Shepard, E. Fleming L'Engle, G. B. Corless.
Born 1863, Morgan Co., Mo.
Been in active charge of properties of the Southwestern Zinc Co., in Howell and Ozark Counties, Mo., since beginning of operations in October, 1915, and in which capacity I am now serving.

Willis Wallace Jourdin, Miami, Ariz.
Proposed by L. O. Howard, M. F. Mazany, W. F. Geiger.
Born 1880, San Antonio, Tex. 1888–96, Public Schools, San Antonio, Tex. 1900–02, Correspondence course, International Correspondence Schools, Mech. Dept. 1904–08, Stanford Univ., A. B. in Mech. Engrg. 1896–98, Machinist apprentice, F. F. Collins Mfg. Co. 1898–1900, Machinist apprentice, M. I. R. R. 1900–03, Machinist, general foreman, Amer. Smelt. & Ref. Co., Monterey, Mex. 1903–04, Master Mechanic, Penoles Min. Co. 1908–10, Supt. of Shops, Darbyshire-Haroic Iron and Mach. Co., El Paso, Tex. 1911, Sales Engr., Mine and Smelter Supply Co. 1912–15, Chief Engr., Power Dept., Chino Copper Co.
Present position: 1915 to date, Chief Engr., Power Plant, Inspiration Cons. Copper Co.

Russell W. Keller, Mogollon, New Mexico.
Proposed by S. J. Kidder, Thomas T. Read, William H. Shearman.
Born 1894, Los Angeles, Cal. 1914, one half year at College in Los Angeles. Started a course on geology but I was forced to discontinue on account of financial reverses. 1915, Times Mirror Co., Los Angeles, Cal.
Present position: Mine Sampler, Mogollon Mines Co.

J. Howard Payne, New York, N. Y.
Proposed by L. D. Ricketts, Burt A. Robinson, Gerard B. Rosenblatt.
Born 1883, Titusville, Pa. 1901, High School, Fostoria, Ohio. 1905, Armour Institute of Technology, Chicago, Ill., B. Sc., in Electrical Engineering. 1905 to date, I have done a variety of engineering work for the Westinghouse Electric & Mfg. Co., having been continuously with them since leaving college.
Present position: Sales Engineer, Westinghouse Electric & Mfg. Co., in charge of all mining work handled through the New York office.

Charles Hammond Shaw, Lawton, Okla.
Proposed by W. H. Kobbé, H. A. Wheeler, W. E. McCourt.
Born 1877, Near Union City, Tenn. 1900, Degree B. L., University of Tex. Practiced law seven years. 1898–99, Two years' study of geology. Took no degrees. 1909–12, Mined Rock Asphalt in Oklahoma, made chemical tests of same for street paving. Have always been in business for myself.
Present position: Am drilling with associates an oil well in Cotton County, Okla.

Numan Reid Stansel, El Paso, Tex.
Proposed by John C. Greenway, A. G. McGregor, R. W. Kerns.
Born 1877, North Carolina. 1898, A. & M. A., N. C. College, B. Sc. 1903,

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Cornell Univ., M. M. E. 1902–05, U. S. Navy Yard, Norfolk, Va., Electrician Y. & D. Dept. 1905–10, U. S. Treasury Dept., Washington, D. C., Engineering Inspector, Supervising Architect's Office. 1910–13, General Electric Co., Schenectady, N. Y.

Present position: 1913 to date, Local Manager Southwest General Electric Co.

Junior Members

Sylvain Sleyman Abouchar, New York, N. Y.

Proposed by Robert Peele, R. M. Raymond, Charles P. Berkey.

Born 1892, Canta. 1904, Certificat d'Etudes Primaires, Ecole Communale de la Ville de Liège, Belgium. 1911, B. Sc. Athenée Royal de Liège, Belgium. 1913, Degree, "Candidat Ingénieur," School of Mines, Royal Univ. of Liège. 1913–14, Student in 3d yr. Univ. of Liège. 1915–16, Junior Student, School of Mines, Columbia Univ., New York.

Present position: Senior Student, School of Mines, Columbia Univ.

Martin F. Bowles, Rolla, Mo.

Proposed by Charles Y. Clayton, Horace T. Mann, C. R. Forbes.

Born 1893, Bonne Terre, Mo. 1912, Grad., Neodesha High School, Neodesha, Kans. 1908–16, summer vacations, Neodesha Plant, Granby Min. & Smelt. Co. (Zinc Smelter), Neodesha, Kans.

Present position: Student, Missouri School of Mines.

Henry S. Burnholz, New York, N. Y.

Proposed by Robert Peele, H. L. Carr, R. M. Raymond.

Born 1894, New York, N. Y. Public Elementary School. Self prepared for entrance to Columbia Univ.

Present position: Student, Columbia Univ.

Thornton Davis, New York, N. Y.

Proposed by R. M. Raymond, Homer L. Carr, William Campbell.

Born 1892, Winthrop, Mass. 1911, Noble and Greenough's School, Boston, Mass. 1911–15, Harvard College, Cambridge, Mass., S. B. 1915–16, On the Wyoming Division Engineering Corps, Lehigh Valley Coal Co., Wilkes-Barre, Pa.

Present position: Student, Columbia School of Mines.

William Myron Davy, New York, N. Y.

Proposed by Robert Peele, E. J. Hall, Charles P. Berkey.

Born 1894, Washington, D. C. 1900–08, Public and private school education in Washington, D. C., Baltimore and Atlantic City. 1908–12, Classical and Scientific diploma from Atlantic City High School. 1916, Princeton Univ., Degree of B. Sc. in Geology.

Present position: Student, Columbia School of Mines, class of 1918.

Alan Edmond Dynan, Bethlehem, Pa.

Proposed by Howard Eckfeldt, Benjamin L. Miller, Joseph W. Richards.

Born 1896, Bethlehem, Pa. 1904–13, Student at Bethlehem Preparatory School, graduating 1913. 1913, Summer, with Bethlehem Steel Co., Open Hearth Dept., as weighmaster. 1914, Summer, with Bethlehem Steel Co., as weighmaster Open Hearth Dept. No. 2. 1915, Summer, with Bethlehem Steel Co., Projectile Dept.

Present position: Student at Lehigh University.

Frank Stillman Elfred, Jr., Rolla, Mo.

Proposed by Charles Y. Clayton, Austin L. McRae, Horace T. Mann.

Born 1894, Denver, Colo. 1908–10, St. Joseph Central High School. 1910–11, St. Mary's College, Kans. 1911–12, Grad., St. Joseph Central High School. 1912–13, Irrigation in Grand Valley, Colo. 1916, Research Chemist and general work at E. I. & F. C. Wallower R. and R. works, Joplin. Mo.

Present position: 1913 to date, Missouri School of Mines.

Joseph Austin Holmes, 2nd, So. Bethlehem, Pa.

Proposed by Howard Eckfeldt, Joseph W. Richards, Benjamin L. Miller.

Born 1895, Chapel Hill, N. C. 1900–02, Private School, Chapel Hill, N. C. 1902–07, Public Schools, St. Louis, Mo. 1907–10, Public Schools, Washington, D. C.

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1910–14, Western High School, Washington, D. C. 1914–15, Univ. of North Carolina. 1913, Field Asst., U. S. Navy Alaskan Coal and investigations expedition. 1914, Field Asst., U. S. Bureau of Mines. 1915, Field work, B. F. Hall, Banners Elk, N. C. 1916, Asst. Surveyor, Boone Fork Lumber Co., Shulls Mills, N. C.
Present position: 1915 to date, Student, Lehigh Univ.

Wendell Theodore Jones, Reno, Nev.
Proposed by J. C. Jones, Francis C. Lincoln, J. G. Scrugham.
Born 1892, Grand Junction, Colo. With Tonopah Belmont Development Co., during the summer months for the past six years.
Present position: Attending Mackay School of Mines, Reno, Nev.

Walter Peter Klugescheid, New York, N. Y.
Proposed by R. M. Raymond, Homer L. Carr, William Campbell.
Born 1894, New York, N. Y.
Present position: 1912 to date, Student, Columbia School of Mines.

Edgar Kraus, New York, N. Y.
Proposed by Robert Peele, R. M. Raymond, Homer L. Carr.
Born 1896, New York, N. Y. 1901–09, P. S. 170, 184, 186. 1907–13, Stuyvesant High School, New York. 1916, Underground laborer, Champion Copper Co., Painesdale, Mich.
Present position: 1913 to date, Student, Columbia School of Mines.

Harold Arthur Lask, Rolla, Mo.
Proposed by C. R. Forbes, H. A. Buehler, H. T. Mann.
Born 1893, Pasadena, Cal. 1901–08, Garfield Grammar School, Pasadena. 1908–12, Pasadena High School. 1914–15, Univ. of Arizona. 1915, summer, Miami Copper Co., Miami, Ariz. 1916, summer, St. Joseph Lead Co., Bonne Terre, Mo.
Present position: Student, Missouri School of Mines.

Henry Alfred Ley, Aspinwall, Pittsburgh, Pa.
Proposed by Stephen L. Goodale, H. C. Ray, H. B. Meller.
Born 1894, Pittsburgh, Pa. 1901–09, Public Schools, Pittsburgh, Pa. 1909–13, Aspinwall High School, Aspinwall. 1913, summer, Chemical Laboratories, Carnegie Steel Co., Pittsburgh, Pa. 1915, summer, Laborer, Miner and Drill-man, Pioneer Mine, Oliver Iron Min. Co., Ely, Minn. 1916, summer, Geol., Southwestern Utilities Corpn., Independence, Kans.
Present position: 1913 to date, Student, Univ. of Pittsburgh and Geol., Southwestern Utilities Corpn.

Theodore I. Linn, So. Bethlehem, Pa.
Proposed by Howard Eckfeldt, Benjamin L. Miller, Joseph W. Richards.
Born 1892, Tsinan, China. 1906, Peking Preparatory School. 1910, Peking Univ., Peking, China, B. A.
Present position: Student, Lehigh Univ.

William Smith Livingston, Jr., New York, N. Y.
Proposed by E. J. Hall, R. M. Raymond, Robert Peele.
Born 1888, New York, N. Y. Early schooling received at Craigie School and Cutler School, New York City.
Present position: Student, Columbia University.

Maurice Cecil Lucky, Jr., Rolla, Mo.
Proposed by Charles Y. Clayton, H. T. Mann, C. R. Forbes.
Born 1895, Exeter, Mo. 1909–11, High School, Cassville, Mo. 1911–12, High School, Dallas, Tex. 1913–14, Texas A. and M. College 1916, summer, Chino Copper Co., Hurley, N. Mex.
Present position: 1914 to date, Student, Missouri School of Mines.

Robert P. Lyons, Rolla, Mo.
Proposed by C. R. Forbes, A. L. McRae, H. A. Buehler.
Born 1896, Springfield, Mo. 1908–12, Conception College, Conception, Mo. 1914, Mining, Goldfield Cons. Mines, Goldfield, Nev. 1916, Milling, Butte and Superior Min. Co., Butte, Mont.
Present position: Student, Missouri School of Mines.

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Sydney See Ho Ng, New York, N. Y.
Proposed by Charles P. Berkey, E. J. Hall, Robert Peele.
Born 1896, Sydney, N. S. W. 1913–16, Mining Student, Royal School of Mines, England.
Present position: Mining Student, Columbia Univ.

James W. Rice, Reno, Nev.
Proposed by Francis Church Lincoln, J. G. Scrugham, J. C. Jones.
Born 1893, So. Omaha, Nebr. 1912–13, Mackay School of Mines. 1911–12, Sampler, Giroux Cons. Min. Co., Kimberly, Nev. 1913, summer, Millman, Utah Cons. Min. Co., Garfield, Utah. 1913, winter, Millman, Boston Cons. Copper Co., Lark, Utah. 1914–15, summers, Millman, Nevada Cons. Copper Co., McGill, Nev. 1916, summer, Prospecting, Pend O'rielle Co., Wash.
Present position: 1914 to date, Student and Asst. Instructor in Chemistry, Mackay School of Mines, Univ. of Nevada.

Donald D. Riddle, Golden, Colo.
Proposed by H. J. Wolf, W. G. Haldane, G. J. Young.
Born 1894, Crawfordsville, Ind. 1900–08, Grade schools of Indianapolis, Ind. 1908–13, Manual Training High School, Indianapolis. Summer 1912, Drafting room Engineering Dept., Central Union Telephone Co., Indianapolis. Summer 1913, Drafting, City Civil Engr., City of Indianapolis. 1913–14, Central Engineering Dept., Fundamental plans, Bell Telephone Co., Chicago, Ill.
Present position: Student, Colorado School of Mines.

John M. Schloss, New York, N. Y.
Proposed by Robert Peele, Homer L. Carr, R. M. Raymond.
Born 1894, New York, N. Y. Public School No. 87, New York, N. Y. 1906–10, Stevens High School, Hoboken, N. J. 1912–14, School of Mines, Columbia Univ. 1910–12, Draftsman and outside work, Fleischman Bros. Co. Building construction. 1914–15, worked underground as miner for one year to gain practical experience, Crown Reserve Min. Co., Cobalt, Ont., Canada.
Present position: Senior, School of Mines, Columbia Univ.

Fred Pine Shayes, Rolla, Mo.
Proposed by C. R. Forbes, H. T. Mann, C. Y. Clayton.
Born 1894, Chittenango, N. Y. 1909–12, East Rochester High School, Graduate. 1912–13, Rochester Business Institute. 1913–16, Student Missouri School of Mines. 1913, summer, Underground work Daily West Mining Co., Park City, Utah. Summer 1916, Surveyor's helper, Burro Mountain Copper Co., Tyrone, New Mex.
Present position: Student, Missouri School of Mines.

Harold Jandorf Sloman, So. Bethlehem, Pa.
Proposed by Howard Eckfeldt, Benjamin L. Miller, Joseph W. Richards.
Born 1896, Baltimore, Md. 1910–14, Baltimore Polytechnic Institute. 1915, Bethlehem Steel Co., So. Bethlehem, Pa. 1916, summer, Survey, Lehigh Valley Coal Co., Hazleton, Pa. Survey and inspection, Western Maryland Div., Consolidation Coal Co. Inspection, Georges Creek Coal Co.
Present position: 1914 to date, Student, Lehigh Univ.

William B. Sommerville, Jr.,
Proposed by William Campbell, R. M. Raymond, Robert Peele.
Born 1893, Birmingham, Ala. 1901–12, Horace Mann School. 1912–13, Williams College.
Present position: 1913 to date, Student School of Mines & Metallurgy, Columbia University.

Howard Willey, New York, N. Y.
Proposed by R. M. Raymond, Homer L. Carr, Robert Peele.
Born 1894, Mt. Upton, N. Y. 1902–10, Mt. Upton Union School. 1910–12, Norwich High School. 1912–16, School of Mines, Columbia University. 1916, Champion Copper Mine, Painesdale, Mich.
Present position: Student, Columbia University.

Guy K. Young, Golden, Colo.
Proposed by W. G. Haldane, Harry J. Wolf, George J. Young.
Born 1893, Spokane, Wash. 1908–10, Spokane High School. 1912–13, Valparaiso Univ. 1916, Interstate-Callahan Cons., Wallace, Idaho.
Present position: 1914 to date, Student, Colorado School of Mines.

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CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Oct. 10, 1916 to Nov. 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 119, February to November, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of Nov. 10, 1916.

AARONS, J. BOYD, Major, Australian Infantry, 74 Upper Gloucester Pl., Dorset Sq., London, W., England.
ADAMS, JOHN H.....512 8th Ave., West Graymont, Birmingham, Ala.
AHLERS, RUDOLPH O.....Rua do Cospo Tanto, 13, Lisbon, Portugal.
ALLEN, ARTHUR P.....Highland Valley Min. & Dev. Co., Ashcroft, B. C.
ALLEN, MILTON A.....Arthur L. Pearse & Co., 52 Broadway, New York, N. Y.
ANDERSON, CARL N.....920 Woodward Ave., Portland, Ore.
ANDERSON, RAY B.....Wasatch Ozokerite Co., Soldier Summit, Utah.
ARKELL, HARRY J.....'The Hobby,' Marden, Kent, England.
BABB, PERCY A., Blaisdall Coscotitlan Synd., Apartado 92, Pachuca, Hidalgo, Mexico.
BARROWS, WALTER A., JR.....Hokandauqua, Pa.
BASSETT, ARTHUR F.....522 Seventh St., Huntington, West Va.
BLOOMFIELD, EDWIN C., Lieut., Royal Canadian Engineers, Engineer Training Depot, St. John, Prov. of Quebec, Canada.
BOLGIANO, J. RALPH..... Efficiency Dept., Malleable Iron Co., Dayton, Ohio.
BORDWELL, EDWARD M.....Aurora Cons. Min. Co., Aurora, Nev.
BOYD, DANIEL.....348 Albany Ave., Toronto, Ont., Canada.
BROWN, FREDERICK C., Min. Engr.....R. F. D. 4, Boise, Idaho.
BRYDEN, CHARLES LAZARUS.....Room 303 E., 30 Church St., New York, N. Y.
BULL, ROBERT ALEXANDER..... 2237 E Street, Granite City, Ill.
BUTLER, MEL C.....4750 17th Ave., N. E., Seattle, Wash.
CAIRNS, JOHN M, Baluchistan Mining Syndicate, Ltd., Quetta, Baluchistan, India.
CALDWELL, FOREST B.....Woodland, Cal.
CALVERT, WILLIAM R., Cons. Geol. and Engr., 225 Kearns Bldg., Salt Lake City, Utah.
CARUTHERS, A. WAYNE,.....Coal Dept., D. L. & W. RR. Co., Scranton, Pa.
CHARLES, LAVERN JOHN.....Const. Engr., U. S. R. S. Lakewood, New Mexico.
CHASE, ROGER E.....3005 N. Proctor, St., Tacoma, Wash.
COGGERSHALL, GEORGE W.....2229 California St., Washington, D. C.
COLLEY, HYLTON H.....Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
COLLINS, HENRY F.....66 Finsbury Pavement, London, E. C., England.
CORBIN, J. ROSS NOEL.....Boyertown, Pa.
CORWIN, FRANK R., Asst. Supt., Smelter, International Smelting Co., Miami, Ariz.
CRAIG, JOHN J.....P. O. Box 854, Morenci, Ariz.
CROSS, C. GARCIA.....Casilla No. 2034, Santiago, Chile, So. America.
CROSTON, JOHN J.....Box 64, Atlanta, Ga.
CUNNINGHAM, NOEL, Met. Engr.,.....Room 1301, 90 West St., New York, N. Y.
DAULTON, THEODORE M.....208 New York Bldg., Seattle, Wash.
DAVID, JOSEPH.....110 Eighth St., Calumet, Mich.
DAVIES, FRED A.....P. O. Box 837, Anaconda, Mont.
DEKALB, COURTENAY.....Instructed to hold everything.
DESHLER, GEORGE O.....Supt., Mines Operating Co., Butte, Mont.
DONDERO, F. NICHOLAS.....Loinghouse, Nev.
DOUGLAS, THEODORE.....36 Flatbush Avenue Extension, Brooklyn, N. Y.
DRAKE, MORRIS CLAIRE.....342 E. Hewitt Ave., Marquette, Mich.
DUNBAR, DOUGLAS MAC D..... Federal Lead Co., Flat River, Mo.
DWYER, C. EUSTACE, International Smelt. & Ref. Co., McIntyre Bldg., Salt Lake City, Utah.
ELTON, JAMES O.....607 5th Ave. N., Great Falls, Mont.
FARNHAM, C. MASON.....5 Sumner Road, Cambridge, Mass.
FLETCHER, ARTHUR R., Sociedad de Minas y Fundiciones de Carrizal, Carrizal Bajo, Chile, South America.
FLYNN, FRANCIS N....Chile Exploration Co., Chuquicamata, Chile, South America.
FRASER, DONALD DICKINSON, Highland Valley Min. & Dev. Co., Ashcroft, B. C., Can.
FRASER, EVERETT L.....Asst. Supt., Oro Grande Mines, Callahan, Cal.
FULTON, CHESTER A.....Obispo 6½, Havana, Cuba.

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GABY, WALTER E.....Davis-Daly Copper Co., Butte, Mont.
GOLD, CHARLES BARLIAND.....5008a Cates Ave., St. Louis, Mo.
GRAHAM, HERBERT W.....219 Halket St., Pittsburgh, Pa.
GREENSFELDER, NELSON S., Asst. Prof. Chemistry, Colorado College, Colorado Springs, Colo.
HAGEMANN, WILHELM, Cia. Aluminio y Corcho Corona, 2a Sta. Teresa No. 47, Mexico City, Mexico.
HERR, L. B. JR.....511 Seneca St., So. Bethlehem, Pa.
HIGGINS, THOMAS F., Broadwell House, McKean Road, Oldbury, Worcestershire, England.
HINCKLEY, ELMER R.....809 S. Montezuma Ave., Phoenix, Ariz.
HOLDERER, GEORGE B.....General Chemical Co., 25 Broad St., New York, N. Y.
HOLLISTER, SCOVILLE E., Butte & Superior Copper Co., P. O. Box 197, Butte, Mont.
HOLME, PERCY E.....Apartado 6, El Oro, Mex., Mexico.
HOOVER, H. C.....3 London Wall Buildings, London, E. C., England.
HUNTER, CHARLES, Mine Mgr., Royal Colonial Institute, Northumberland Ave., London, W., England.
HUTCHINSON, RANDOLPH B.....1008 Central Bldg., Los Angeles, Cal.
IMHOFF, ALEXANDER.....Box 1548, Miami, Ariz.
JARMAN, ARTHUR.....13 Tunstall Vale, Sunderland, England.
JARVIS, ROYAL P.....United Verde Extension Min. Co., Jerome, Ariz.
KEENEY, ROBERT M.....1418 Elizabeth St., Denver, Colo.
KELLOGG, RALPH MURRAY.....Bullion, Nevada.
KENNEY, JOHN R.....2210 Morse Ave., Chicago, Ill.
KEPNER, ROSS B.....1705 South Union Ave., Los Angeles, Cal.
KINNEY, HARRY D.....Mines Operating Co., Butte, Mont.
KOHL, E. WILLIAM, JR., Cons. Engr. and Contractor, Apartado 333, Santiago de Cuba, Cuba.
KROGH, ALVIN T.....P. O. Box 837, Anaconda, Mont.
LATHAM, EVERETT B.....Engineers' Club, 57 Post St., San Francisco, Cal.
LAURIE, FRANK C., Min. Engr.....331 Norton St., New Haven, Conn.
LAVERY, VAUGHAN M., Lieut., 3d Tunnelling Co., Canadian Engineers, B. E. F., France.
MCINTOSH, FRANK K.....Box 236, San Francisco, Cal.
MAC ARTHUR, JOHN S.....Mina da Serra da Caveira, Grandola, Portugal.
MATHEWSON, EDWARD P., Met., U. S. Smelt. Co., Inc., Room 3260, 120 Broadway, New York, N. Y.
MATTHEWS, CHARLES H.....406 East Arch St., Marquette, Mich.
MAY, DECOURCY.....Instructed to hold everything
MEANS, ALAN H.....Instructed to hold everything.
MENEFEE, ARTHUR B., Min. Engr.....Puritan Apts., Louisville, Ky.
MILDON, REGINALD B.....Westinghouse Machine Co., East Pittsburgh, Pa.
MILLER, HARRY H.....2757 West 8th St., Los Angeles, Cal.
MILWARD, MILES S.....Mocorito, Sinaloa, Mexico.
MILLWARD, WILLIAM.....818 House Bldg., Pittsburgh, Pa.
MOHRMAN, EDWIN M., The Cott-Mohrman Co., 1116 Citizens Bldg., Cleveland, Ohio.
MOORE, CARL F.....Room 400, 55 Congress St., Boston, Mass.
MORGAN, HARRY J.....Big Indian Co., La Sal, Utah.
MORRISON, GEORGE A.....P. O. Box 1425, Sudbury, Ont., Canada.
MOTHERWELL, WILLIAM, Cons. Engr., on Flotation Concentration, General Delivery, Nelson, B. C., Canada.
MOULE, JOHN W., Genl. Mgr., Corella Copper Co., via Ballara, Cloncurry, North Queensland, Aust.
PATTERSON, SEELY B., JR., Asst. Genl. Mgr., The Spanish American Iron Co., Felton, Cuba.
PAYMAL, GEORGE W.....435 W. 119th St., New York, N. Y.
PEARSON, WILLIAM R.....54–57 Morningside Drive, New York, N. Y.
PHILLIPS, DRURY MCNEILL.....104 Lakeshore Apartments, Port Arthur, Tex.
PRIOR, CHARLES E., Engr.....La Blanca y Anexas Mina, Pachuca, Hidalgo, Mex.
RHODES, CLARENCE E.....Apartado 30, El Oro, Mex., Mexico.
RICHARDSON, CHARLES.....94 St. George St., Toronto, Ont., Canada.
RISSMANN, OTTO.....3615 Central Ave., Kansas City, Mo.
ROBERTSON, JASPER T.....Montana Block, Butte, Mont.
ROGERS, BENJAMIN CARROLL.....P. O. Box 697, Idaho Springs, Colo.
RUGGLES, GUY H.....Inspiration Cons. Copper Co., Miami, Ariz.

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SALE, ANDREW J., Assayer and Chemist, Battle Mountain Mines and Dev. Co., Battle Mountain, Nev.
SCHLEICHER, HENRY M.....10 Williams St., Roxbury, Mass.
SCOBEY, JESSE C., Min. Engr., Asst. to the Pres., La Luz and Los Angeles Min. Co., Rooms 1003–4–5, Postal Life Ins. Bldg., 511 Fifth Avenue, New York, N. Y.
SHARP, JOHN R., Genl. Mgr.....Hailey-Ola Coal Co., Haileyville, Okla.
SHIPP, ELLSWORTH M., Min. Engr.....120 Broadway, New York, N. Y.
SMITH, ELWYN L.....Ray Cons. Copper Co., Ray, Ariz.
SMITH, LYON.....1547 Beacon St., Brookline, Mass.
STEWART, JOHN BURGOYNE.....Trussville, Ala.
THOMPSON, ARTHUR P.....P. O. Box 224, Jerome, Ariz.
TIBBY, BENJAMIN F., Mine Statistician, U. S. Bureau of Mines, Washington, D. C.
TOENGES, ALBERT L.....252 High St., Dayton, Ohio.
VALENTINE, MALVERN R., Min. Engr..The Portland Gold Mining Co., Victor, Colo.
WAITE, ALLAN G.....Vice-Pres., Dragoon Tungsten Min. Co., Dragoon, Ariz.
WENTWORTH, HENRY A., Technical Director, American Zinc, Lead and Smelt. Co., 55 Congress St., Boston, Mass.
WHITFELD, HUBERT E.....1510 Union Bank Bldg., Pittsburgh, Pa.
WICKES, L. WEBSTER.....Kingman, Ariz.
WOO, WAI KYI.....140 Range Road, Shanghai, China.
WROTH, JAMES S.....Casilla 49-D, Santiago, Chile, South America.
WYSOR, RUFUS JOHNSTON., Supt., Blast Furnaces, Bethlehem Steel Co., So. Bethlehem, Pa.
YAMASHITA, M.....Min. Dept., Mitsubishi Co., Marunouchi, Tokyo, Japan.
YOUNG, GEORGE J., Prof. of Met.....Colorado School of Mines, Golden, Colo.
ZENTNER, ARTHUR A., Zinc Roasting Dept., Boston and Montana Smelter, Great Falls, Mont.
ZIMMERMAN, ADOLPH A.....P. O. Box 1385, Helena, Mont.

MEMBER'S ADDRESSES WANTED

Name.	Last Address of Record from which Mail has been returned.
BAUMANN, A. P.	939 Boulevard East, Weehawken, N. J.
COMINGS, GEORGE R.	Apartado 126, Oaxaca, Oax., Mexico.
DIETRICH, WALDEMAR F.	Humboldt, Ariz.
EVANS, DAVID J.	Cons. Arizona Smelting Co., Humboldt, Ariz.
GORMAN, THOMAS CLARENCE	Creighton Mine, via Sudbury, Ont., Canada.
HAGEMANN, WILHELM	2a San Augustin #53, Mexico City, Mex.
MCKANNA, EDWIN A.	Savanna, Okla.
MACAULAY, R. M.	Grande Allee St., Quebec City, Canada.
MURPHY, FRANCIS JOSEPH	Great Cobar, Ltd., Cobar, N. S. W., Australia.
PITTMAN, FRANK L.	Contact, Nev.
STROUD, BENJAMIN K.	333 Pacific Ave., Oakland, Cal.
ROGERS, GEORGE R.	47 St. Vincent St., Toronto, Ont., Canada.
TAYLOR, CHARLES H.	University of Oklahoma, Norman, Okla.
WELLS, RALPH EVANS, JR.	Nogales, Ariz.
WILMOT, H. CLIFFORD	Santiago, Chile, South America.
WROTH, JAMES S.	Casilla 46-D., Santiago, Chile, South America.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Oct. 10, 1916, to Nov. 10, 1916.

Date of Election.	Name.	Date of Decease.
1906	* Bevington, Richard George	——, 1916.
1879	* Ely, Theodore N.	Oct. 28, 1916.
1889	* Hare, A. W.	Oct. —, 1916.
1893	* Pollok, James H.	— —, 1916.
1900	* Revell, George Ernest	— —, 1916.
1916	* de Ryck van der Gracht, C. M. A.	July 26, 1916.

* Member.

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BIOGRAPHICAL NOTICE OF ECKLEY B. COXE, JR.

By HENRY S. DRINKER

The Institute has lost a valued member by the death of Eckley B. Coxe, Jr. He bore the name of one who was largely instrumental in founding the Institute, and to whose support and guidance as active member, Vice-President, and President, the Institute owes much. Eckley B. Coxe, Jr., was the nephew of the late Eckley B. Coxe. He held a large interest in the coal estates owned and developed by his family, and he was thus naturally led to take a continuing interest in mining matters, and for years and up to the time of his death was a full member of the Institute.

He died at his residence, Drifton, Pa., on Sept. 20 last, aged 44. He was a member of the class of '93 of the University of Pennsylvania, of which great institution he was a devoted alumnus. His educational and charitable activities were numerous, but his chief life-work and interest was in archæological investigation and research in Egypt, carried on through the Museum of the University of Pennsylvania of which Museum he was President. The discoveries made by the various expeditions financed and directed by him, and the notable publications issued by his direction and at his expense, chronicling the results, are among the most important researches ever made of buried Egypt.

Mr. Coxe was a member of the International Historical Society and of many educational and charitable organizations. By his will, among other bequests, he left $500,000 to the Museum of the University of Pennsylvania and $100,000 to the University, the income of the latter fund to be applied to increasing the salaries of the professors of the Institution. He also left $100,000 to the Children's Hospital of Philadelphia, of which he was a manager, and in addition thereto $10,000 of which the income is to be applied in providing Christmas presents for the child patients, and in providing an annual dinner for the patients and attendants of the hospital. He bequeathed $50,000 to Leonard Hall, an associate mission of the Episcopal Church, at South Bethlehem, Pa., $25,000 to the Epileptic Hospital and Colony Farm (Philadelphia); $50,000 to the Philadelphia Orthopœdic Hospital and Infirmary for Nervous Diseases; $15,000 to the United Charities of Hazleton, Pa., the income to be applied to the salaries of nurses; and $25,000 to the Mining and Mechanical Institute of the Anthracite Coal Region of Pennsylvania, founded by his uncle, Eckley B. Coxe, and of which Eckley B. Coxe, Jr., was an active trustee up to the time of his death. This Institute is known as one of the best preparatory schools for technical education in Pennsylvania, and is mainly devoted to extending an opportunity for education by day and night schools to the boys and young men of the Anthracite Region. Its principal is an alumnus of Lehigh University; it has an able local Board of Trustees, and an Advisory Board consisting of the Provost of the University of Pennsylvania, the President of Lafayette College, and the President of Lehigh University, all three of which institutions have established prize scholarships in the Institute.

Mr. Coxe came of a distinguished ancestry, the record of which is fully given in the biographical notice of his uncle, Eckley B. Coxe, by Dr. R. W. Raymond, in Vol. XXV of the *Transactions*.

A man of charming personality, Mr Coxe was greatly esteemed by a

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wide circle of friends, and his familiar presence at the University Club of Philadelphia will be greatly missed. His frail body sheltered the spirit of a courteous gentleman of high culture and lovable and gentle character. There are many who feel that in his death a man of rare quality and parts has passed away—an American gentleman of the truest and highest type.

GEORGE W. RITER

George W. Riter was born in Salt Lake City, Feb. 22, 1871, the son of Levi E. Riter, a pioneer of Utah and California. His common school education was received in the public schools of his native city.

At the age of 14 he went to work in the joint offices of the Utah and Nevada and Salt Lake and Western Railways, which were later absorbed into the Union Pacific system. In a service of six years' duration, he worked his way up from office boy to that of Chief Clerk, serving in this capacity three succeeding general managers of the Utah Division of the Union Pacific system, W. W. Riter, George Cumming, and J. H. Young.

Desiring to obtain a college education, Mr. Riter left the employment of the railway and entered the University of Utah for two years of college preparatory work. In 1893 he entered Leland Stanford, Jr. University, from which institution in 1896 he received the degree of Bachelor of Arts in the department of Geology and Mining.

On arriving home from Stanford University the position of Secretary and Assistant Manager of the Eureka Hill Mine was tendered him by John Q. Packard, who had become acquainted with his ability during the period of his service on the railroad. In this capacity he served for three years (1899) when he became Secretary and General Manager of the company, which position he retained until 1914 when the mine became inactive as a producer.

In 1905–1906 Mr. Riter served as City Engineer of Salt Lake City, and during his occupancy of this office the engineering problem of making East South Temple Street the present beautiful thoroughfare it is, was planned and executed.

He also had charge of the engineering in the construction of both the water and sewerage systems for the Town of Eureka.

In 1914, he was employed by the Engineering Department of the California Highway Commission as assistant engineer of construction on the contract for the southern branch of the great state highway.

When the metal schedule of the present tariff law was under discussion by Congress in 1913, Mr. Riter was selected by the Lead and Zinc Producers of the Western States to represent their interests at Washington, in the discussion of the revision of import duties on lead and zinc ores. For several years, Mr. Riter served the association as secretary and statistician.

Mr. Riter was an active member of the University Club of Salt Lake City, of the Stanford University Alumni Association and the following professional societies: the Mining and Metallurgical Society of America; the American Society of Municipal Improvements; the American Society for the Advancement of Science, Pacific Division; the Seismological Society of America; the Utah Society of Engineers; the American Institute of Mining Engineers; the American Mining Congress.

In this latter society Mr. Riter was Chairman of the Committee on

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Mineral Statistics, from 1911 to 1915. He was also a member of the committee on Smelter Rates, and chairman of the committee appointed by the association to prepare the report on "The Vertical Side Line Law." At the meeting of the Congress held in Salt Lake City, a paper on "The Mining Industries of Utah" was presented by him.

For the American Institute of Mining Engineers, Mr. Riter served as a member of the Committee on Mining Laws, 1914 to 1916. He became greatly interested in the matter of the revision of laws regarding mineral lands and mining, writing a great many articles on this subject for various publications. He was also a frequent contributor to the various magazines and technical publications of the mining and metallurgical world, writing articles on: "The Elastic Hydro Carbons of Utah;" "The Oil Flotation Process;" "The Making of Mine Survey Notes;" "Mineral Land Laws;" "Character of Title to Mineral Land that should be granted by the Government;" "Cost of Silver-Lead Smelting;" "Selective Flotation."

In addition to his active engineering work, in field as well as office, he spent a great deal of time conducting extensive laboratory experiments for the treatment of various classes of ores, together with experiments into the technology of asphalts and related bitumens.

At the time of his death, Aug. 20, 1916, Mr. Riter was developing a large tungsten property near Lucin, Box Elder County, Utah. He was also consulting engineer for the Emma Copper Mine at Alta.

F. McMILLAN STANTON

F. McMillan Stanton was born in New York City on May 23, 1865, the son of an English father and of a mother whose family was linked with that of Peter Stuyvesant.

Mr. Stanton took his degree from the School of Mines of Columbia University in 1887, worked for two years as an assayer and surveyor and then went into the mining field on his own account. After a few years, he was offered a position with the Atlantic Mining Co., and began the work which was to continue with marked success for twenty-three years.

In 1910, Mr. Stanton retired and went to Europe for a rest. Upon his return in 1914, he was elected treasurer and director of the Mohawk Mining Co., the Wolverine Copper Mining Co. and the Michigan Copper Mining Co., to which group was later added the White Pine Extension Copper Co., and continued to serve these companies until the time of his death, Sept. 12, 1916.

Mr. Stanton was also president and director of the Ft. Mountain Talc Co., a director of the First National Bank of Houghton, Mich., of the Ohio and Kentucky Railroad, and of the Copper Range Co. He was on the membership list of the following organizations: American Society of Civil Engineers, American Chemical Society, American Forestry Association, American Society of Mechanical Engineers, American Mining Congress, Navy League of the United States, American Association for the Advancement of Science, Sons of the Revolution, Lake Superior Mining Institute, National Security League, National Rifle Association, and many others.

He became a member of this Institute in 1889.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Countercurrent Decantation*

BY LUTHER B. EAMES,† E. M., TIMMINS, ONT., CANADA

(New York Meeting, February, 1917)

THE recovery of dissolved gold from slime pulp in the cyanide process was first accomplished by intermittent decantation. This simple process consists in mixing with the pulp containing the values in solution, a solution of lower gold content, settling the mixture in a tank and decanting the clear supernatant fluid. The thick pulp remaining in the tank is pumped to a second tank together with more barren solution and again settled and decanted. After several repetitions of this operation, values are so far reduced that further washing is not profitable. The gold recovery of this process is high, but the plant required is bulky, labor cost is high and the amount of solution to be precipitated is excessive.

HISTORICAL DATA

As early as 1901, a plant was built in the Black Hills of South Dakota by John Randall, employing the same principles but attempting to make the process continuous by substituting for flat-bottomed tanks, cones which operated continuously, receiving a constant feed and discharging a steady stream of thickened pulp. These cones were operated in series, the thick underflow of the first one forming, with a stream of diluting solution, the feed to the second cone of the series. Barren solution was added to the tank immediately preceding the discharge tank and, after being slightly enriched by the low-grade pulp in this tank, overflowed to form a diluting solution again for the richer feed entering the third tank from the end of the series, and so on back to the richest tank of the series. Clear water was used for the wash in the final tank. This is the principle on which all successful countercurrent decantation plants operate at the present time, but Randall's plant was not successful because of mechanical difficulties in getting a continuous thick discharge from his cone tanks. A similar plant was built in South Africa although there the washes were not repeatedly used, as in Randall's case, but were precipitated after each contact with the ore. This also was abandoned

* Also published at this time in the *Bulletin* of the Canadian Mining Institute.

† Mill Superintendent, Hollinger Consolidated Gold Mines, Ltd

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because of mechanical difficulties and the cost of precipitating the large quantities of solution that had to be used. For a number of years the process was not used, and it was not until the introduction of the Dorr thickener that the minds of metallurgists began to turn again to the continuous decantation principle.

In 1910, two decantation plants were built making use of flow sheets similar to that used by Randall nine years before, but substituting Dorr thickeners for the cones. One of these was at Mocorito in Sinaloa, Mexico, and was installed under the direction of C. Dupre Smith, while the other was designed by J. V. N. Dorr, assisted by the writer, for the Vulture Mines Co. of Wickenburg, Ariz. While perhaps not perfect at first, both of these pioneer plants were so successful as to encourage further installations, few and scattering at first but in considerable numbers during the past three years.

Theoretical Considerations

In view of this increasing importance, the following discussion of the principles and characteristics of the process is offered. For the purpose

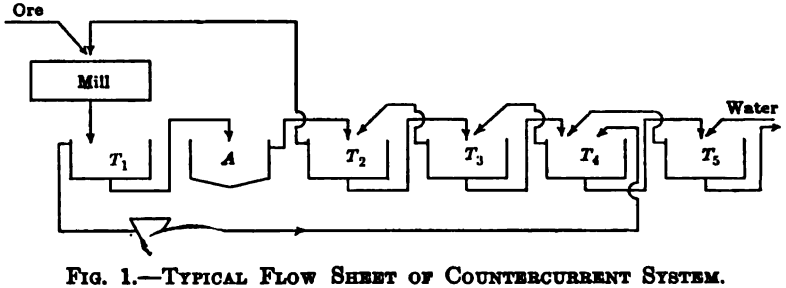

Fig. 1.—Typical Flow Sheet of Countercurrent System.

of investigation a simple yet typical flow sheet has been selected. This is shown in Fig. 1.

This flow sheet assumes that crushing is done in solution, the overflow from the tank T_2 being used for the crushing solution. This crushing solution leaves the grinding circuit with the ground pulp and enters T_1, and that part which does not pass to the agitators with the pulp overflows T_1 and goes to precipitation. After depositing its gold contents, it is used to dilute the underflow of T_3 as it enters T_4. The overflow of T_5 is also mixed into the feed to T_4. The overflow of T_4 mixes with the underflow of T_2 to form the feed to T_3, and so forth, as indicated in the flow sheet. At each succeeding mixture the solution meets a pulp of higher dissolved content than itself and is enriched while the pulp is correspondingly impoverished. The pulp at each step approaches the discharge end of the mill while the solution goes to the feed end—hence countercurrent decantation.

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Variables Affecting Decantation Process

The principal variables that may affect the efficiency of the process are:

1. Grade of ore.
2. Ratio of solution precipitated to ore treated.
3. Thickness at which pulp can be discharged.
4. Cost of chemicals.
5. Rapidity of dissolving, and the place in the circuit where it takes place.
6. Efficiency of precipitation.

Since the decantation process is one involving volumes and dilutions, it is possible to calculate accurately what distribution of values should take place under any given set of conditions. As far as possible, each one of the above variables has been mathematically considered independently of the rest and the results have been plotted.

The effect of variations in the grade of ore is shown in Fig. 2 and scarcely needs comment. In practice, of course, no such increase as is shown in the gold-solution curve would ever be allowed to take place on account of the difficulty of precipitating such high-grade solution and the danger of leakage, but the graph shows conclusively that all solutions increase in value in direct proportion to the increase in the grade of the ore and that the higher-grade solutions increase at a much more rapid rate than the final washes.

The important part played by the ratio of solution precipitated to ore milled is shown in Fig. 3. For the particular grade of ore considered, which in this case was $10 recoverable per ton, with precipitation to 3 c., the economic ratio may be roughly determined by inspection of the lowest curve, which represents the value of the solution leaving the last tank of the series with the tailing. It will be noted that the loss in gold increases very fast as the amount of solution precipitated is decreased, while after a certain point the increased recovery due to increasing the volume precipitated is very slight.

The final choice of the exact ratio to be used must be influenced by the cost of precipitation, which is mainly the cost of zinc. To show clearly the effect of precipitation cost on the economic precipitation ratio, Fig. 4 was plotted. I have considered it safe to assume that the cost of increasing the amount precipitated is due to the additional zinc used. The loss in dissolved gold and the cost of the zinc used in precipitation are both plotted to the same scale in cents per ton of ore. By adding the ordinates of these two curves a third is formed which represents the total loss in gold and zinc. The lowest part of this curve represents the economical range and indicates that for a $10 ore, under the conditions assumed with zinc at 26 c., from 200 to 250 tons should be precipitated

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per 100 tons of ore treated. In dotted lines are shown the corresponding curves for a $5 ore under similar conditions. In this case the range is from 150 to 200 tons. While there is a considerable range within which practically identical results may be expected, it is apparent that each operator should figure out for his own conditions just what his range is.

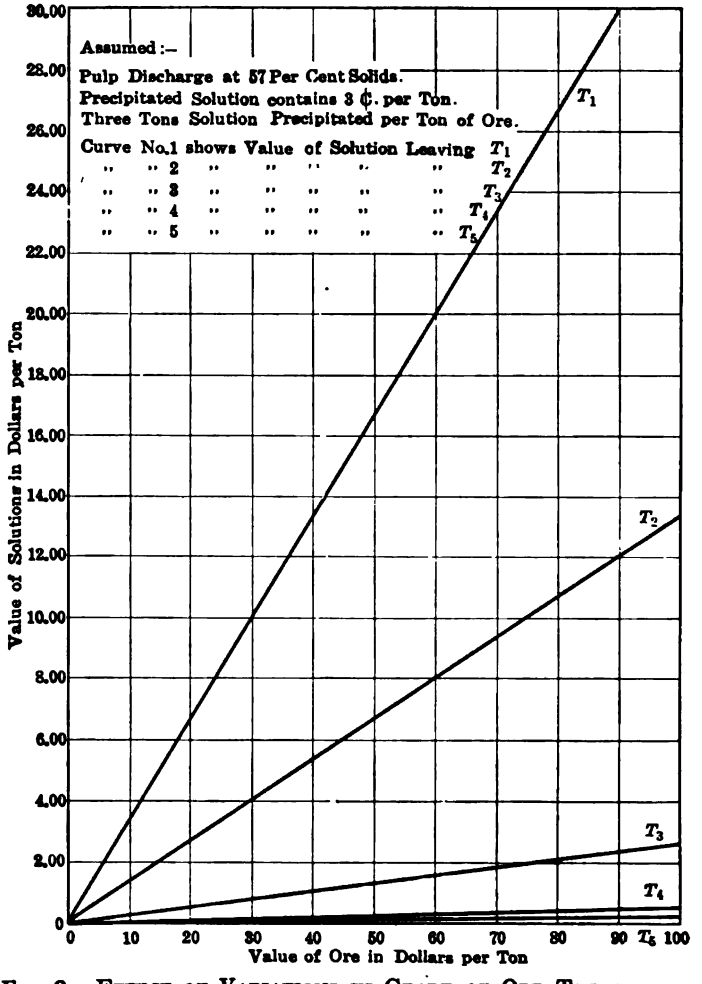

FIG. 2.—EFFECT OF VARIATIONS IN GRADE OF ORE TREATED.

The next, and one of the most important variables to be considered, is the thickness, or percentage of solids, to which pulp can be settled. In Fig. 5 the values of the various overflows have been plotted so as to show the effect of variations in the moisture in the underflows. The full-line curves represent the values of the solutions—that is, they are shown as per ton of solution—but in calculating losses per ton of ore, the ratio

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of the solution to the ore present must be considered. The loss in dissolved gold per ton of dry ore for the last tank has, therefore, been plotted in a separate curve, and it is this curve which should be given the most serious attention in determining the suitability of an ore for decantation. The moisture in the pulp also has a direct bearing on the cyanide loss,

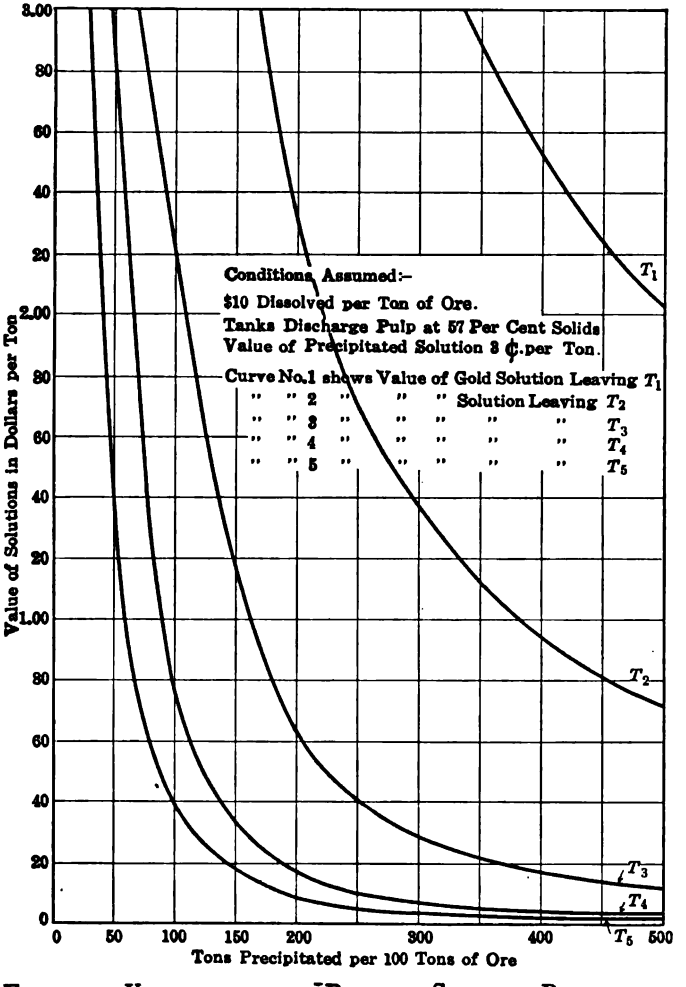

FIG. 3.—EFFECT OF VARIATION IN THE RATIO OF SOLUTION PRECIPITATED TO ORE TREATED.

which is also shown in Fig. 5. This has been shown in pounds rather than in cents because solution strength and the price of cyanide both affect its value. Enough is shown, however, to make it plain that there is a very decided limit to the density of the pulp that can be handled economically, and one is forcibly reminded that the cyanide strength

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should be kept as low as possible. Operators as a rule seem to be inclined to "play the game safe" as regards solution strength and it is probable that in many cases cyanide could be saved without any considerable loss in gold by using solutions of a lower strength.

In making the calculations upon which the foregoing curves are based, it was assumed that 75 per cent. of the gold was dissolved in the grinding department and the remaining 25 per cent. in the agitators. This is, of course, an approximation which cannot be accurate, and even with the cleanest ores some gold is dissolved in passing through the tanks following the agitators. This is a condition that has always been met, no matter

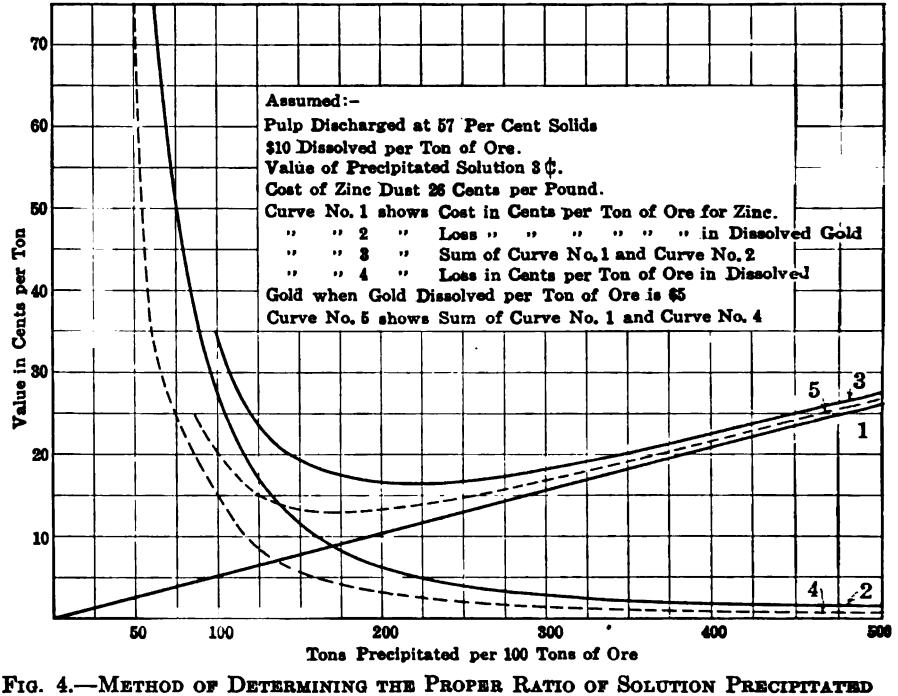

FIG. 4.—METHOD OF DETERMINING THE PROPER RATIO OF SOLUTION PRECIPITATED TO ORE TREATED.

what the method of recovering the dissolved value. Changing solution during agitation has been practiced for years in the treatment of silver ores, and in the treatment of concentrates at Goldfield with a view to reducing this lag of dissolving. I have also found that on Goldfield ore, pulp from the final agitators, reagitated without change or addition of solution, would give up no more gold, while reagitated filter tails from the same ore would show a distinct reduction. The same condition exists in the pulp fed to decantation plants and there is little doubt that some dissolving takes place even in the last tanks of the series. Some of this value is lost, particularly that freed near the final discharge,

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but much is recovered that would probably not be won by any other method. The superintendent of one of the decantation plants treating Tonopah ore informed me that his sampling indicated that the additional dissolving taking place in his plant was sufficient to offset his entire dissolved loss and 3 c. per ton additional.

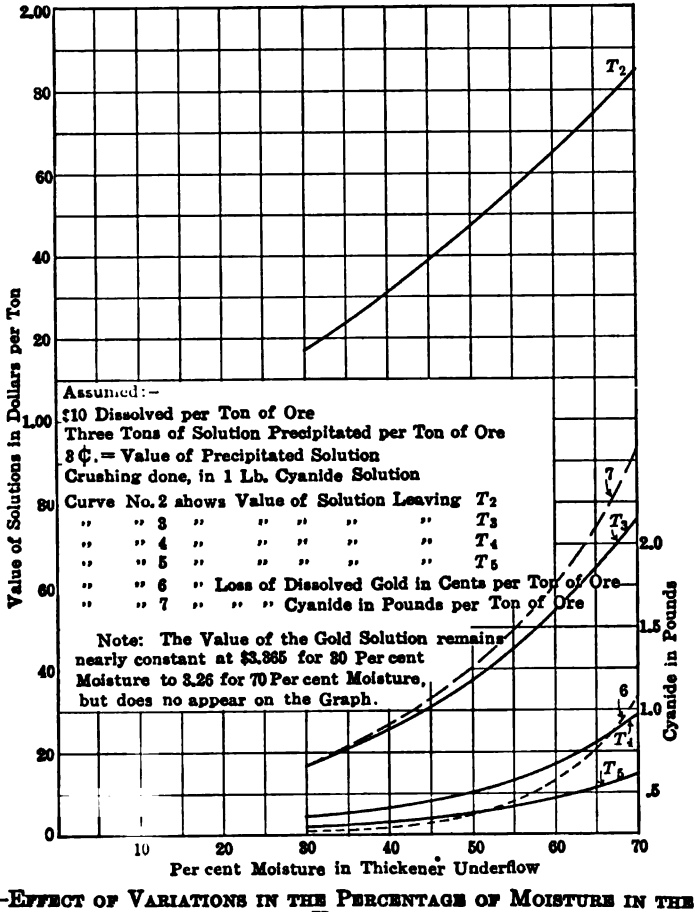

FIG. 5.—EFFECT OF VARIATIONS IN THE PERCENTAGE OF MOISTURE IN THE THICKENER UNDERFLOW.

Any considerable dissolving during decantation will be indicated by a difference in the assay value of the solution in the underflow of the tanks as compared with the overflowing solution. In practice there is always more gold per ton in the underflow solution than in the overflow of any given tank, but in the ores of Porcupine district this difference is very small. Other causes may, and no doubt do, tend to produce this difference between the overflow and the underflowing solution. Adsorption is probably the most important and perhaps the least understood of these. In the case of the ores of the Porcupine district this phenomenon

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is of small importance, as the ore is composed of crystalline schists and quartz and there is little tendency for the ore to flocculate under the influence of the solutions used. The gold and silver ores of the Western States are in many cases in eruptive rocks; these ores usually flocculate in solution and in doing so seem to entrap a portion of the value in the solution. At any rate there is a much more noticeable difference in the assays of tank effluents in the treatment of these ores. Faulty mixing of the products fed to the tank has in some cases been blamed for this. In most plants riffles and other devices are put in launders to insure thoroughness in mixing, but our experience in Porcupine would indicate that the ordinary launder makes a perfect mixture. Adsorption, then, would seem to be the more important cause of these differences, and should be a profitable field for further investigation.

Proper precipitation is essential in decantation, as it has always been in every other process, and, as shown by Fig. 6, dissolved gold is lost in proportion to the amount of value in the barren solution used. It will be observed, however, that the loss does not increase as fast as the value of the barren solution does.

It will be observed that the value of the solution leaving T_4 increases at nearly the same rate as the barren solution and that all the preceding overflows increase by exactly the same amount as T_4. The water dilution cuts the final loss down to a slower increase than the barren solution itself.

Mechanical Features

Dorr Thickener

The mechanics of the decantation system is of the simplest, but is worthy of study none the less. The Dorr thickener is universally used for the separation and is so well known and so standardized that it need only be mentioned in passing. It has been fully described by Mr. Dorr himself in a paper read before the Institute.[1]

The arrangement of the tanks in plan may be largely governed by space available and other local conditions. In elevation it is very convenient to have the last tank the highest and the preceding ones successively lower, so that the overflowing solutions may be transferred by gravity with a minimum of attention. Where such an arrangement is not possible, the tanks may all be placed on the same level using automatically regulated air lifts to transfer the solutions from tank to tank.

Diaphragm Pumps

As a means of transferring pulp, the diaphragm pump has distanced all competitors. The main reasons for this are that it not only pumps

[1] The Dorr Hydrometallurgical Apparatus, *Trans.*, vol. 49, pp. 211–237 (1914).

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but at the same time measures the volume of pulp transferred. The flow of pulp is not easily obstructed by foreign matter, such as chips, waste and

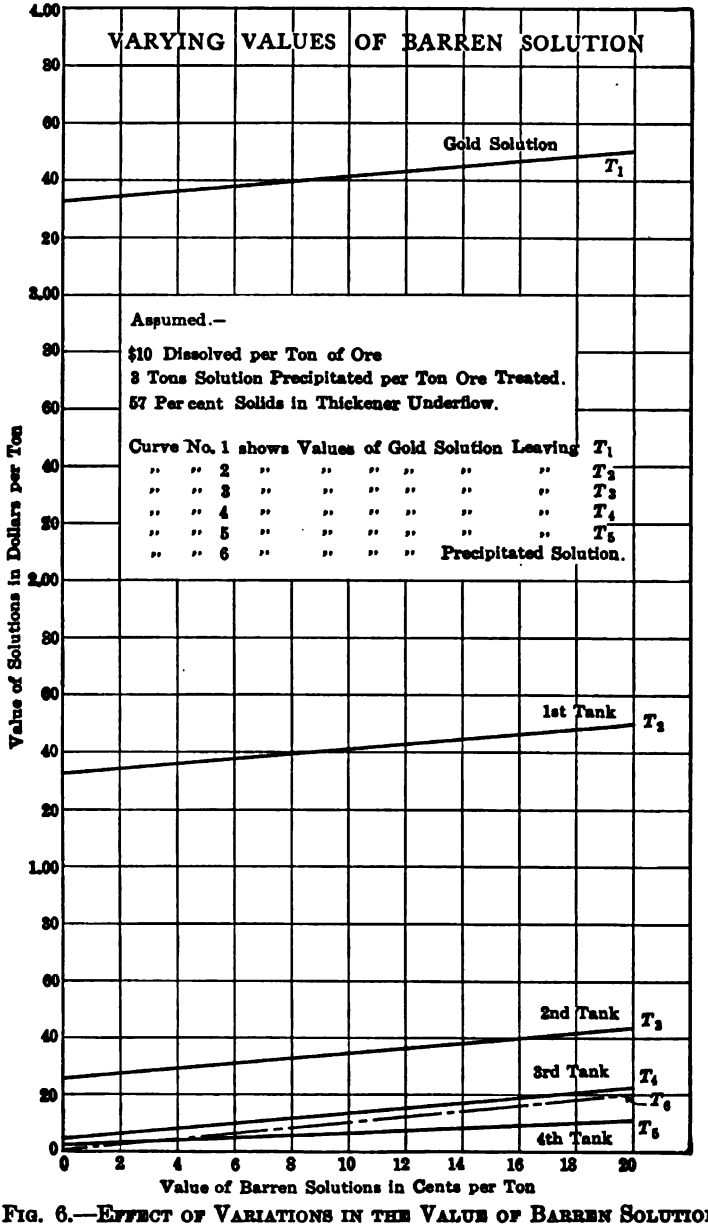

FIG. 6.—EFFECT OF VARIATIONS IN THE VALUE OF BARREN SOLUTION.

the like, since the openings are full pipe size straight through the pump. The capacity of the pump can be controlled with certainty by means of

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cone pulleys. The attendant's duties are all on one floor, as it is operated from the overflow level instead of at the bottom of the tank. Operating cost is very low on a properly designed pump.

Air lifts were used for a time, owing to their cheapness and simplicity, but they are hard to regulate and are wasteful of power. The most careful watching will not prevent them from "running away," that is, transferring too fast, consequently thinning the pulp and sending large quantities of rich underflow solution toward the discharge. On the other hand, if they are set too slow the gradual thickening of the discharge slows up the flow and finally stops it.

Spigot discharge to a bucket elevator or centrifugal-pump sump has also been tried, pulp and solution being mixed and elevated together. There are several objections to this arrangement, the principal one being high power and maintenance cost for the pumping or elevating machinery. Both the solution and the pulp must be lifted more than the full height of the tank for each decantation. The small high-velocity stream of pulp passing through a spigot at relatively high pressure is subject to frequent stoppages due to foreign matter, and even when this does not cause a complete plugging it interferes with uniformity in operation. There is also the objection that work is done on two floors, one below and one above or near the top of the tanks.

There has been a certain amount of prejudice against diaphragm pumps due to the fact that some of the earlier pumps were poorly designed for the work they had to do. Faulty valves were responsible for much of this. Poor methods of regulation also had their effect.

The practice in most plants in the Porcupine district now is to use cone pulleys for the regulation of capacity, although some of the operators favor regulation by varying the length of stroke.

The valves should be of the floating type, as any hinge device will catch the wood chips that are present in the best-screened pulp. The chips lodged in the hinge of the valve cause leaks which, though small in amount at first, cause cutting of the seat and consequently permanent leakage. With the floating valve there is no place for chips to lodge and the whole circumference of the seat is washed by pulp at every stroke. This type of valve also has the advantage that the lower valve may be placed directly below the upper one and made small enough to be lifted out through the upper-valve seat when the upper valve has been removed. No tools are required for the removal of these valves and it is a simple matter to inspect them.

The best results have been obtained when the working surface of both the valve and seat were of high-grade rubber. Belting was used at first, but it was found that minute leaks were almost sure to start, due to the fact that belting is not yielding enough to close over any chip that may lodge on the valve seat, and that a leak once started would ruin both

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valve and seat in the course of a few days. On the other hand, valves of rubber seating on rubber have operated 6 months without the slightest decrease in efficiency and with scarcely perceptible wear.

Diaphragm pumps have been operated at speeds varying from 15 to 100, the higher speeds usually in conjunction with a short stroke.

The practice at the Hollinger mill has been to use a low speed and a stroke as long as the diaphragm could safely stand. Measurements taken on the Hollinger pumps equipped with standard No. 4 Gould diaphragm at 3-in. stroke gave results as shown in the following table:

Number of Strokes	Volume per Stroke, Cubic Feet	Specific Gravity	Per Cent. Solids
14.5	0.139	1.54	54.5
23.0	0.148	1.48	50.5

Tons Solids Pumped per day	Per Cent. Increase in Speed	Per Cent. Increase in Volume per Stroke	Per Cent. Increase in Tonnage
76.3			
114.5	58.5	6.5	50

From the above figures, which are typical of numbers of tests made, it may be inferred that the volume pumped is roughly proportional to the speed of the pump but that leakage is slightly greater on the lower speeds.

The low speeds and the placing of the discharge lips high enough above the discharge valve to leave 3 or 4 in. of pulp over the valve at the end of the upstroke have rendered the pumps of the Porcupine district practically free from the splash and dirt that have been one of the chief objections to diaphragm pumps in the past.

Where tanks have a settling capacity of over 125 tons of solids per day, it may be found advisable to use two diaphragms in parallel, making a duplex or even a triplex pump. This arrangement has several advantages; with the lowered speed, strains of the pump are lessened and distributed, and repairs can be made on one unit without complete stoppage of the tank discharge. Diaphragm life appears to be roughly proportioned to the number of strokes, so that an increase in the number of diaphragms employed, with a corresponding decrease in the strokes per diaphragm does not result in an increased cost for diaphragms.

As a safeguard against waste and other foreign matter it is advisable to screen the pulp before it goes to the decantation tanks, and where small wood chips are to be expected a fairly fine screen, usually of punched plate, will be of great service in protecting thickeners and pumps.

Measurement of Solution Precipitated

Every cyanide plant has some method of determining the amount of solution precipitated, and in decantation plants having only one series

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of tanks the entire tonnage of precipitated solution is used at one place. If, however, more than one series of tanks is used it becomes important to split this precipitated solution into parts proportional to the tonnage of ore being washed in each series. This can not be done by regulating the valve at the outlet of the barren line, as trial has shown that under some conditions an increase of as much as 50 per cent. can be made in the amount flowing from a pipe line without any visible change in the stream.

This difficulty has been overcome by the use of V-notch weir boxes, one for each series of tanks. In each weir box is a float compartment and a float operating an indicator on a scale which reads in tons per 24 hr. While the weir box, which can be readily made at any mine, is not a recording instrument but gives only an instantaneous rate reading, its use greatly simplifies the problem of proper distribution of barren solution. Water also should be added in proper amounts and uniformly distributed. A smaller weir box has been found convenient for this purpose.

It is usual to determine the specific gravities of various pump discharges about a decantation plant at least once a shift. In a small plant this consumes little time but if there is a large number of measurements to be made the distance to be covered by the operator is considerable, especially if he has to return to a central point for each weighing. In such plants, a spring balance with a dial and revolving hand, that can be carried about the plant, is a good type. I have used a milk scale having an adjustable tare-indicating hand, which makes one complete revolution for 10 lb. I use a narrow-necked can which holds just 10 lb. of water when level full. This makes it possible to add a paper dial which can easily be divided so as to read directly specific gravities. The large sample makes possible accurate readings and enables the operator to determine gravity at the place where the sample is taken.

Tray Thickeners

In his paper on the Dorr metallurgical apparatus, Mr. Dorr touched on the future of tray thickeners, and spoke of the possibility of a complete decantation plant being installed in one tank. While this has not been done as yet, a four-step decantation followed by a continuous filter is in operation, the four steps of the decantation being completed in two tanks equipped with single trays. While no figures have been made public the results are said to be creditable.

Another use of the tray in decantation, which is shortly to be tried at the Hollinger mill, is to increase the capacity of existing plant without increasing the number of tanks installed. Since the same grade of solution is handled in both tank and tray there is no danger of any

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mixing of solutions of different grades. This plan should decrease the cost of buildings, tank foundations and building site in almost direct proportion to the increase in capacity gained, while tank cost, power and labor should all be decreased to a marked degree. The outcome of this experiment should have a direct bearing on the future of the decantation process. It must, however, be borne in mind in this connection that some slimes require time for their final thickening and consequently necessitate a tank of some depth, while other slimes find their capacity limit in the thin-pulp settling rate. For this latter class, as explained by Messrs. Coe and Clevenger in their paper on slime settling,[2] a shallow tank or tray is sufficient while the former class requires a tank of carefully calculated depth to give the required time for thickening.

The Decantation Plant of The Hollinger Co.

The Hollinger decantation plant consists at present of five rows of 40-ft. tanks, four tanks to a row, forming a plant of five units. The tanks are arranged with a difference in elevation of 2 ft. 6 in. between steps with the final tanks of the series the highest, so that all solutions gravitate through and out of the plant to precipitation. The Barrett specification roof is supported on flat trusses, the lower chords of which pass just above the tank rims. These trusses also serve to support the thickener mechanisms and the walks between the tanks.

The diaphragm pumps used were designed by the company's staff, and have been very reliable and economical. They are all three-throw or triplex pumps so that in spite of the large tonnage handled the duty on each diaphragm is light. It is not uncommon for diaphragms to last 300 days while the life of the present type of valves and seats has yet to be determined.

The pumps are used not only for pulp transferral, but also for the final discharge. This makes regulation of the final discharge for moisture much easier, more reliable, keeps the work of the operator all on the upper floor and allows the tailing to be discharged at a considerably greater elevation than would otherwise be the case.

The barren solution and water wash added to each row are measured by separate float-reading weir boxes assuring uniform results from the various units.

Labor

The plant is operated by one man per shift who oils all machinery, watches and adjusts the pumps and records their performance. The

[2] Laboratory Method for Determining the Capacities of Slime-Settling Tanks, *Bulletin* No. 111, p. 597 (March, 1916).

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solution man makes titrations and regulates the addition of water solution but has no other duties in the decantation plant. A repair man on day shift makes all repairs and has time for other work.

Power

The power for each tank including motor and line-shaft losses is under 1 hp., while each three-throw pump consumes about the same amount.[3]

Cost

The costs for the 12 weeks from Jan. 28 to Apr. 21, 1916, have been taken as typical of what is done by this plant at its present capacity. During this time 85,854 tons were decanted at a cost of $599 for supplies, including power, and $1,194 for labor, or $0.007 per ton for power and supplies, and $0.0139 for labor, making a total of $0.0209 per ton for decantation. Labor is no doubt higher here than it will be in the future, as a greatly increased tonnage is to be treated while supplies and power should remain nearly the same. The cost as it stands is about 40 per cent. of the cost of filtering on leaf filters at about the same daily tonnage.

Extraction and Recovery

In the ores of the Porcupine district the recovery by dilution seems to be almost the theoretical maximum. Adsorption does not seem to have any appreciable effect. There is a slight dissolving during decantation which, while it adds to the recovery, makes the soluble loss somewhat greater than it would otherwise be.

The figures quoted below on chemical consumption and recovery refer to only two units of the Hollinger plant. The figures on these units are given because the other units of the mill share their feed with the original Moore filter plant, and likewise their barren solution, while for commercial reasons the two units in question have been given a separate solution system and separate precipitation presses. These two units are therefore the only ones upon which all the figures are available.

In comparing the results quoted, however, it should be borne in mind that the flow sheet has been modified in this plant somewhat, because of limitations of space, so that the overflow of T_2 instead of that of T_1 goes to precipitation. The effect of this is to raise the theoretical value of the overflow of the last tank 3 c. at 3 to 1 precipitation.

A statement of results follows:

Period covered, same as that for which costs were given—from Jan. 28, 1916, to Apr. 21, 1916.

[3] More recent measurements show that 20 tanks, of which four contain trays, consume 9.2 hp. motor input; while 24 three-throw pumps take 11.1 hp., or a total of 20.3 for the plant.

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Tons of ore treated	38,885
Value per ton of ore treated	$8.92
Ratio of ore to solution precipitated	100 to 285
Tons solution precipitated	110,604
Strength of cyanide used	0.9 lb. per ton, or 0.0045 per cent.
Cyanide added per ton of ore	0.46 lb.
Difference between pulp feed and pulp discharge for first tank after agitators	25 c.
Average moisture in tails	45 per cent.
Average value of barren solution	3.2 c.
Dissolved gold per ton of solution discharged	11.71 c.
Dissolved gold per ton of ore discharged	9.57 c.

It is theoretically possible, taking into consideration the flow sheet, the grade of ore treated, the barren solution used and the thickness of pulp attained, to have reduced the overflow of the last tank to 7.6 c. leaving a difference of 4.1 c. to be accounted for by continued dissolving, adsorption, etc.

Viewed in one way it may be said that actual losses are 54 per cent. higher than theoretical, but where one is dealing with samples so easily affected by faulty manipulation and where any error except losses in assaying tends to raise the results, a check to 4 c. does not seem bad. The average loss would have been somewhat less if the occasional high results had been omitted, but this was not done.

From the foregoing, I believe one is warranted in concluding that a reasonably accurate forecast can be made of the results to be expected from a decantation plant and that these results may compare very favorably with the results obtained from filter plants.

In conclusion I would say that I am indebted to P. A. Robbins, Managing Director of the Hollinger company, for permission to quote results from the Hollinger plant.

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A Method for Distinguishing Sulphides from Oxides in the Metallography of Steel

BY GEORGE F. COMSTOCK, A. B., MET. E., NIAGARA FALLS, N. Y.*

(New York Meeting, February, 1917.)

IT seems a common opinion among metallographists that all light-gray inclusions seen with the microscope in polished sections of steel are manganese sulphide. Examples of this belief are continually appearing, as for instance in the paper by Dr. Henry Fay on manganese sulphide as a source of danger in steel rails,[1] and in Lieutenant-Commander Cook's paper on Metallography of Steel for U. S. Naval Ordnance.[2] Slate-

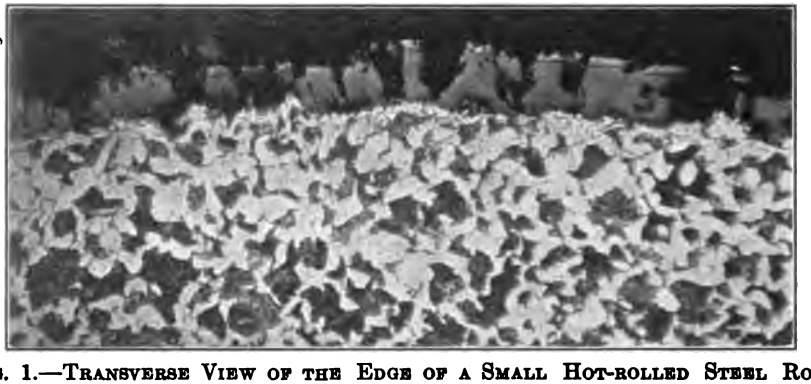

FIG. 1.—TRANSVERSE VIEW OF THE EDGE OF A SMALL HOT-ROLLED STEEL ROD. ETCHED WITH PICRIC ACID. × 200.

colored inclusions are considered to be silicates, and dove-gray inclusions, manganese sulphide. To show the danger in the latter unqualified assumption, it should be sufficient to examine the edge of any piece of steel that is covered with a fairly thick scale, and that has been polished in such a way as to prevent the scale from breaking away entirely below the polished surface. This can be done by protecting the scaled edge of

* Metallographist, Physical Testing Laboratory, Titanium Alloy Manufacturing Co.

[1] A Microscopic Investigation of Broken Steel Rails: Manganese Sulphide as a Source of Danger. *Proceedings of the American Society for Testing Materials*, vol. 8, p. 74 (1908).

[2] *Trans.*, vol. 53, p. 238 (1915).

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the specimen with some soft metal, or fusible metal applied molten, or ordinary red fiber such as electricians use for insulating. An examination of this scale, if the polishing has been done well, and the sample is preferably unetched, will show it to be light gray in appearance, and apparently "manganese sulphide." As a matter of fact, however, we know that scale is chiefly iron oxide, and contains sulphur only as an impurity. Its similarity in appearance to manganese sulphide is shown in Fig. 1, which is a transverse view of the edge of a small hot-rolled steel rod, magnified 200 diameters. The steel has been etched with picric acid, but the gray scale can be plainly seen at the edge of the metal. The very dark material beyond the scale is the red fiber in which this specimen was clamped to prevent the scale from falling away during polishing.

In the course of several years' work with the microscope on rail steel and other commercial steels the writer has not infrequently seen light-gray inclusions that, from evidence given by sulphur prints, or for other reasons, did not seem likely to be sulphides, and in some cases these were known without any doubt to be oxides, like the scale mentioned above. Finally, a case of some importance arose, where it was desired to know definitely whether certain light-gray inclusions were sulphides or not, and it was not possible to decide the question from their color alone. Reference to the standard textbooks on metallography did not help much, for the only real test described for distinguishing oxides from sulphides involved heating the polished sample in a current of carefully purified hydrogen, and this method was considered as too complicated and necessitating too much preparation and practice. Etching with weak organic acids was not found satisfactory, and neither was the method of using a drop of sulphuric acid and watching for bubbles of hydrogen sulphide gas. Tried with known sulphide inclusions, both these methods attacked the steel itself too strongly, or else, if the acids were weakened, had no effect either on steel or sulphides. It was finally decided to get together a few specimens containing sulphides, and others in which oxides were known to be present, and to try various solutions on them until one was found that would attack one type of inclusion and not the other.

This work had not progressed very far when it was remembered that boiling alkaline sodium picrate, which is used to darken cementite, having no effect on ferrite and very little on pearlite, was always found to attack the sulphide inclusions in a sample, leaving black pits instead of the light-gray spots seen before etching. Fig. 2, for instance, shows some sulphide inclusions in a polished section, unetched, cut from a segregated streak in a steel rail, where sulphur prints indicated the presence of high sulphur. This same spot etched with boiling alkaline sodium picrate is shown in Fig. 3, where the sulphides are seen to have been blackened by this reagent. The darkening seems to be due to an actual solution of the manganese sulphide in the etching liquid, so that instead of a smooth

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polished surface, we now have a surface full of pits or hollows, where the sulphides were, and as these hollows do not reflect any light back into the microscope with vertical illumination, they appear black in the field of view. Figs. 4 and 5 show some other sulphide inclusions in a shell-steel billet, before and after etching, respectively. In this lower-carbon steel

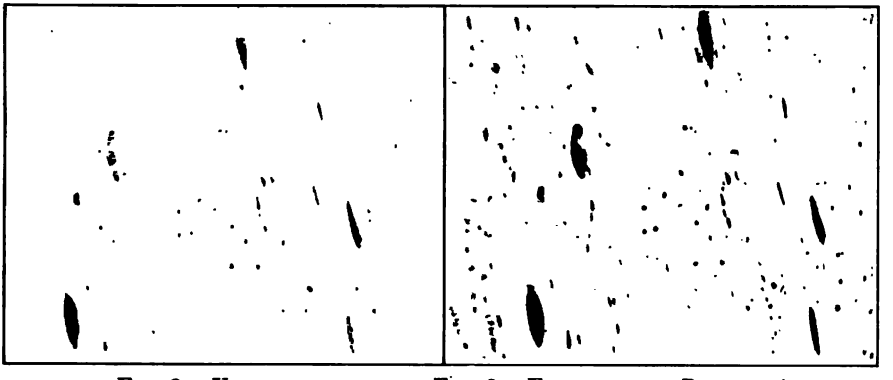

FIG. 2.—UNETCHED. FIG. 3.—ETCHED WITH BOILING ALKALINE SODIUM PICRATE.

Sulphide Inclusions in a Polished Section Cut from a Segregated Streak in a Steel Rod. × 130.

the ferrite-pearlite structure is brought out faintly by the etching, but not enough to interfere with the distinct blackening of the sulphides.

When this etching with boiling alkaline sodium picrate was tried on steel known to contain oxides, the problem of distinguishing oxides from

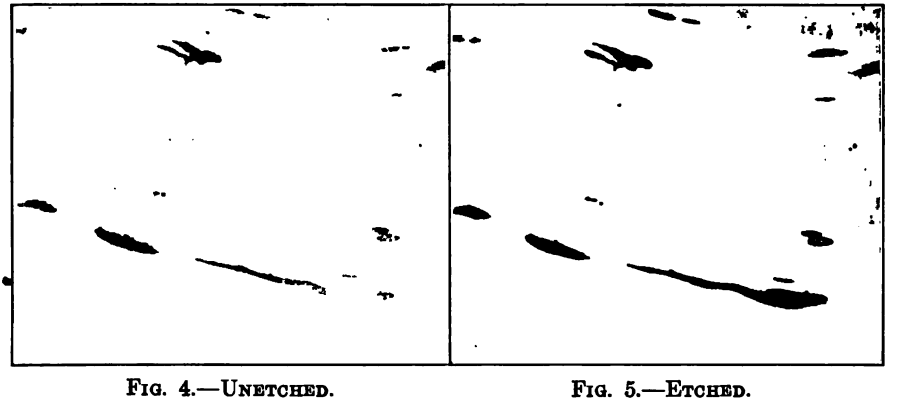

FIG. 4.—UNETCHED. FIG. 5.—ETCHED.

Sulphide Inclusions in a Shell-Steel Billet. × 130.

sulphides was solved at once, for the oxide inclusions were found to be absolutely unattacked. Fig. 6 shows some oxides in a polished unetched sample of clean rail steel treated while molten in a crucible with nickel oxide. The steel undoubtedly reduced the nickel, forming iron oxide.

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Fig. 7 shows this same spot after etching, and looks exactly like the unetched view. Fig. 8 shows a cross-section, unetched, of a seam in the base of a steel rail. These surface seams are always filled with scale, or oxide, although the light-gray color has sometimes led observers to believe that manganese sulphide was present. The fact that such seams give

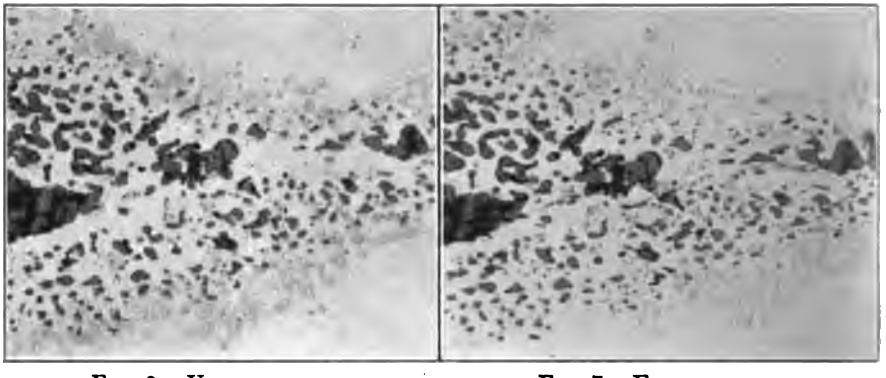

FIG. 6.—UNETCHED. FIG. 7.—ETCHED.
Iron Oxides in Rail Steel. × 130.

light streaks instead of black spots on sulphur prints and etched sections, however, shows that oxide and not sulphide is really present in them. This seam, etched with boiling alkaline sodium picrate, is shown in Fig. 9. The oxide is seen to be entirely unattacked.

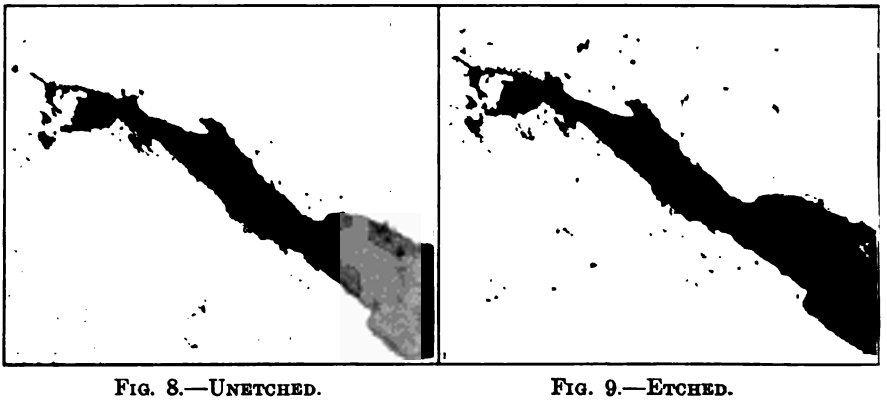

FIG. 8.—UNETCHED. FIG. 9.—ETCHED.
Cross-section of a Seam in the Base of a Steel Rail. × 130.

The application of this solution to etching all grades of steel for distinguishing between oxides and sulphides has not been described before, so far as the writer is aware. In ease of application and certainty of results it is much superior to other methods that have been suggested for this purpose. One advantage is that the solution used is one that every

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metallographist should have on hand for etching cementite in high-carbon steel. Thus this same solution can be made to serve two purposes. It was first discovered by Kourbatoff, who recommended it for darkening cementite because it does not attack ferrite or pearlite. The writer makes up the solution according to the formula given in Sauveur's *Metallography and Heat Treatment of Iron and Steel*, the procedure being about as follows: 25 g. NaOH are dissolved in 60 to 70 c.c. of water, 2 g. of picric acid are added, and the solution is heated until the picric acid is dissolved, when the volume is brought up to 100 c.c. by adding more water. In using this solution to etch polished-steel specimens, it is brought to boiling in a beaker on a hot plate, the specimen is immersed in the solution, and boiling is continued for 10 min. Then the specimen is removed, washed, and dried. The polished and etched surface may be wiped dry with chamois skin without affecting the results. If the speci-

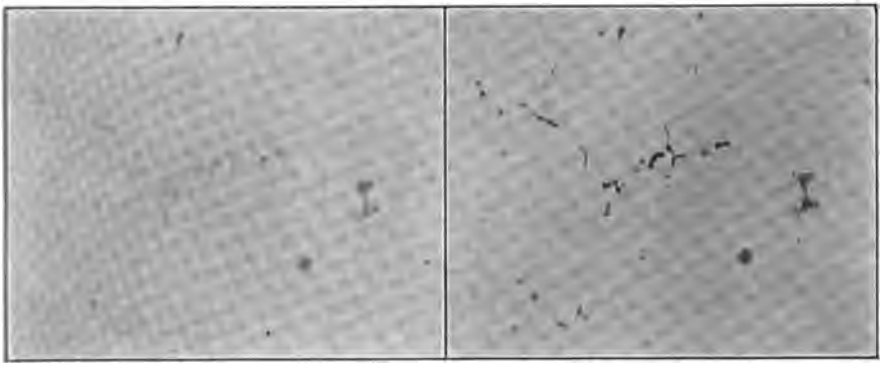

FIG. 10.—UNETCHED. FIG. 11.—ETCHED.
Polished Section of a Steel Casting which Cracked while Hot. ×130.

men contains cracks, this wiping is necessary, and repeated washing and drying may also be necessary, to remove the last traces of the sodium picrate from the surface to be examined. The solution may be used many times, but as the boiling makes it more concentrated water should be added occasionally to keep the volume about 100 c.c.

Before closing, some illustrations will be given of the practical use of this method for distinguishing oxides from sulphides. Fig. 10 shows a polished section of a steel casting which cracked while hot and in which the presence of oxides might therefore be expected. An experienced eye could perhaps tell which of these inclusions are sulphides and which are oxides, but the distinction at best would not be convincing. After etching by the method described above, however, the difference between the two kinds of inclusions becomes evident, as shown in Fig. 11. The arrangement of these sulphides is noteworthy, as a network is strongly suggested, and by this method of etching it can easily be shown that the

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small inclusions often found in a network arrangement in the ferrite of well-deoxidized steel castings are merely finely divided sulphides. An instance of such inclusions was described by Dr. Henry Fay in a paper on Some Causes of Failures in Metals,[3] and well illustrated by his Figs. 16 and 17 on page 450. These are said to be "slag" by Dr. Fay, but their appearance is exactly similar to numerous occurrences of sulphides which have come to the writer's attention.

A large complex slag inclusion, also in a cracked steel casting, is shown unetched in Fig. 12. The light-colored rough-looking part of this inclusion at the center of the field of view was bright yellow, and hence considered to be sulphide of iron. The other light-gray portions, including the spots in the dark-gray part, were supposed at first to be manganese sulphide, and the darker part, manganese silicate. But when etched with boiling alkaline sodium picrate, only the yellow constituent was

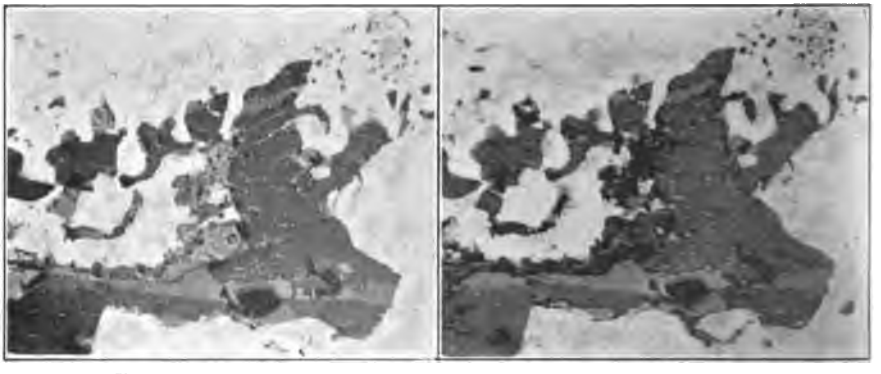

FIG. 12.—UNETCHED. FIG. 13.—ETCHED.
Large Complex Slag Inclusion in a Cracked Steel Casting. × 130.

attacked, showing that it alone was sulphide, and the light-gray constituent was oxide instead of manganese sulphide. The appearance after etching is shown in Fig. 13.

A section cut from a bar of "ingot iron" that was red-short and cracked in rolling is shown in Fig. 14, taken before etching. Most of the inclusions found here were light gray in color, but many of them had yellow parts attached to them at one or both ends. The yellow substance, which is very rarely seen in steel, was taken to be sulphide of iron, and the gray inclusions were at first supposed to be manganese sulphide. But the manganese content of this ingot iron was found to be very low, and the sulphur also was not high enough to allow the presence of so many sulphides. Consequently, it was doubtful whether these gray inclusions were sulphides or oxides, until the sample was etched with the boiling alkaline sodium picrate solution, when it was apparent at once

[3] *Proceedings of the American Society for Testing Materials*, vol. 11, p. 439 (1911).

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that they were oxides, and only the yellow substance was sulphide. The inclusions of Fig. 14 are shown after etching in Fig. 15, where the distinction between oxide and sulphide is very plain. One of the largest particles of the yellow constituent was noticed, before etching, to have

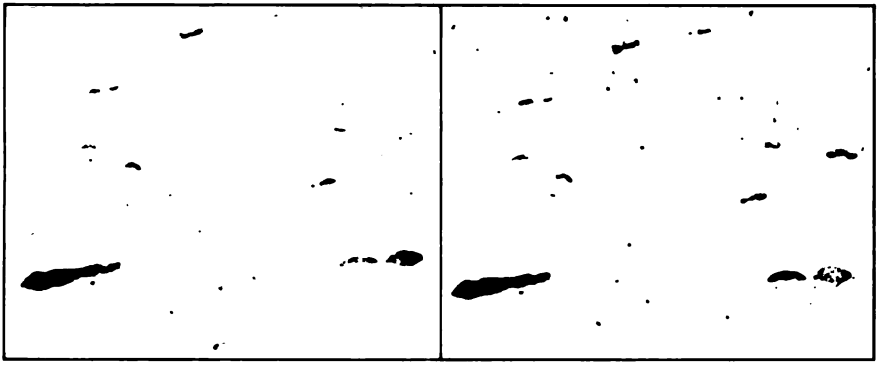

FIG. 14.—UNETCHED. FIG. 15.—ETCHED.
Section Cut from a Bar of "Ingot Iron" that was Red-short and Cracked in Rolling.
× 130.

a spotted appearance, and when examined under high magnification it was seen to be full of very fine gray spots. This is evidently a eutectic of iron oxide and iron sulphide. It is shown in Fig. 16, magnified 570

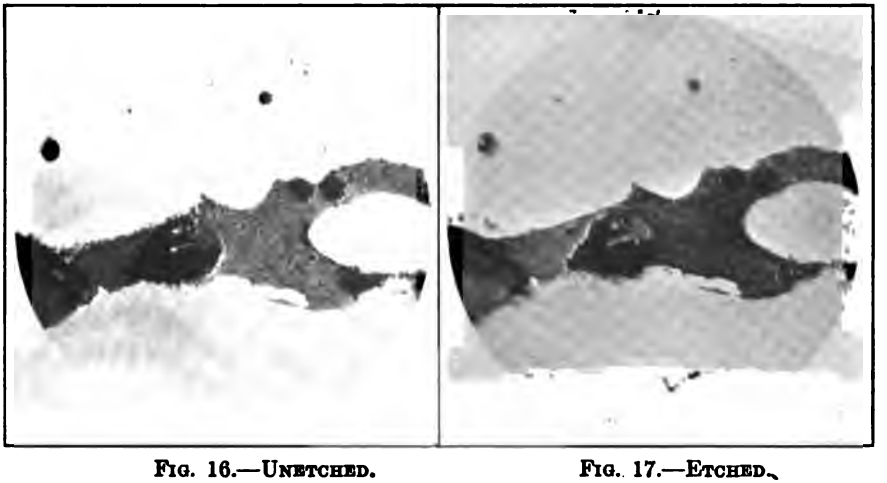

FIG. 16.—UNETCHED. FIG. 17.—ETCHED.
One of the Large Inclusions in Fig. 14. × 570.

diameters, the large unspotted dark-gray areas, with the black pits due to fracture in polishing, being oxide, and the spotted area being the eutectic. The oxide is here seen to be darker than the sulphide. Fig. 17 shows this same spot after a short etching with boiling alkaline sodium

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picrate, and it is apparent that the eutectic, but not the oxide, has been attacked, for the former is now darker than the latter, and its structure is blurred and indistinct.

These notes are presented in the hope that the simple method here described will appeal to metallographists in general as being worthy of trial in cases where the identity of light-gray inclusions in steel is in question, so that in future sulphides need not be called "slag" and the error of calling oxide or scale inclusions "manganese sulphide" merely because their color is light gray may also be avoided.

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On Grain Growth

BY HENRY M. HOWE, LL.D.,* BEDFORD HILLS, N. Y.

(New York Meeting, February, 1917)

THE brilliant and very original matter in Professor Jeffries' discussion† should rank not only as an independent paper, but as a most important one. In particular, the explanation which it gives of the remarkable Sauveur phenomenon, over which so many of us have puzzled long and hard, is so clear, complete, and cogent as to go far toward establishing Professor Jeffries' hypothesis. The very features of Professor Sauveur's photographs which fit this hypothesis so exactly are reproduced in Chappell's photographs of steel coarsened under somewhat different conditions.[1]

The ideas developed by these discoveries are so novel that convenience of language seems to call for new conventional terms for expressing them. In what follows here I propose certain terms for this end, for criticism and improvement, and I attempt to generalize from Professor Jeffries' hypotheses.

Generally speaking, a given set of grains which undergo no growth in the cold will, when the temperature is raised progressively, start to grow on reaching a certain temperature, and will continue growing at all higher temperatures.

The "germinative" temperature is that at which growth begins, quite as the conditions which cause a seed to grow are "germinative." We have then an *inert temperature range* below the germinative temperature, and a *growth range* above it, the germinative temperature being the limit between these two ranges. So, too, a germinant grain is one which is at its germinative temperature, and hence beginning to grow, while inert grains are those below their own germinative temperature, and the term "growing grains" includes those which are germinant and those which are above their germinative temperature. These last could be called "supergerminant" if the distinction should be needed.

"Germinative temperature" seems better than "critical temperature

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*Emeritus Professor of Metallurgy, Columbia University.

†Of the paper of C. H. Mathewson and Arthur Phillips, *Bulletin* No. 113, p. 987 (May, 1916).

[1] *Journal of the Iron and Steel Institute*, 1914, Pt. I, Plates 46 and 47, following p. 496.

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for grain growth" and "equiaxing temperature," first because "critical" has been so much overworked and suggests neither critically fast nor critically slow growth, and second because a germinative temperature is what the hypothesis postulates, and because the further hypothesis that this coincides with the equiaxing temperature is only an inference and for the present purpose an unnecessary one. Indeed, Professor Jeffries[2] informs me that he has found cases in which the germinative temperature is far above that of equiaxing.

When only a limited part of the metal is at its germinative temperature, this becomes a "fast-growth" temperature, because the germinant grains feed unopposed on the inert ones against which they abut.

The late results of White and Wood[3] tend to complicate our ideas, for they indicate that germinance depends not alone on temperature but also on time, so that germination may occur on a long exposure to a temperature at which none can be detected after a shorter exposure. This indeed agrees better with our general knowledge of "aging." In this view inertness is not absolute but only relative.

Grain growth, that is the absorption of certain grains by certain others, is favored, and the *germinative temperature* is consequently lowered, (*A*) by grain fineness, (*B*) by grain-size contrast, and (*C*) by prior plastic deformation.

(*A*) *Grain fineness* favors grain growth because it implies extensive grain boundary surface at which alone can absorption occur. To vary the angle slightly, a growth which doubles the size of extremely fine grains means only that each alternate very fine grain absorbs its adjoining still smaller neighbor, a much faster process than the absorption of one enormous grain by its even greater neighbor, faster whether we hold that the whole of a given grain is absorbed simultaneously, or, as seems more probable, that the swallowing is gradual, as for instance by some kind of boundary migration.

(*B*) *Grain-size contrast*, that is the difference in size between adjoining grains, favors grain growth because the absorptive and the resistant powers of a grain increase with its size, so that, when there is great grain-size contrast, growth is favored jointly by the great absorptive power of the larger grains and by the weak resistance of the smaller ones.

(*C*) *Plastic deformation* or *overstrain* favors grain growth, perhaps by leading to fineness of grain and to grain-size contrast. The fact that the germinative temperature is the lower the greater the deformation has been tallies with the former explanation, because increasing deformation would naturally lead to finer grain size. It may also increase the grain-size contrast. But the fact that grain growth will not occur at all unless

[2] Private communication, June 24, 1916.

[3] Recrystallization as a Factor in the Failure of Boiler Tubes, *Proceedings of the American Society for Testing Materials*, xvi, 1916, to appear.

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there has been plastic deformation suggests that some additional cause applies here.

(*D*) Grain growth occurs the more readily the higher the temperature, and the greater consequently the mobility.

Grain growth is opposed, and consequently the germinative temperature is raised, by:

(*E*) Obstructions, such as cementite, slag, and other sonims in iron, and thorium oxide in tungsten; and by

(*F*) Grain-size equalization, or the decrease of grain size contrast.

The rate of growth increases:

(*G*) With the difference in absorptive power between adjoining grains, whether this is due to a difference in their:

(*H*) Temperature; or in their

(*J*) Size, that is to grain-size contrast; or in their

(*K*) Prior plastic deformation. It further increases:

(*L*) With the temperature;

(*M*) With the initial grain fineness; and

(*N*) With the prior plastic deformation.

(*P*) The rate of growth decreases as the grain size increases, because this increase implies a corresponding decrease in the extent of grain boundary surface.

The very fastest growth seems, from Professor Jeffries' results, to be due to (*H*), which is thus an extremely strong influence. From the fact that low temperature, if it does not arrest growth, at least retards it very greatly, we must admit that (*L*), too, has great influence. Further light on the strength of (*J*), grain-size contrast, is needed. Whether exact identity of grain size would arrest growth completely if the temperature and the degree of plastic deformation were identical, seems hardly capable of proof, because that identity can never be attained.

The reported cases of grain-size equilibrium, in which growth is reported to cease, seem to me more easily explained as resulting from (*P*) the progressive decrease in the extent of grain boundary, and from (*J*) a progressive lessening of the grain-size contrast as the smaller grains are eliminated successively, and a progressively smaller number of larger ones remains. From this point of view the apparent arrests of grain growth and the cases of so-called "grain-size equilibrium" are to be regarded as only retardations, however marked these may be.

Grain growth may be divided into the *normal* and the *exaggerated.* In most industrial heatings there is normal grain growth, that is to say a coarsening which is leisurely unless the temperature approaches the melting point. In strong contrast to this is the exaggerated growth noted by Stead, Carpenter, Sauveur, Ruder, Chappell, and Jeffries, which may be extremely rapid and may lead to extreme coarseness.

If we leave out of consideration temperatures approaching the melting

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point, exaggerated grain growth seems to occur only at the contact of germinant and inert grains. Thus in Jeffries' experiments accelerated coarsening occurs only and always in case the mass is held under a current of such a strength as to bring only part of it to the germinative temperature, the outer and colder parts being inert because below this temperature. Here growth occurs at the germinant internal surface, the germinant grains feeding on the colder inert ones next outside them, thus increasing in size and thus creating a grain-size contrast, which according to Principle (*A*), enables them to feed on the still more inert grains still nearer the outside, and also to begin feeding on the neighboring grains nearer the axis, which, though growing because they are in the growth range of temperature, are growing more slowly because the grains on which they are feeding are themselves active, and thus offer more resistance to absorption than the inert grains against which the grains in the germinant layer abut.

Turning from this case, in which the contact of germinant and inert grains is brought about by the existence of a progressive change in temperature from shell to axis, we have the Sauveur cases in which it is due to a variation in the degree of plastic deformation which the different layers have undergone. When a piece which has been bent or has received a Brinell impression is next heated to a temperature which is at or above the germinative temperature of the most deformed layers, the germinative temperature of some one layer, or in case of a bent bar of some two layers about equidistant from the neutral axis, will equal this existing temperature. This layer and all the more deformed ones will begin growing, but the germinant layer will grow faster than the others, because it feeds on the inert because less deformed layers beside it.

A probably more accurate statement is that exaggerated growth occurs where there is a sharp increase of inertness from layer to layer. Here the grains of a certain layer are so much more active than the more inert ones in the adjoining layer against which they abut that they have a marked advantage as regards germinative power. The lead which this advantage gives them increases progressively as growth proceeds.

The experience of the wire-drawers, that though annealing often causes marked coarsening after the slight deformation of the wire-straightening process, it does not after the extreme deformation of industrial wire-drawing, conforms exactly with Professor Jeffries' hypothesis.

Here we may suppose that, after wire-drawing, the temperature is so low that in the early stages of the heating up it is passed through too rapidly to permit appreciable grain growth at this relatively low temperature, whereas after the slight deformation of straightening the germinative temperature may well lie at the relatively high temperature of annealing, at which first the sojourn is long, and second the rapidity of growth is relatively great. The reason why extremely slight deforma-

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tions do not lead to coarsening on annealing is that the position of the germinative temperature which they determine is so high that it is not reached in annealing, if indeed it does not approach closely the melting point of the metal.

When a principle explains so much, the existence of certain things which it does not explain argues, not against its truth, but for the existence of additional principles not yet discovered. The two difficulties which I now raise should be taken in this sense.

My first difficulty concerns the Sauveur specimen of Professor Jeffries' Fig. 2, Professor Sauveur's Fig. 30. In this long stay at the germinative temperature of the outer parts of the coarsened bands, the grains in these bands should have fed to an important degree on the inert fine-grained core, and their resultant growth, proceeding as it should at right angles to the contact between the fast-growing band and the central core, should have developed columnar grains. Yet in fact the grains look equiaxed. On the other hand, such a columnar arrangement is prominent in the Chappell specimen just referred to.

My second difficulty is that this five-banded arrangement, with two outer bands of intermediate coarseness and a central fine one, should be the rule and not the exception, and should not have passed undetected the vigilant scrutiny of so many observers, for the coarsening is easily visible to the naked eye, and should not have had to wait till 1912 for its discovery. Thus the Sauveur coarsening is lacking in so large a proportion of cases as to suggest that it is held in check by one or more additional principles. I suggest the obstructing effect of foreign bodies as a possible explanation of one set of cases in which coarsening is lacking, a set illustrated by Professor Sauveur's finding that marked coarsening does not occur unless the carbon content is between 0.04 and 0.12, and that it occurs most readily when the carbon content is between 0.05 and 0.07 per cent.[4]

The presence of 0.12 per cent. of carbon, implying that of 13 per cent. of pearlite, might well offer enough mechanical obstruction to prevent coarsening, for each pearlite mass would act as a wholly foreign body to bar the union of the two grains between which it lies. Even as little as 0.07 per cent. of carbon, implying the presence of about 8 per cent. of pearlite, might have an important obstructive effect. On the other hand, when the carbon content is less than 0.04 or 0.05 per cent., there may well be enough iron oxide or minute gas bubbles to offer serious obstruction. Here we should remember Benedicks'[5] discovery that though the specific volume of his steels was strictly proportional to the carbon content between the limits of 0.45 and 1.20 per cent., that of his iron with 0.08 per

[4] H. M. Howe: *The Metallography of Steel and Cast Iron*, p. 363 (1916).

[5] Carl Benedicks: Recherches physiques et physico-chimiques sur l'acier au carbone, pp. 2, 30 and 35, Paris, 1904.

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cent. of carbon and 0.03 per cent. of silicon was abnormally great. It is true that this last was wrought iron, and hence that its specific volume would be made abnormally great by the presence of slag. But with only 0.03 per cent. of silicon the slag content should be extremely small, probably not more than 0.20 per cent., and hence insufficient to explain the surprisingly small density, which I refer therefore to the presence of minute gas cavities so often present when the carbon content is so small.

This brings us to a hitherto unsuspected element of fitness for man's needs which iron has, that the coarsening of its ferrite is limited to temperatures so far below the melting point as to give great viscosity, and hence to restrict the coarsening tendency. On rising past *Ac*1, the 8 per cent. of pearlite which we have just considered turns into 8 per cent. of austenite, and this increases very rapidly in quantity, till at *Ac*3 it forms the whole. As it increases in quantity it forms a more and more complete obstacle to the coarsening of the ferrite with which it is now associated, so that in fact the coarsening of ferrite, even in lower-carbon steels, is probably confined to temperatures below 800°, and thus many hundred degrees below the solidus, and hence corresponding to great viscosity.

Moreover, we may suspect that at *Ac*2 the change from alpha to beta ferrite, which Burgess and Scott[6] seem now to have established clearly, will break up any coarsening which has occurred in the alpha ferrite below this temperature. This, if true, would confine the coarsening range to below 768°.

These considerations lead us to expect to find iron less subject to coarsening than most metals.

It is true that Professor Jeffries' hypothesis explains satisfactorily and without needing this obstruction principle certain other cases of unexpected failure to coarsen, for instance that of my tapered bar of steel of about 0.01 per cent. of carbon from the American Rolling Mill Co. This, after straining with tensile stresses varying from 40,640 to 49,000 lb. per square inch, failed to coarsen noticeably in a 22-hr. exposure to 680°. Because the yield point of this steel is only about 20,000 lb. per square inch, even the least of my stresses may well have caused a degree of deformation so great that the corresponding germinative temperature was far below 680°, so that, in heating up to 680°, the germinative temperature of each of the various parts was passed through too rapidly to lead to material coarsening.

Yet this explanation hardly applies to Professor Sauveur's failure to coarsen his bent specimen in a 7-hr. exposure to 650° in case the carbon content is less than 0.04 per cent., for here there should be layers near the neutral axis in which the plastic deformation should vary progressively from nil to far above that for which the germinative temperature is 650°. Somewhere in this progressive series there should be a layer with

[6] *Comptes Rendus*, vol. 163, No. 2, p. 30 (July 10, 1916).

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the deformation for which 650° is the germinative temperature, and this layer should have coarsened at the expense of its less strained neighbors nearer the neutral axis. But here the failure to coarsen may be explained by the suspected presence of gas bubbles, to which such low-carbon steel is so subject.

The important discovery that, though long heating at a low temperature might cause great coarsening, yet a quick passage up to a high temperature might prevent it, was communicated to me by Professor Jeffries on July 1, 1915, and is recorded, together with most of the information on which his present remarks are based, in a report of his which I have read, dated May 14, 1915.

Turning to Professor Jeffries' contention[7] that uniformity of etching tint is not valid evidence of uniformity of orientation within any given grain, though he would be right if he confined himself to saying that the retention of a uniform etching tint by a single grain is not proof positive of uniformity of orientation throughout, yet when each of many grains in a single field retains a uniform etching tint which differs from that of its neighbors, the inference is irresistible that there is a far greater approach to uniformity of orientation within each grain than between adjoining grains. The fact that certain specific rotations need not alter the brightness of the reflection is beside the mark. What is important is that the great majority of random rotations will alter it.

The irregularity of the slip bands and X bands, in the absence of corresponding irregularity of etching tint, is explained more readily as representing the effort of slip to integrate into a line parallel to the major stress, by stepping back and forth from one to another of two or more sets of conjugate slip planes, than as representing irregularity of orientation. The fact that this irregularity of the slip bands, from being negligible in the first stage of deformation, increases as deformation proceeds, accords with my interpretation, for whereas increasing deformation does not cause any corresponding irregularity of etching tint, it ought to facilitate this change from set to set of planes, by increasing both the frequency and the thickness of the amorphous sheaths surrounding the residual crystalline blocklets, of which the crystallographic slip planes are the paths of that slip. For whatever tendency slip has to persist along a given plane as long as it is passing through crystalline matter is clearly lessened when that slip comes to the amorphous sheath which parts each crystalline grain-fragment from the next.[8] The regularity of the etching pits in each grain even after extreme plastic deformation and the sharp change in their direction from grain to grain are very cogent evidence of the retention of substantial uniformity of orientation during deformation.[9]

[7] *Bulletin* No. 113, p. 987 (May, 1916).

[8] Howe: *The Metallography of Steel and Cast Iron*, pp. 316, 483.

[9] *Op. cit.*, pp. 287, 485.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Geology of the Bawdwin* Mines, Burma, Asia

BY M. H. LOVEMAN, BAWDWIN, BURMA, ASIA

(New York Meeting, February, 1917)

THE orebody described below has been rediscovered and developed within the last 3 years. It has, however, been known and worked by the Chinese for hundreds of years. When assay values and size are considered, it contains what is probably one of the largest single bodies of zinc and lead sulphides yet found. The tonnages developed at other places, as at Broken Hill, are much greater but the average zinc and lead sulphide contents are considerably lower than at Bawdwin.

Bawdwin is situated in the semi-independent State of Tawng Peng, one of the numerous units which make up the Northern Shan States. These States are under British rule and are generally considered as a portion of Burma. They are governed as a separate administrative unit, however, under the Lieutenant Governor of Burma. Each State is under its own chief, or Sawbwa, who has considerable power over his own subjects.

Bawdwin is approximately in latitude N 23°6′, longitude E 97°20′, about 50 miles south of the nearest point on the Chinese border and 450 miles north of Rangoon (Fig. 1). The period at which mining was first begun at Bawdwin is unknown. The most reliable records have been obtained by deciphering local Chinese inscriptions and from these it is concluded that the work dates back to at least the beginning of the 15th century. The first undoubted reference to the mines by a European is by Symes in 1795. Crawford in 1827 estimated the output of silver from the mines at $600,000 annually. At about the middle of the 19th century the rebellion of the Chinese Mohammedans in the neighboring Chinese province of Yunnan so weakened the power of the Chinese that the incursions of hostile tribes (Kachins) made work extremely dangerous. The mines were finally abandoned by the Chinese about 1868. The closing down of the mines was probably not entirely due to outside causes but may have been influenced somewhat by the increasing difficulty of handling the water, as the workings gradually descended below the water level of the district. Several attempts were made by the Burmese kings, Mindon Min and Thebaw, to resume operations, but because of a

* Bawdwin—from Burmese: baw = silver and dwin = well or mine.

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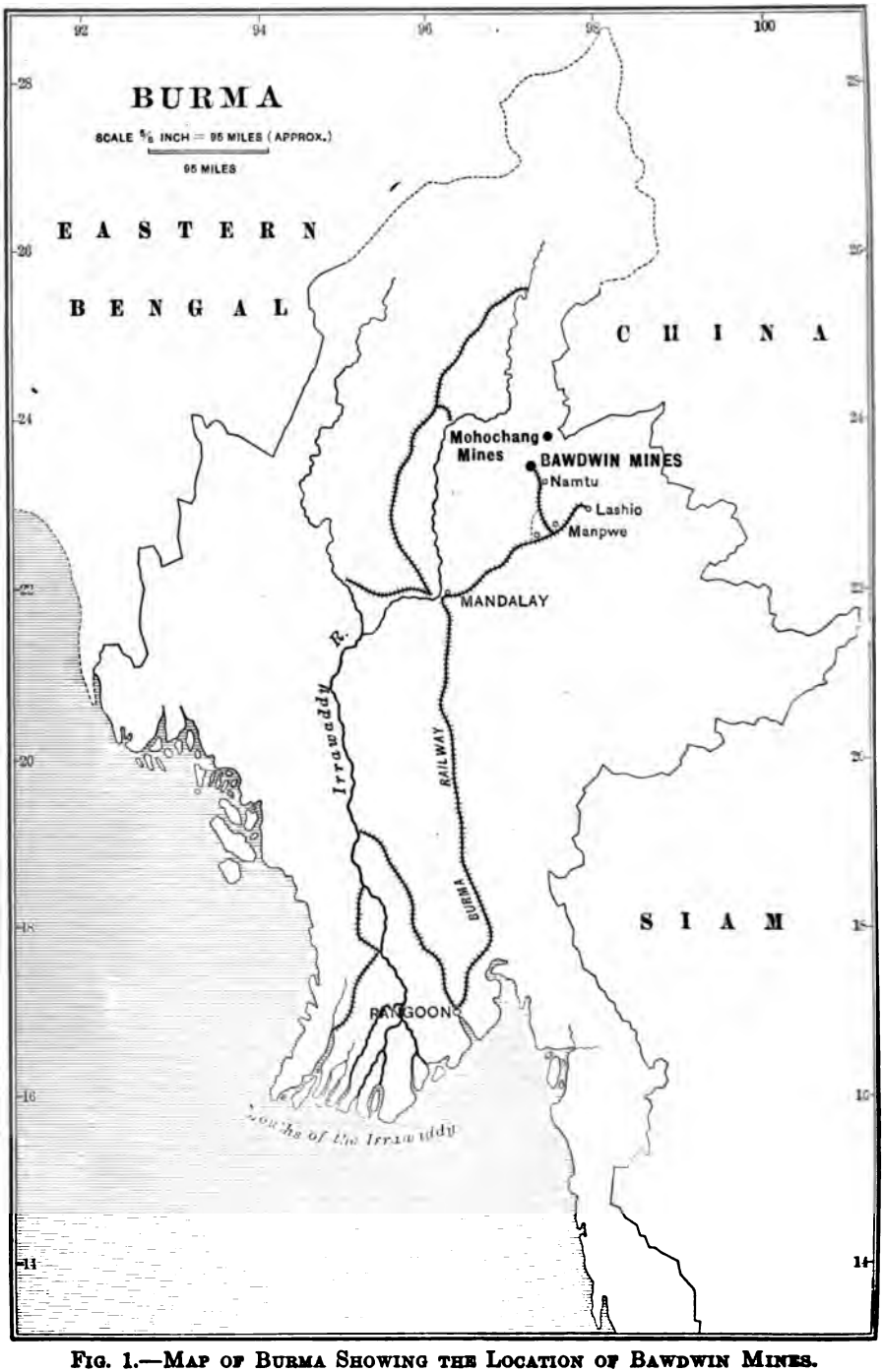

FIG. 1.—MAP OF BURMA SHOWING THE LOCATION OF BAWDWIN MINES.

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combination of diseases and lack of mining skill such efforts proved unsuccessful.

During the course of the long occupation by the Chinese an immense amount of work was done and the amount of rock removed would certainly total well over 1,000,000 tons. The attention of Europeans was first attracted in 1901 by the extensive slag dumps which lined the valley for a considerable distance north and south of the mine workings. During the removal of the slag, prospecting work was started on the orebodies themselves. The multitude of Chinese workings proved a drawback rather than a help as they were the cause of impounding great bodies of standing water which made the reopening of the mine a dangerous undertaking. It is only within the last 3 years that this difficulty has been overcome and the orebodies have been developed.

Topography

The State of Tawng Peng is extremely rugged and covered with a fairly dense jungle over its entire area. The present drainage has now produced the maximum of relief and further erosion without elevation of the land will tend toward a reduction in the inequalities. The ruggedness is intensified by the folding and possibly to some extent by faulting. The only cleared spots are the paddy fields along some of the stream bottoms and occasional small hillside farms. The inhabitants, Shans, Palaungs and Kachins, live in small villages of from 10 or 20 people to a few hundred. There is very little commingling of the different races. The Shan settlements are in the river bottoms; the Palaungs and the Kachins generally live well up the hillsides or along the high ridges. All communication is by mule tracks and footpaths which generally run along the ridges and descend to the valleys only for short stretches. All travel is done either on foot or by mules. Small Shan ponies are occasionally used.

The seasons are divided into a rainy and a dry period. The rains extend from the end of May to the beginning of November; the dry season embraces the balance of the year. The amount of rain is not excessive. It averages approximately 65 in. The months just preceding the beginning of the rains are hot and sultry; but in January and February the temperature drops to a point where fires become almost a necessity.

General Geology

The rocks of Tawng Peng are entirely of the pre-Cambrian and Paleozoic eras. The youngest strata belong to the Carboniferous period. The western and central portions of the State are mainly composed of mica schists and of severely folded, unfossiliferous shales and quartzites. There are also some fairly extensive granite intrusions of an undetermined age. These rocks are followed unconformably by Ordovi-

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cian sediments and are themselves therefore either Cambrian or pre-Cambrian in age. This series of rocks formed an ancient land surface along the eastern shore of which Paleozoic sediments were laid down. The sequence from the unfossiliferous basement to the Ordovician series is not an invariable one. Silurian sandstones are often found directly upon the unfossiliferous series. Their presence in this relation is accounted for either by faulting or overlap. At a few points along the borders of the pre-Ordovician rocks are found small exposures of rhyolite, tuffs, breccias and flows. The most important of these rhyolite bodies is that at Bawdwin. Overlying the rhyolites at Bawdwin, and probably conformably, is a great thickness of unfossiliferous sandstones and shales. For the present their position must be placed after the pre-Ordovician shales and quartzites and before the conformably overlying fossiliferous Ordovician marls and sandstones.

Topography and Surface Geology of Bawdwin

The topography of the immediate vicinity of the mine is similar to the general description given above for the State of Tawng Peng; extremely rugged, narrow valleys with precipitous inclosing hills. The hills rise in instances to more than 2,000 ft. above the valley bottoms. One marked distinction, however, between the immediate vicinity of Bawdwin and the remainder of the surrounding country is the fact that the Bawdwin hills are absolutely devoid of the jungle covering so characteristic of all the rest of the country (Figs. 2 and 3). The hills are, however, covered by long grass, 3 to 6 ft. high, which is largely burnt off toward the end of the dry season and springs up again during the rains. The explanation of the absence of jungle is the obvious one of removal by the Chinese for fuel for smelting and domestic purposes. The demand for fuel was evidently so great that the hills were stripped clean of all trees and 50 years' desertion has been able to effect only slight reforestation. The greatest extension of the cleared area is to the northwest, evidently due to the fact that the carry from that direction to the site of smelting operations was a downhill one. The hills surrounding Bawdwin to the west, north and northeast are crowned by an elaborate network of earth fortifications. These fortifications were undoubtedly constructed by the Chinese for the protection of the mine but at what period and against whom is not known.

The rocks at Bawdwin fall into two divisions: first, the volcanic rocks, rhyolite tuffs, breccias and flows; and, second, the overlying and underlying unfossiliferous sediments, consisting of sandstone, shale, and occasional conglomerate beds, the last only in the overlying sediments. No limestone has been found in these sediments.

On the west section of the 653-ft. level, the deepest point in the mine,

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sediments are present. These are presumably the underlying sediments, and appear to be unconformably overlain by the rhyolite tuff, although

FIG. 2.—VIEW OF BAWDWIN SHOWING RUGGED CHARACTER OF THE TOPOGRAPHY.

this cannot be definitely stated, as the only point where the contact has been observed is a fault contact. The rocks are sandstones (practi-

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FIG 3.—'CHINESE MINERS' VILLAGE WITH AN ANCIENT CHINESE STONE ARCH BRIDGE IN THE FOREGROUND

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cally quartzites) and shales. The sandstones are red to gray with small irregular quartz grains cemented together by sericite. The shales are light gray to purplish in color, compact with the original lamination largely obscured, although the bedding on a large scale is still observable in both the sandstone and shale.

The rhyolite in its various phases forms a rough band running northwest and southeast, but of irregular outline with numerous offshots which

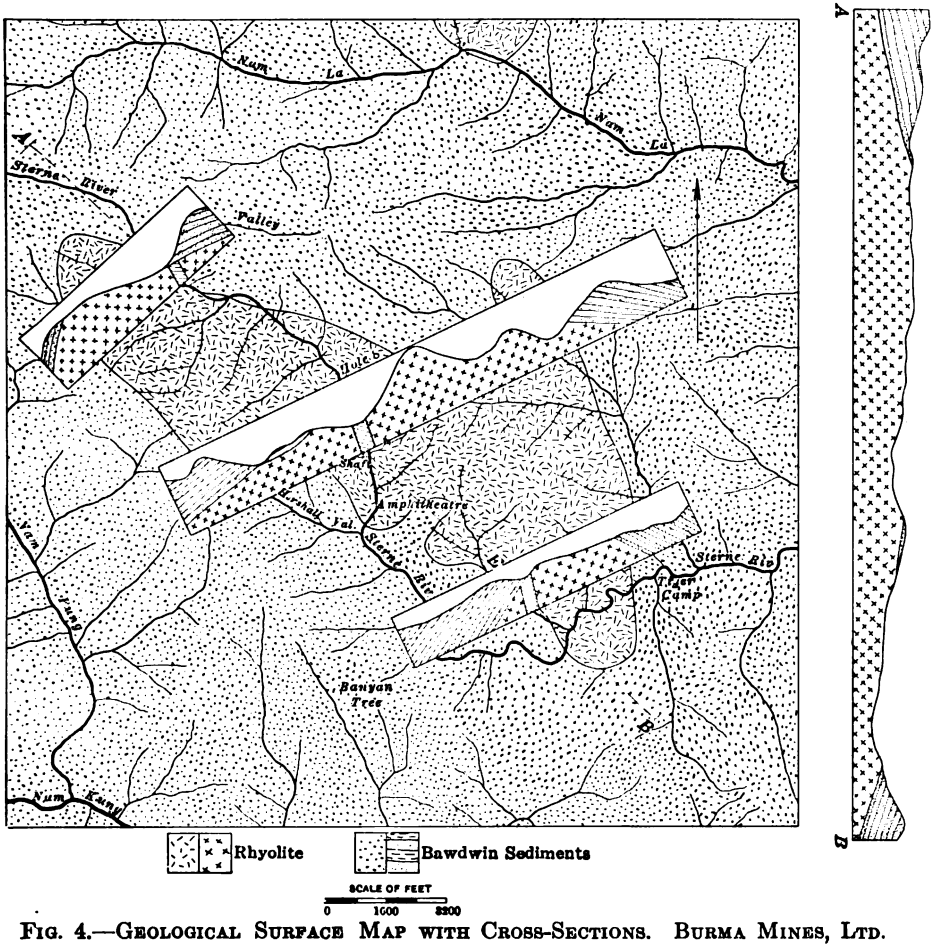

FIG. 4.—GEOLOGICAL SURFACE MAP WITH CROSS-SECTIONS. BURMA MINES, LTD.

are sometimes directly connected with the main mass and sometimes separated in outcrop by intervening, overlying sediments. The exposure of rhyolite does not represent its actual shape or extent but simply shows that portion from which the overlying sediments have been eroded. The general direction of the rhyolite marks the axis of an anticline, from the crest and flanks of which the sediments have been removed, exposing

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the underlying eruptive. The anticline plunges both to the north and south with a resultant dome formation, having the main axis of the dome, however, running northwest-southeast (Fig. 4). This anticline is the principal one of a series of folds which parallel the rhyolite band on the west. The folding is further complicated by the great amount of faulting which accompanied it. Almost the entire exposure of rhyolite is a tuff with local areas where the included fragments attain sufficient size to justify the term rhyolite breccia for the rock at that point. Over a great portion of the area the pronouncement that the rhyolite is a tuff depends entirely on microscopic examination and even then the determination is not an absolute one. A rhyolite flow with all signs of flow structure obliterated by later changes, such as could easily have taken place in a rock of such great age as the one under discussion, would be indistinguishable from the tuff. The gradual gradation, however, from an undoubted coarse-grained tuff to the fine-grained type strengthens the belief in the tuff theory. Other observers have stated the presence of thin flows of rhyolite among the tuffs with flow structure plainly observable. The writer has up to the present been unable to confirm this.

The rhyolite away from the zone of mineralization is a fairly hard dense rock. Its color varies from light gray through pink to a slight purplish color. The groundmass is in excess as a rule but occasionally the phenocrysts of quartz and feldspar become so abundant as to comprise the major portion of the rock. The relative proportion of the quartz and feldspars vary considerably in different specimens, but as a rule the feldspars are in excess. The phenocrysts of both quartz and feldspar attain at times large dimensions, especially the feldspar, which reaches an inch or more, measured parallel to the prism. The feldspars are never found unaltered but are generally entirely changed to some secondary mineral, in most cases sericite. The change of the feldspars to sericite is characteristic of the rhyolite outside of the ore channel, while in the ore channel the change has been largely to kaolin. The feldspars are largely prismatic in form, occasionally tabular. Owing to the change to sericite, remarked above, only the outlines of the feldspars are left and these in a good many cases are largely obliterated. For the same reason the type of feldspar present cannot be definitely determined, but it is believed to be almost universally orthoclase. The feldspars occasionally alter partially to calcite with some quartz and sericite which may indicate the presence of some lime-bearing feldspars.

The quartz phenocrysts are idiomorphic to hypidiomorphic. Embayments due to the eating away of portions of the crystal by the liquid groundmass are common. The crystals are often cracked and broken and the parts separated from each other. There are occasional traces of what was possibly a ferromagnesian mineral, probably an amphibole.

The groundmass shows no glass but is entirely devitrified and ap-

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pears to consist of submicroscopic feldspars and quartz. Small amounts of sericite in the groundmass probably resulted from the alteration of the feldspars. Zircon and apatite are present in small amounts as well as occasional grains of tourmaline.

The following rock analysis is typical of the rhyolite away from the ore channel. The alkalies were obviously about 4 to 5 per cent.

SiO_2	Al_2O_3	Fe_2O_3	FeO	CaO	MgO	K_2O	Na_2O	Loss	Total
70.20	14.20	4.85	....	1.70	0.70	not determined		4.00	95.65

The variety of rhyolite just described is typical of the major portion of the exposure. There is, however, a small section north of the upper end of Hershall Valley in which the rock is a coarse breccia. Again, other portions, as that comprising the mass of Mt. Teddy, consist of a multitude of small rock fragments in a groundmass of unresolvable volcanic dust. The nature of the rock comprising the ore channel has been so entirely transformed from its original character by crushing, silicification and other alterations consequent on the mineralizing solutions that it must be discussed separately. It should be noted, however, that the southwestern portion of the rhyolite tuff form the head of Hershall Valley to where it disappears under the sediments to the south of the Sterne River is a rock composed largely of small rock fragments, of broken crystals of quartz and feldspars and of great amounts of unresolvable volcanic dust, and even before the changes due to the faulting and mineralization was different in its character from the tuff characteristic of the rest of the rhyolite exposure.

The passage from the rhyolite tuffs to the overlying sandstone and shale is a gradual one and the exact boundary between the tuffs and sediments cannot be generally drawn. This is especially true along the southwest contact where the greatly altered rock of the mineralized zone abuts on the sediments. It appears that the basal members of the overlying sediments (coarse impure sandstones with large amounts of feldspars and rock fragments) were formed immediately after the deposition of the coarse tuffs and possibly before their consolidation. Thus the lower beds of sediments are simply the worked-over and somewhat sorted tuffs. Above these coarse impure sandstones, is a great thickness of well-bedded red and white sandstones, purplish and micaceous shales and occasional conglomerate bands. No fossils have been found in these beds. Conformably overlying these beds is a great thickness of Ordovician sandstones and marls with abundant characteristic Ordovician fossils. The geological sequence in the vicinity of the mine would thus appear to be as follows: A basement series of Cambrian or pre-Cambrian shales and quartzites followed by rhyolite tuffs and breccias. Conformably above the rhyolite is a great thickness of unfossiliferous sandstone and shale which is in turn conformably overlain by Ordovi-

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cian beds. The unconformity (as observed at other points in the Northern Shan States) between the Ordovician beds and the basal series is a very marked one, the lower beds having been severely folded and greatly eroded before the deposition of the Ordovician strata. The sequence from the rhyolite through the unfossiliferous sandstones and shales to the Ordovician beds is, however, a conformable one. It does not seem probable that the Bawdwin rocks correspond in time of formation to the unconformity between the basal beds and the Ordovician, especially as a portion of the interval was one of erosion and not of deposition. Consequently the Bawdwin rhyolites are most probably of early Ordovician or late Cambrian age while the basal beds should be assigned to early Cambrian or pre-Cambrian. The rhyolite probably lies unconformably on the basal beds.

Ore Zone

The orebodies occur in a wide zone of displacement (300 to 1,000 ft. across) in rhyolite tuff in which the faulting and crushing have been intense (Fig. 5). The zone has no well-defined boundaries but dies away gradually into hard undisturbed rock. As has been mentioned previously, the southwestern portion of the rhyolite area differs considerably in its lithological character from the remainder of the rhyolite. In this portion the tuff-like nature of the rock is much more pronounced, the constituents are considerably more numerous and variations in the type of rock are frequent. It is through this portion of the rhyolite that the zone of faulting passes, and, therefore, superposed on the original mixed character of the rock, are the great secondary changes due to the introduction of foreign materials and to the rearrangement of the minerals already present. The faulting extends through this type of tuff into the fine-grained tuff which comprises the major portion of the rhyolite area, but its effect there has been much less, due partially to the more homogeneous and compact nature of the rock and probably partially to a dying away of the severity of the faulting in that direction.

The rock of the ore channel varies from a fine-grained greatly silicified tuff consisting largely of introduced quartz to a rock largely composed of included fragments of other rocks, often dark blue or black angular fragments of shale. A type very characteristic of the neighborhood of the large orebodies is one consisting largely of a great number of large kaolinized feldspars set in a fine-grained groundmass. The alterations of the rock in the ore channel are so great that the original character is often almost entirely destroyed. The greatest single factor is the silicification which has penetrated all through the rock, replacing the groundmass, feldspars and included fragments. The feldspars are, however, generally altered to kaolin, the alteration being complete, the whole mass of the feldspars being changed. The change sometimes takes place

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to a mixture of calcite, quartz and sericite instead of to kaolin. Considerable sericite also forms in the groundmass and occasionally from included fragments. Large amounts of a light-green non-pleochroic alteration product, probably a form of chlorite, are found. At a few points in the ore channnel small portions which escaped the intense crush-

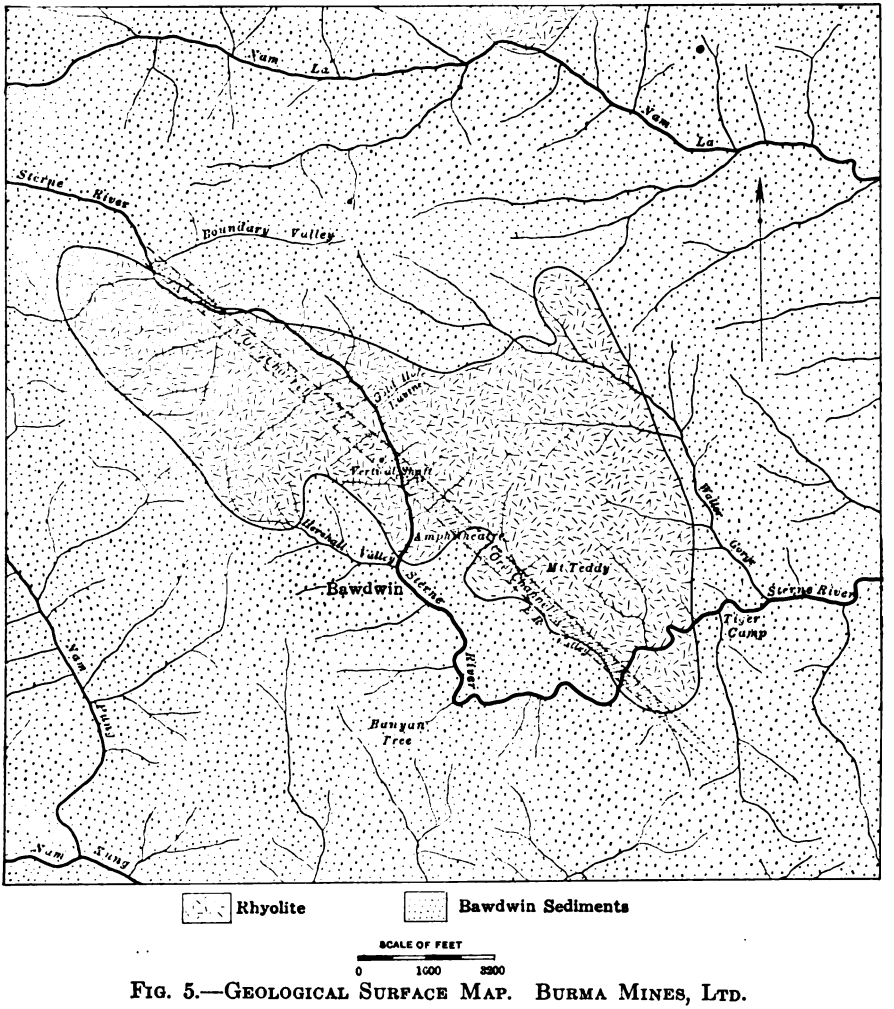

FIG. 5.—GEOLOGICAL SURFACE MAP. BURMA MINES, LTD.

ing and silicification and which retain the characteristics of the fine-grained tuff are preserved.

In a general way the following changes take place in the rock on approaching the ore channel. At a distance of 500 or 600 ft. the rock is a normal rhyolite containing considerable sericite and calcite due to ordinary secondary changes. On approaching the ore channel, the silica

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increases and the alumina decreases. The amount of sericite in the groundmass increases and microscopic cubes of pyrite become numerous. Still closer to the ore channel, chlorite becomes abundant and the alteration of the feldspars takes place partly to kaolin instead of to sericite. Finally, in the ore channel, itself, the rock is a mixture of quartz, sericite, chlorite and kaolin with some calcite.

The following are rock analyses from various points in the ore channel. Owing to the extremely variable character of the rock itself, a tuff with included fragments, it is unsafe to draw any conclusions by a comparison of these analyses with the one given as typical of the unaltered tuff.

	SiO_2	Al_2O_3	Fe_2O_3	FeO	CaO	MgO	(K_2O Na_2O)	Loss
(1)	77.20	8.28	0.71	1.09	1.07	0.48	2.55	
(2)	71.60	8.74	2.67	4.35	1.30	1.19	2.69	
(3)	80.40	10.25	0.46	1.28	1.40	0.37	2.91	
(4)	74.65	9.05	5.35	.....	3.05	1.70	determined	4.48
(5)	83.65	8.00	2.85	.....	1.45	0.45	determined	0.94

(1) Close to footwall of Chinaman Lode.
(2) Several hundred feet from hanging wall of Chinaman Lode.
(3) In Chinaman Lode—a block of country rock with large kaolinized feldspars and included in the lode.
(4) 300-ft. level—south of Shan Copper Lode.
(5) Close to surface on hanging-wall side of lode.

Lead, zinc and sulphur bring the totals of the above analyses up to approximately 100 per cent.

Orebodies

The metals of commercial importance found at Bawdwin are zinc as sphalerite (ZnS), lead as galena (PbS), copper as chalcopyrite ($CuFeS_2$) and silver whose manner of occurrence is not definitely known but which is probably present largely as argentite in submicroscopic mixture with the galena.

The ore occurs as solid masses of sulphides with practically no admixture of gangue; as veins; as interlacing seams forming small stockworks; and as an impregnation of the country rock. The gangue when present is quartz, calcite or country rock (rhyolite tuff). The principal orebody, known as the Chinaman Lode, is at present developed for about 1,200 ft. in length and varies in width from a few feet to over 100 ft., maintaining on some levels an average width of 50 ft. for over 1,000 ft. along the strike (Fig. 6). It is primarily a zinc-lead-silver orebody with small amounts of copper along the edges. Some of the diverging seams and faulted portions of the Chinaman Lode are, however, variable in their metal content, showing rapid alternations from zinc-lead to copper

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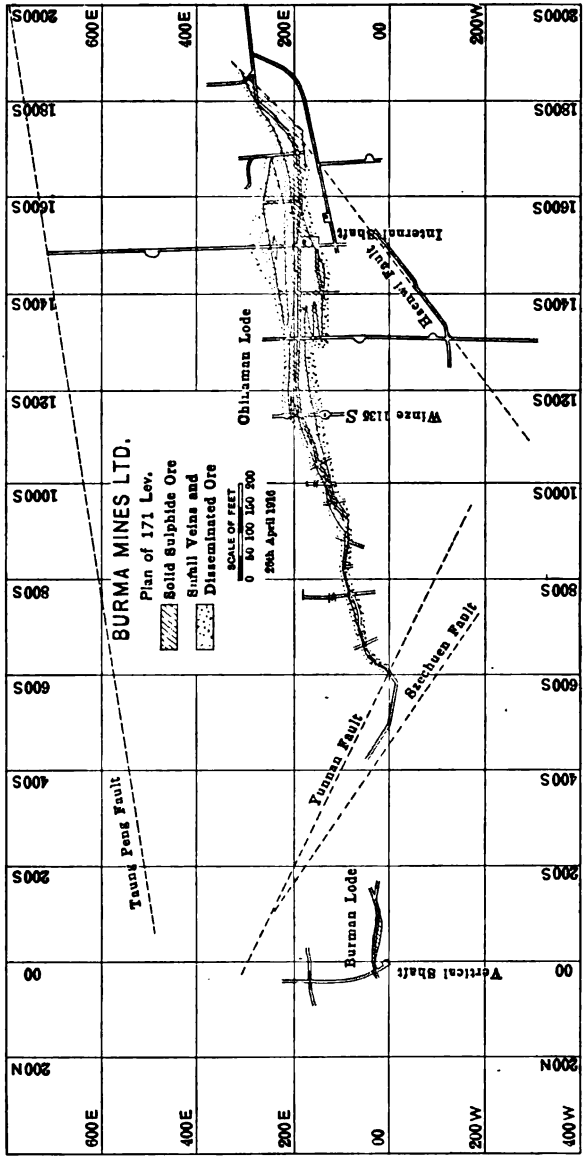
ore and *vice versa*. These changes take place both horizontally and vertically, sometimes as a gradual transition, sometimes marked by a fault plane. The largest single body of copper occurs on the 300-ft. level,



north of the Chinaman Lode. It is about 130 ft. long with an average width of 25 ft. running 14 per cent. copper, 7 oz. silver, and with only small percentages of zinc and lead on this level. It grades, however,

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about 75 ft. above into zinc-lead ore and does the same at 30 ft. below. This lode is probably the faulted northern extension of the Chinaman.

The orebodies thus far opened up occur entirely in the altered silicified rhyolite tuff. The mineralization extends to a slight degree into the overlying sediments. It is possible, however, that this mineralization in the overlying sediments belongs to a later period than that in the tuff and is more of the nature of a slight retaking into solution of the minerals in the tuff and redeposition in the sediments by ground-water circulation. The nature of the orebodies in the underlying sediments is as yet unknown. It is believed that they will probably be more restricted as to size and be more in the nature of fault deposits, with replacement of the country rock assuming a subordinate rôle, the opposite of the conditions in the tuff.

A cross-section through the Chinaman Lode shows a central core of solid zinc-lead ore, with the zinc generally but not invariably in excess of the lead. On both sides of this central core are alternating bands of solid ore and heavily mineralized tuff. These bands parallel the main body in strike and dip but are not persistent themselves, coalescing and pinching out and in reality forming a sort of stockwork. The bands are generally high in lead and comparatively low in zinc. A slight percentage of copper is generally found on their edges. From both sides of these bands the mineralization extends far out into the tuff, gradually merging into barren rock. Occasional seams and patches of ore are found at considerable distances. There is no sharp boundary between mineralized and unmineralized country rock as a general thing, although this condition is approximated in a few places by fault planes. Pyrite is found to a limited extent in the main orebody, but attains a much greater abundance along the edges where it is disseminated all through the rock as a multitude of very small (microscopic) cubes. The central core of solid ore in the Chinaman Lode attains at points a thickness of more than 80 ft., and on some horizons maintains an average width of 55 ft. for over 800 ft. in length and, as stated above, an average of 50 ft. for 200 ft. additional. In this core the gangue is generally very finely disseminated quartz grains. The extreme richness in metal content of the orebody is best shown by the fact that a block roughly 800 ft. long by 600 ft. deep by 30 ft. wide contains about 1,750,000 long tons with an average value of approximately: Ag, 30 oz.; Pb, 31 per cent.; and Zn, 29 per cent. A theoretical block of the same size of solid galena and sphalerite with equal amounts of Pb and Zn would contain approximately 2,300,000 long tons. Thus the block in the mine is over 75 per cent. solid lead and zinc sulphides. The accompanying cross-section (Fig. 7) through the orebody shows that this solid sulphide core has a high dip to the west in the upper levels and turns gradually over to a high dip to the east in the lower levels. A gradual thinning of the orebody takes place on approaching the underlying sediments. This is probably due to the greater effect of the fault-

ing on a heterogeneous rock like the tuff than on the sandstone and shale. This wider zone of shattered rock in the overlying tuff and, in addition, the much more easily replaceable character of the rock, would result in a considerably wider area of mineralization.

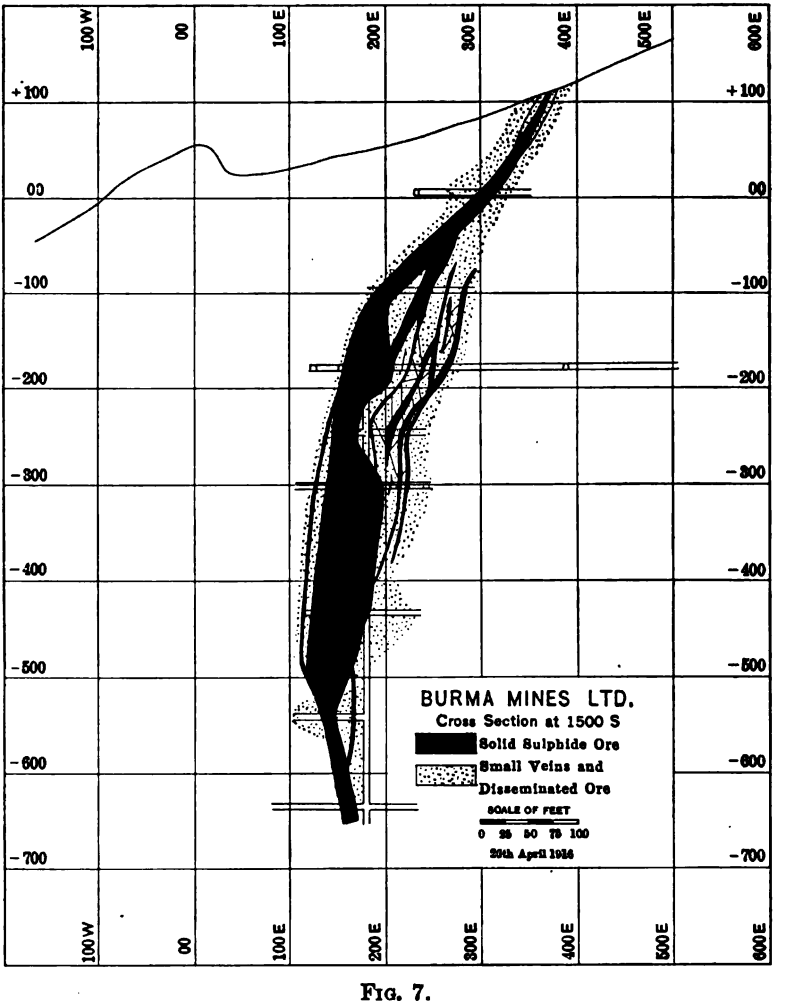

FIG. 7.

The original condition of the upper portion of the orebody is largely concealed by the work of the Chinese, but there appears to be a decided thinning of the orebody on approaching the surface. The outcrop, in fact, when considering the size of the orebody below, is scant, but this is probably partially accounted for by the oxidizing and subsequent leaching of the surface ores. At several points sphalerite and galena outcrop on the surface, but as a rule the surface ores have been largely

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leached and the outcrops are marked by a wide zone of soft decomposed rock colored by iron oxides, copper carbonates and, to a lesser extent, by oxidized lead ores. This gossan for considerable stretches runs 3 to 4 oz. in silver and about 5 per cent. in lead. It is quarried at one point for a siliceous flux for smelting purposes. A characteristic assay follows: Ag, 3 oz.; Pb, 5.8 per cent.; Zn, 0.7 per cent.; SiO_2, 58.6 per cent.; Fe, 17.7 per cent. The depth of the gossan is extremely variable but very rarely is more than 50 ft. At approximately this depth it is succeeded by a zone of secondary copper sulphides, principally chalcocite with some bornite. The chalcocite occurs largely as a replacement of sphalerite. The secondary copper ores cannot be considered as occurring as a well-marked zone over the whole orebody, but simply at a few favorable places, at some points stretching well up into the oxidized ores; at others, well down into the normal sulphides. This secondary copper is

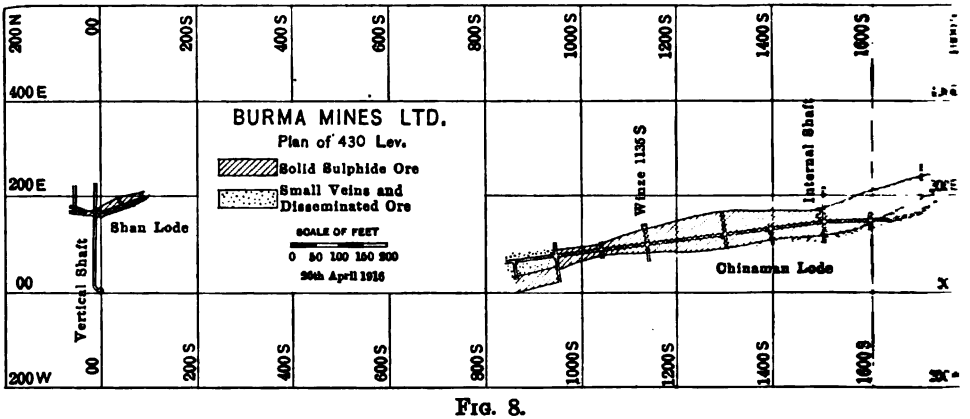

FIG. 8.

not present in large enough amounts to be commercially important. The points at which the chalcocite occurs are about 100 ft. above the present water level, but as erosion has been rapid it probably marks the water-table level at a not very distant period. The zinc and lead sulphides which, as remarked previously, at some points outcrop on the surface, extend down to the greatest depth yet reached, about 725 ft., with practically no change in their character or relationship to each other. The normal zinc-lead ore of the mine is an extremely intimate mixture of sphalerite and galena; the sphalerite, which is the older, being of a very fine granular structure, the grains varying from about 0.015 to 0.16 mm. in diameter. The galena forms around the separate grains of sphalerite, and as thin filaments running through the zinc. The ore grades off in both directions from this fine mixture, toward the zinc end to solid masses of soft, earthy sphalerite with occasional stretches of hard, dark sphalerite probably rather high in its iron content, and toward the lead end to masses of pure, coarsely cubical galena.

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The silver content rises and falls consistently with the lead and independently of the zinc, indicating that the silver is largely contained in the lead. The presence of copper even in small amounts destroys the silver-lead ratio, as the copper present in the small veins adjoining the main orebody carries a high silver content (Fig. 8). As a general statement, it can be said that 1 per cent. of lead carries 1 oz. of silver. Silver minerals have not been detected in any form but, as stated above, it seems most probable that the silver is present as argentite intimately associated with the galena.

The accompanying assays show the relations between the silver and the lead and also represent typical assays of high-lead, high-zinc, zinc-lead and lower-grade ores.

	Ag, Oz.	Pb, Per Cent.	Zn, Per Cent.
Silver-lead Ore	47.6	50.4	19.2
	54.4	50.0	24.1
High-zinc Ore	11.0	11.2	38.8
	15.5	13.6	40.7
Zinc-lead Ore	38.5	33.9	36.3
Second-grade Ore	11.8	22.5	7.0
	18.0	20.0	12.5

Faulting

The faulting along which the ore occurs was not concentrated in one fault plane, but occupies a zone reaching in places well over 1,000 ft. in width. Along this zone a multitude of faults of varying strike and dip—some normal, some reverse—were developed. The faulting is extremely complicated and of such varying strike and dip that a division into rigid systems cannot be made. The majority of the faults can be traced only for short distances and are either displaced by later faulting or die out. Roughly, however, two main fault systems can be determined, one with strikes about north and south and with high dips to either the east or west; and a second series with strikes varying from N35W to N75W and with high dips generally to the southwest. The displacement due to most of the faulting has been very slight, as is evidenced where faults have cut the orebodies. The displacement in these intersections is generally so small as to be barely noticeable. A few dominant faults, however, which can be traced for long distances, have caused severe dislocations. One such fault cuts off the Chinaman orebody on the southern end. So extended was the movement that the ore south of the fault has not yet been located. There are, in addition to the above faults, a great multitude of smaller ones of all strikes and dips but of subordinate importance. The faults as a rule are marked by varying thickness of soft gouge and rounded rock fragments. The gouge is often colored black by the minute sulphides contained in it. Faults when passing through the solid bodies of sulphides have produced smooth

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slickensided surfaces, often with a brilliant polish which may be modified by etching. This is especially true of fault planes in the high-grade galena ore.

The faults just described appear to be later than the mineralization, although it is possible that some of the faults that strike north and south along the edges of the orebody are pre-mineral and acted as bounding walls for the main ore deposition. It is also most probable that some of the small veins branching off from the main Chinaman orebody represent ore deposits along fault planes. That these smaller veins are contemporaneous with the Chinaman Lode is proved in some cases by direct observation of the points where they merge into it. In the large replacement orebody, the Chinaman, the practically complete replacement of country rock by sulphides has entirely obliterated the original character of the ground. This orebody represents, however, a replacement in a wide faulted and crushed zone rather than a replacement that has spread out into the walls from a central fault plane. Many of the faults that are later than the main orebodies contain ore due to drag and, in some cases, ore probably formed by solution from the original orebodies and by redeposition along the fault planes which have naturally served as water channels.

The plan of the 171-ft. level (Fig. 6) shows some of the more prominent faults. The large amphitheater, directly above the Chinaman Lode, is bounded to a great extent by faults, and partly owes its existence to land slips along these planes. Its precipitous sides have probably been slightly accentuated by Chinese work but the amphitheater taken as a whole is probably almost entirely due to natural causes.

Underground Waters

As is to be expected in a tropical region of fairly heavy rainfall, the ground-water level is close to the surface. The ore channel strikes diagonally across the trend of the ridges and the ground-water level was tapped by the company's drainage tunnels only a few feet below the valley bottoms (Fig. 9). There did not, however, appear to be a completely saturated zone below the water level. Diamond-drill holes at a greater depth than any of the workings occasionally go completely dry, and water pumped into them entirely disappears. This experience was met even in cases where the holes were cemented practically all the way down. The great number of old workings and the extremely shattered condition of the ground with numerous fault planes along which water circulates freely, tend to obscure the natural underground circulation. A heavy rainfall has an almost immediate effect on the amount of water in the upper levels, and a portion of the surface water probably drains down along faults to the lowest levels. Two analyses of water are given here-

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with: No. 1 from a shaft sunk in the Chinaman Lode; No. 2 from a drainage tunnel which at the time the sample was taken was about 6,500 ft. from the orebody and probably represents with fair accuracy the underground water of the region away from the influence of the orebody.

FIG. 9.—SETTLEMENT AT THE PORTAL OF THE TIGER TUNNEL. A DRAINAGE AND WORKING TUNNEL 7,000 FT. LONG.

Parts per 1,000,000

No. 1		No. 2	
SO_3	2,160	$CaCO_3$	138
CaO	227	$MgCO_3$	78
Fe_2O_3 and Al_2O_3	273	Na_2SO_4	19
SiO_2	35	Fe_2O_3 and Al_2O_3	13
MgO	227	SiO_2	136
Pb	30	Undetermined	9
ZnO	1,179		
NiO and CoO	250	Total	393
Organic and volatile matter	30		
Cl	Tr		
Undetermined	21		
Total	4,434		

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No. 1 is a strong sulphate water with an extremely high content of zinc. The large amount of nickel and cobalt in solution is remarkable inasmuch as the sulphides of these metals have not been definitely recognized in the orebody. The cobalt arsenate, erythrite, however, forms rapidly along some exposed faces. This sample was taken at a point about 400 ft. below the surface, the lowest point in the mine at the time, and it does not appear probable that the large amount of zinc is due to any extent to solution from exposed faces on the upper levels, because, if this were true, copper should also be present, as the descending water of the upper levels has a high copper content. The entire absence of copper is noteworthy and appears to indicate either that the body of underground water is virtually stagnant and its mineral content is derived from the immediate vicinity, or that the contained copper in the water above is redeposited before reaching the lower levels. A large number of the faults have impervious gouges and probably break the ground up into blocks, some saturated, some practically dry.

Minerals

The list of minerals found at Bawdwin is not extensive. The principal ones follow:

Quartz as phenocrysts in the rhyolite; as a large portion of the cryptocrystalline groundmass; as a secondary product from the alteration of other minerals; as a later introduction in veinlets; and as a product of general silicification. It appears in veins in massive form, and is hard and compact, but it may also yield well-formed pyramids in cavities. It may constitute ribbon quartz with lamellar structure. It may even be powdery. It can be detected as very fine grains, intimately associated with the sulphides.

Feldspar, probably largely orthoclase, occurs in crystals both large and small. Often only fragments of crystals appear, and both these and the well-bounded ones are generally badly decomposed. With quartz it forms the groundmass of most of the eruptive rock.

Calcite is abundant as a secondary mineral, especially as a replacement of other minerals, notably feldspar. It may be one of the gangue minerals and favors the copper ore in which relation it often composes the entire gangue.

A ferromagnesian mineral, probably an amphibole, has suffered such extreme alteration as to be almost entirely destroyed.

Barite appears in large masses as vein-filling on the surface and as large weathered blocks. It has not been observed underground. Wherever seen, it is always in association with the sediments, never with the rhyolite.

Siderite is common as a gangue in some of the smaller veins but is of

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subordinate importance. Iron-stained calcite may sometimes be taken for it.

Apatite and *zircon* are fairly common as small crystals disseminated through the rock.

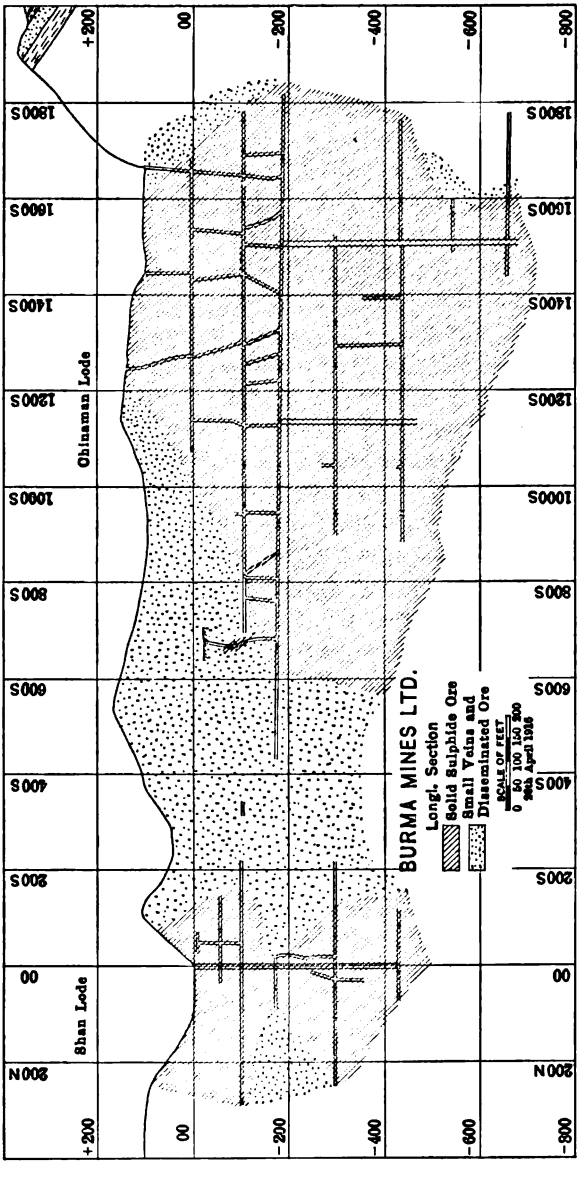

FIG. 10.—LONGITUDINAL SECTION OF OREBODY.

Tourmaline, occasional grains in tuff.

Biotite, occasional shreds, probably secondary.

Sericite is the predominant alteration product of the groundmass,

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and away from the ore zone also of the feldspars. It occurs as a mat of slender microscopic crystals.

Kaolin is the principal alteration product of the feldspars in the ore zone.

Chlorite. A light-green, non-pleochroic, alteration product is common in all types of the rock, and is probably a form of chlorite.

Galena occurs as coarse cubical galena in quartz stringers; as impregnations in the country rock; as replacements of silicates or quartz; and as a mixture in all degrees of intimacy with sphalerite and to a lesser degree with chalcopyrite.

Cerussite, as well-formed crystals in vugs in galena. It is present in considerable amounts as low as the 171-ft. level and to a lesser extent further down. It occurs also in the altered outcrops.

Anglesite appears in relations similar to those of cerussite.

Pyromorphite, as well-formed crystals in relations similar to those of the two preceding minerals.

Sphalerite occurs as large solid masses; as an impregnation of the country rock often finely disseminated; as replacements of other minerals, especially of the groundmass and feldspars of the rhyolite; as an intimate mixture with galena and occasionally with chalcopyrite, and as large masses of earthy sphalerite. The normal oxidized zinc minerals have not been observed, but goslarite forms rapidly and in large amounts along the walls of the drifts in the upper levels.

Chalcopyrite occurs in the smaller veins and along the edges of the Chinaman Lode and as the principal ore of the Shan Lode on the 300-ft. level (Fig. 10). Is often associated with galena and sphalerite but the relation is never so intimate as between the galena and sphalerite.

Malachite, azurite and native copper occur in small amounts in the oxidized zone.

Chalcocite appears as a secondary product above the water level; largely as a replacement of sphalerite, the replacement being generally not complete, so that a core of sphalerite is surrounded by chalcocite, usually of a soft earthy variety.

Bornite occurs in small amounts with the chalcocite.

Pyrite occurs in small amounts in the orebody as the last sulphide deposited. Great amounts of it lie along the edges of the orebodies, largely as a multitude of microscopic cubes disseminated all through the rock.

Hematite and *limonite* are found in the gossan above the orebodies.

Erythrite constitutes a pink incrustation along the drifts.

Nickel and cobalt sulphides are present but have not been identified.

Genesis

The orebodies at Bawdwin have been formed by the metasomatic replacement of the rhyolite tuff by sulphides deposited from hot solutions, which rose from below along an intensely crushed and sheared zone. From

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the data at hand it is not possible to make any definite assertion as to whether the mineralizing solutions are themselves a late phase of the rhyolite activity or not. Whatever evidence there is, however, appears to indicate that the rhyolite is not responsible for the metal-bearing solutions. About 25 miles northeast of Bawdwin, at Mohochaung, there are orebodies in sandstones and shales which are entirely similar to Bawdwin as regards the character of the ore itself. The beds in which they occur are probably roughly contemporaneous in age with the Bawdwin sediments. There is no rhyolite present at this point. Granite is, however, present about 3 miles to the west of Bawdwin and at roughly the same distance to the west of Mohochaung. The granite is a coarsely crystalline muscovite-biotite granite. It is a reasonable presumption to consider the granite mass as extending below the orebodies at both Mohochaung and Bawdwin, probably at a great depth below the surface, and it may be to the influence of the granite that the mineralizing solutions should be ascribed. This is, however, simply a hypothesis with no actual facts to support it.

The fault zone at Bawdwin passes through all the varying types of tuff, from the type composed largely of included fragments of other rocks to the fine-grained homogeneous rhyolite tuff. The effect of the kind of rock upon the ore deposition has been marked. The fine-grained homogeneous tuff is practically barren, the fine-grained highly silicified rock composed of fine rock fragments contains small bodies of ore along fault planes but with practically no replacement of the rock itself. With an increase in size of the fragments composing the rock there is an increase of replacement by metallic sulphides. The type composed of large numerous feldspar crystals set in a fine-grained groundmass is distinctly the most favorable rock and is, in fact, the rock in which all the large zinc-lead bodies are found.

The favorableness or unfavorableness of the various types of rocks toward ore deposition are due to several reasons.

1. The compactness or, inversely, the porosity of the rock.

2. The nature of the rock constituents as regards their ease of attack by solution and replacement.

3. Effect of the faulting on the rock—whether the rock gives way along a few prominent planes or whether the movement and consequent crushing is spread over a wide zone.

The coarse heterogeneous tuff, especially the one with large feldspars, was due to the nature of the rock itself which was not a compact one and consequently allowed an easy entrance of mineralizing solutions. The feldspars offered an easy first point of attack for the solutions and the rock itself, owing to its mixed character, suffered crushing and faulting over a wide area. All the above factors become progressively less active as one passes toward the fine-grained homogeneous type of tuff.

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The rhyolite tuff was greatly shattered and crushed along the zone of faulting, probably to a much greater extent than is now observable, as the rock has been reconsolidated by later silicification. The ascending solutions permeated all through the crushed mass, and along the zone of greatest shattering practically entirely replaced the original rock with sulphides. The feldspars which were probably the first point of attack were either replaced by sulphides or altered to sericite or kaolin. The alteration to kaolin may, however, be due to later descending solutions carrying sulphuric acid, although somewhat mitigating against this theory is the fact that the kaolin alteration persists several hundred feet below the water level. After the feldspars, the fine-grained groundmass was attacked, and finally the quartz. Deposition also took place in a subordinate degree in open spaces, as is evidenced by banding and rough comb structure and by drusy and botryoidal forms of sphalerite. Vugs and openings along fault planes are present in all parts of the mine, being, however, considerably more numerous in the upper levels where they are probably largely of recent origin. In the central core of solid ore none of the original country rock is left; where any gangue exists it is a very fine-grained quartz probably introduced with the sulphides. The sulphides were deposited in successive stages which, however, probably overlapped. The sphalerite is the oldest, followed by chalcopyrite and galena and finally by pyrite, the iron sulphide being to a great extent the last, although it was probably introduced in small amounts all through the period of the ore deposition. The origin of the chalcopyrite was contemporaneous with the introduction of considerable calcite, the latter mineral being a characteristic gangue of the copper ore. The relative ages of the sulphides as outlined above are shown not only in their relation to each other but also to some extent by their general location. The zinc forms the central core with the lead and copper along the edges of the zinc. This generalization is true only to a certain degree as the successive movements which opened up new channels for ore deposition along the edges of the already existing bodies also shattered these bodies. The mixture between the lead and zinc is, however, such an extremely intimate one that it is difficult to assume that it is formed solely by galena introduced into openings, however minute, in a shattered body of sphalerite. The relation of the two minerals to each other as shown in thin sections clearly indicates that they were not deposited simultaneously but that the lead is distinctly younger than the zinc. It would appear that the intimate relationship may be partly due to a replacement of the sphalerite by the galena. The galena having been introduced along minute cracks in the sphalerite gradually spread out from these cracks by replacement of the zinc, thus reproducing on a microscopic scale the method of formation of the orebody as a whole.

It does not appear that changes of any magnitude have occurred

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in the condition of the orebodies since their original formation. There is no distinct zone of secondary enrichment (barring the small amounts of chalcocite), a fact best indicated by the silver values which show no marked change from the upper part of the orebodies to the lowest levels, several hundred feet below the water level. There is a gradual increase of silver values on descending, but this is entirely due to a corresponding increase in lead. As has been stated previously, the lead-zinc ore maintains the same character from the surface to the lowest levels, varying somewhat in the proportion of one to the other but not due to any later rearrangement. As stated above, there is a gradual increase in silver and lead on descending. This is not, however, at the expense of the zinc, which also increases slightly, but is due to the almost entire disappearance of gangue minerals and of low-grade patches in the solid core.

The age of the orebody itself is extremely difficult of determination. The overlying sediments have been eroded from directly above the orebody and whatever patches of ore have been found in them at other points are of small size and may have been formed from already existing bodies in the tuff. This is not, however, certain and the barrenness of overlying sediments may be due to their unfavorable character as regards replacement by sulphides. If the overlying sediments are later than the ore deposition, it would mean that the orebodies are early Ordovician or late Cambrian. If the sediments, however, antedate the period of ore formation it becomes impossible to determine the age of the orebodies.

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Features of the New Copper Smelting Plants in Arizona

Discussion of the paper of A. G. McGregor, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1257 to 1279.

L. D. Ricketts, New York, N. Y.—The advance which has been made in smelting has been in the line of cheaper cost of handling, due to larger units and decrease in losses. At the International smelter, Mr. McGregor designed everything to handle the ore under cover. But we do find an unaccounted for loss of 0.7 per cent. of our copper, which is serious. Mr. Wallace has made a suggestion that I think metallurgists ought to know of and think about. He made it years ago—and that is that one of the chief losses is undoubtedly in the charging and discharging of the calcine cars. His idea is that possibly the dryers and heaters might be put immediately over the reverberatories so that they could discharge in conveyors. They could feed on both sides, and thereby the dust raised by discharging in the cars from the hoppers could be overcome.

E. P. Mathewson, Anaconda, Mont.—I might say a few words on the plants we have visited so far. I was particularly struck with the cleanliness of the plants, which is next to Godliness. I noticed also that the skull-cracker arrangement they have for handling their converter slag is very good. And they have paid considerable attention to ventilation, and are taking steps to take care of the health of the men. "Safety first" has been well looked after in all the plants we have visited so far. The steel structures are right up to date, and the buildings are of the most modern type, and everything is conveniently arranged. I think the designers of the plants in the Southwest are to be congratulated on what they have done.

The Basic-Lined Converter in the Southwest

Discussion of the paper of L. O. Howard, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1539 to 1543.

The Chairman (Walter Douglas, New York, N. Y.).—I presume, gentlemen, that Mr. Howard's experience with the Great Falls basic-lined converter has been as regards the tonnage produced from a single lining, perhaps, a world's record. About 6 months ago Mr. Howard shut down and tore out what was left of the first basic-lining which was installed at the Old Dominion, the lining having produced something over 70,000,000 lb. of copper. This was due probably not only to the very excellent grade of the matte which was produced by the smelter there,

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but also to the fact that after the tonnage began to reach record proportions Mr. Howard lived with that lining night and day, and watched it most attentively. May we hear from any of the gentlemen present with reference to this paper of Mr. Howard?

E. P. MATHEWSON, Anaconda, Mont.—I would like to say a few words in regard to the basic-lined converter. I noticed at El Paso that they had not adopted the Great Falls type of converter, but had stuck to the Peirce-Smith type of horizontal converter. This type has certain advantages over the Great Falls type and certain disadvantages. We have in the Northwest still larger furnaces of the Great Falls type, many of them 20 ft. in diameter. These furnaces are doing good work, due to the advantage that the Great Falls converter possesses. The sole disadvantage, I think, is that in tilting the furnace, if care is not taken by the operator, some of the tuyères are under a greater depth of matte than others; there is liable to be splashing and loss of blast from this same cause. In the horizontal type the tuyères are at the same level, and the level can be adjusted to meet the blast available, so that very little splashing results. There is a disadvantage on the other hand with the horizontal type converter in that it is hard to distribute the silica evenly over the matte without making several openings. This difficulty is not met with in the Great Falls type, as the silica may be dumped without care, and will distribute itself evenly over the matte. But it will be very interesting, as I say, to note the comparison between these two types on a similar grade of matte. As I understand, the matte used at El Paso is about the same grade as used at the other points in the Southwest and about the same as in the Northwest. We have records of our furnaces extending back over 2 years, and I think we have forgotten that these linings have any life limit. A little care—which means that the operator must not be careless—keeping the heat down to reasonable limits, and you will not have difficulty with the lining. The main part of the lining we seem to be able to make last indefinitely, care being taken to keep the silica low in the slag.

KUNO DOERR, El Paso, Texas.—I am sorry to say that the new converter at our El Paso plant was put into use only about 10 days ago, due to the fact that we have had difficulty in getting magnesite brick to line it with. This converter is of the P-S type, 13 ft. in diameter by 30 ft. long, whereas our old Peirce-Smith converters are only 10 ft. in diameter by 26 ft. long. In the few days in which the new converter has been in operation we have observed a great improvement in it over the operation of the old converters. It has a large stack, or gas outlet, and makes practically no splash at all—that is, what splash there is goes back into the converter itself, whereas in our old converters we have had a great deal of trouble from accretions in the stack and hoods, and there has

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been a very large amount of cleanings to take up and resmelt. Thus far we have had none of this in operating the new converter.

Our average blister production per lining in the 10-ft. Peirce-Smith converters was about 7,000 tons. We have never had to reline the bottoms of any of our converters, and our repairs usually consist of relining with new brick in the tuyère zone only. A patch made without shutting down the converter except between blows is not considered in our records as a new lining, but any shutting down and cooling off of the converter for repairs terminates our production record per lining. Thus we actually get much more than 7,000 tons of blister on the average number of brick in a converter lining. Our mattes vary somewhat, but will not run much under or much over 40 per cent. copper.

One source of trouble with our old converter installation was the fact that the converters were not served by electric cranes, but all the matte from the furnaces to the converters was transported in open ladles over our industrial railway, and the run from furnaces out onto our slag dump and back again, to get the necessary height, was about ½ mile. The consequent cooling, and pouring of matte into converters through launders caused some of the troubles we had from accretions and excessive amount of cleanings; but the greater diameter of the new converter, larger gas outlet, and better hood are principally responsible for the improvement we have accomplished.

Mr. Howard's exceptional record of long life of linings, I imagine to be partly because he has had the pick of such converter silica as he has wanted to use, whereas we, being in the customs smelting business, have had to take what we could get.

You will thus observe that we have no production records as yet that would be of any interest concerning our new converters, but next year at this time we hope to be able to give a good account of our big converters. Our 10-ft. Peirce-Smith converters consumed about 9,000 ft. of free air per minute at, say, 12 lb. pressure, and the last records I secured before leaving El Paso show that our new 13-ft. converter is taking 20,000 ft., and that means that it is doing business.

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Determination of Dust Losses at the Copper Queen Reduction Works

Discussion of the paper of J. MOORE SAMUEL, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 114, June, 1916, pp. 1079 to 1098.

THE CHAIRMAN (WALTER DOUGLAS, NEW YORK, N. Y.).—Perhaps there is no problem that causes the modern metallurgist more worry than the question of unaccounted for loss. He has not the advantage of the metallurgist of the early days here, who could sleep peaceably with the settled conviction that there were certain copper mineral constituents of the ore which were volatile, and there was therefore nothing to worry about. The slag contents and volatilization accounted for the balance of the copper that went in the top of the furnace after the weighing of the bullion. There have been great strides in recent years toward reducing this loss. Perhaps the Anaconda company is the one we have to thank for the original pioneer work along the lines which we are all today following. We would be glad to hear from some gentlemen present with reference to their experience with this question of dust losses.

E. P. MATHEWSON, Anaconda, Mont.—You mentioned the Anaconda company, Mr. Chairman. I would like to say something about what we have been doing there, and the troubles we have been having. We have a tremendous volume of gas to handle. We adopted the system of large chambers, lessening the velocity of gases, and incidentally cooling the gases. Unfortunately in one sense and fortunately in another, we were compelled to increase the amount of gas handled in the flue system, and the results were then not so good. Still, we get a very good recovery. Our system of taking a sample is similar to the one described by the writer of the paper. We have tried all kinds of apparatus for getting a fair sample. We divided the area of the flue into imaginary squares, and have taken samples for a certain period of time from each square; and have used the asbestos bags. At present, there is a large unit of the Cottrell apparatus almost ready to connect up on the roaster flue to catch the dust from flotation concentrates. The worst things smelters have to handle at the present day, they are coming in the Southwest as well as the Northwest, are of material much of which is as small as 500 mesh, and the ordinary methods of catching dust will not apply to flotation concentrates. The Cottrell process is going to be tried out at our plant for this material. We have devised special forms, with a view to preventing the formation of dust from this material, and we are pleased with the results so far obtained. There is another point in connection with this subject that I want to put before the members here, so they will be thinking about it. You know there has been a great deal of litigation between the farmers

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and the smelting companies on account of the damage some smelter smoke causes. In the early days at Anaconda we were turning out a great deal of dust from our small flues and short stacks. That we overcame and we stopped the damage to such an extent that the courts sustained us throughout, and proved that our improvements were really improvements, and we had ceased to materially damage the farmers in our vicinity. The one thing that our good friends of the Bureau of Mines are apt to overlook in checking up smelters is the fact that smelters must be run on a commercial basis. Metallurgy is the art of getting money from ore; and if you cannot get money from the ore, it is not metallurgy. It is a loss. It is not business, either. We have to look out for that when we figure on saving the material in the gases. There is a great deal of good material in the gases that go from smelters, and there is a great deal of material that is not good. There are certain places, as in the center of Germany and England, where it would pay to take everything in the gas and give it a chemical analysis, separating each element and putting it on the market. The best thing to do in handling smoke from smelters is to follow nature's laws as nearly as possible. If you take the smelter smoke and drench it with water, you get a mixture that cannot be deposited in the streams, and there is really nothing than can be done with it. It is worse than if you let the smoke go out, without attempting to purify it. If you cool the gases to a point where it is safe to put them through a bag house, you get sulphur dioxide cooled to a point where it does more harm than if you had let it alone. The thing to do is to find a happy medium, and get the gases escaping at such a temperature that they will go high in the air and diffuse. Gases do not diffuse as quickly as many would have us believe.

S. J. JENNINGS, New York, N. Y.—There is one point in this unaccounted for loss that has not been mentioned. The ore when received at the smelter is weighed, and in most cases thereafter is dumped on to a bed, sometimes from a greater height than others, but always from some height. When you have the dry condition that obtains in the Southwest with occasional high wind, you are going to have a dust loss before it gets in the furnace. My observations this morning have made me believe that this dust loss is material. I was interested in what Mr. Mathewson had to say. We have in Shasta County, a smelter which treats 800 or 900 tons of sulphide ore a day. It is putting out somewhere in the neighborhood of 350 tons of sulphur daily. All the smoke is filtered so that the gas which is going out of the stacks of the bag house is invisible. The gas that comes from the settlers and converter is likewise led into the bag house, so that there is no visible smoke in the smelter. Yet, some 15 miles away from that smelter, farmers have the imagination to claim that damage is being done by sulphur dioxide, their claim being that sulphur

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dioxide forms pistons in the valley of the Sacramento River; when the wind blows with the river current the pistons of concentrated sulphur dioxide travel at least 15 miles and cause discoloration of vegetation. The smelter, on the other hand, claims that there is no material damage done by the sulphur dioxide—that other things cause the discoloration—and the smelter thinks it has proved its case. The material that results from the filtering of the smoke is a most complex fume, containing a great many of the rarer elements—arsenic, bismuth, platinum, copper, silver, zinc and some gold. By means of the electrolytic treatment of zinc, we have found that it is commercially profitable to treat this fume which has accumulated for some 4 or 5 years. The thing to do is to accumulate all material containing values and after a while you will find a process by which a profit can be made. That seemed to be our experience in Shasta County.

C. E. ARNOLD, Miami, Ariz.—I should like to make an inquiry as to the point raised by Mr. Jennings regarding dust losses. At the International smelter at Miami it is noticeable that the bedding bins are entirely housed in, while those at the Calumet and Arizona smelter are exposed to the action of the winds. It would be interesting to hear from Mr. McGregor whether, in this particular, the design at the Miami plant was influenced by the experiencing of excessive dust losses at the Calumet and Arizona plant.

A. G. MCGREGOR, Warren, Ariz.—Of course, we realized that there must be some losses from the wind blowing over the beds. At the International Smelting Co.'s plant at Miami, we have a very valuable material to handle, and we wished to provide against all losses as far as possible, so we housed in the beds.

THE CHAIRMAN.—The question seems to be largely a commercial one, that is how great expenditures are justified in order to increase the savings in dust and fume. A plant smelting a low-grade furnace charge can hardly, for the saving obtained, afford to install a process such as that at the International smelter at Miami, where the charge will run from 25 to 35 per cent., without offsetting the savings obtained by an increased charge which would exceed the commercial profit from the operation.

The point which Dr. Ricketts makes, that there may be a considerable mechanical loss of fines from the bed systems in the Southwest during the high winds of Spring, is well taken and it might and probably would pay to enclose the ore beds to obviate or reduce this loss.

The only Cottrell plant constructed in Southwestern smelters is that of the International smelter, and I hope Dr. Ricketts can give us some of the results of this installation.

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L. D. RICKETTS, New York, N. Y.—Mr. Chairman, with your permission I will speak about the loss. I thoroughly believe that smelters should not allow their ore to drop through the air exposed to the currents of wind. I think the beds ought to be entirely covered, and a sprinkler used, if necessary. The men need not be where the beds are formed. At the Inspiration, designed by Mr. McGregor, we housed in the beds on account of the rich material—I believe that this was suggested by Mr. Flynn—and we are getting results that lead me to believe that it pays to house the beds.

In regard to Cottrells, I had an article in the *Engineering and Mining Journal* a few weeks ago.* I believe the loss of values in the use of the Cottrell system will be down where they can be regarded as nil. Where we have a loss it is in turning down a converter. As far as the material from the converter stack is concerned, I estimated that we were losing about 50 lb. of copper a day. I believe that the Cottrell system, as an element in the recovery of value, has come to stay.

The dust losses from the reverberatory furnaces with our method of feeding, which you will see when you go to Globe, are so slight that we do not feel justified in putting in treaters to recover the copper from that smoke. I believe this corresponds with the experience with the dust chambers for the amount you will recover. As I said, our most serious loss is from the converter and from the charging and discharging of calcine cars.

Leaching Tests at New Cornelia

Discussion of the paper of H. W. MORSE and H. A. TOBELMANN, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1593 to 1610.

THE CHAIRMAN (H. W. MORSE).—Gentlemen, for the first time in the history of the American Institute of Mining Engineers, we have a full session on the subject of leaching—especially on the leaching of copper ores. This branch of metallurgy is rather in its infancy. We have had one big plant running for a year or more, and pretty successfully; and another one which will be started in February or March, the New Cornelia. We have another one at Garfield for the Utah company, which is just designed and construction begun; and from several other points come reports of interesting test work. The first paper on the program this evening is one of some of the test work on the New Cornelia ore by myself and Mr. H. A. Tobelmann.

LAWRENCE ADDICKS, New York, N. Y. (written discussion).—The Ajo proposition has had, as I understand it, two great advantages: it has not been obliged to measure leaching against flotation or some other metallurgical process, and it deals with a copper content high enough

* Vol. 102, No. 9, p. 396 (Aug. 26, 1916).

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to pay for straight sulphuric acid-iron cementation work if forced back upon such a plan.

These advantages have made it possible to prosecute a consistent policy of development along straight hydrometallurgical lines.

The history of these experiments shows a series of investigations into more or less complicated schemes with a gradual return toward simplicity, the present plan involving utilization of SO_2 in regular acid-making practice, precipitation of excess impurities upon the ore itself, final control by cementation and recovery of the copper by the use of lead anodes along more or less copper-refinery lines.

This is about as simple a scheme as could be devised in true cyclical form, and while I think no one will dispute the wisdom of adopting just such a scheme as the first step in large-scale operation, I venture to predict that as experience is gained in actual practice the tendency will be to return to greater complication, as one item after another can be examined separately and tested for cost. The great difficulty in launching a leaching process lies in the multiplicity of interrelated problems which must at first be dealt with jointly. Also in the fact that no two ores seem to present identical problems for solution.

I think this feeling I have outlined is exemplified by the discussion of the reasons for choosing lead anodes as given in the paper and I want to add a few words to this discussion.

The crux of the whole electrolytic problem lies in the mechanism of the oxidation and reduction of iron sulphate. A carbon anode must be thoroughly depolarized or the liberated oxygen will attack the anode itself and the graphite will disintegrate leaving a valueless sludge. In the case of a lead anode, lead peroxide and lead sulphate are formed but the disintegration is very slow as these substances form more or less adherent and conducting coatings. This means that with carbon anodes we impose at once the necessity of dealing with the entire equivalent of ferric sulphate if ferrous sulphate be the depolarizer. With lead anodes the issue can be largely but not wholly dodged through the escape of elemental oxygen. This in turn leads to the necessity of thorough circulation in the case of carbon anodes as against indifferent circulation in the case of lead.

Thorough circulation, however, results in intimate contact between the cathode and the ferric sulphate in the electrolyte with the natural and undesired reaction between these two substances.

Therefore, the price of our increased recovery of copper per kilowatt-hour is the obligation to keep liquors running more than some 0.2 per cent., and preferably much less, in ferric sulphate, away from intimate contact with the cathode, which means either a diaphragm or an efficient utilization of SO_2 or other reducing agent.

Everyone who has advocated the first remedy has come to grief, except

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possibly Hybinette in his two-level system in copper nickel refining, and everyone who has wrestled with the second problem has found that there are intermediate and as yet little-understood reactions to be dealt with. Nevertheless, I believe that as continued operation gives experience and opportunity for experiment, a way out of these difficulties will be found and the carbon anode yet succeed in the practical electrolysis of sulphate solutions.

G. D. VAN ARSDALE, New York, N. Y.—Both those responsible for the tests of which the results are given in this paper and the Institute are to be congratulated; the first for the successful results finally obtained on a difficult and new problem, and the profession on the liberal policy which made possible the publication of this very valuable summary of practical leaching data.

The thoroughness and care with which this work has been done may be judged from the time, nearly 4 years, during which these experiments have been carried on.

It is especially gratifying to the writer that some of the main points developed during the series of experiments on leaching by Phelps, Dodge & Co. at Douglas have been entirely borne out by this work; the first of these being that ferric iron can be controlled by the use of sulphur dioxide, the second that solutions that had previously been considered entirely too "foul" as an electrolyte for copper deposition can safely be used, and the third that by the use of depolarization the anode problem is entirely solved, either by lead or graphite, both of which had previously been considered as impossible for electrolysis of sulphate solutions of copper.

The paper is particularly valuable in giving a discussion of the conclusions leading to the adoption of the final process by the New Cornelia company and as bearing on their choice of lead instead of graphite as an anode material. My reason for discussing, and to some extent differing from, these conclusions is that I feel that from the results of Phelps, Dodge & Co.'s work it is necessary to emphasize the fact that, while the choice of lead in this case may be undoubtedly justified, yet there are conditions under which graphite should be selected and these conclusions therefore should not be taken as general conclusions against its use under other sets of conditions.

Graphite, which was first used under the writer's direction on a considerable experimental scale as an anode material for this kind of work, before this time had been considered as absolutely unsuitable for the electrolysis of sulphate solutions of copper. I have a letter from the manufacturers in which this statement is made, and the same thing is stated in Greenawalt's book on copper leaching. There is no doubt that its use has certain practical disadvantages, the principal one of these being that a tank-room ventilation system will probably be needed to enable it to be used. While this means a radically different tank-room

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design, it need not mean anything at all impracticable or leading to increased tank-room costs beyond the negligible amount of power for ventilation. On the other hand, its use has very considerable points in its favor which in my opinion outweigh those against it, generally speaking.

Regarding its durability, I believe it is entirely safe to say, from the results of careful tests made by Phelps, Dodge & Co., that under proper and easily controlled conditions its depreciation costs, or in other words the amount of disintegration, per pound of copper produced will be negligible.

The ampere efficiency of graphite is, as stated, badly reduced by a ferric iron content of electrolyte, with which still fair efficiencies may be had with lead, this meaning as stated the maintaining of a limit of about 0.2 per cent. ferric iron for graphite as against a limit of about 0.4 per cent. with lead. It should be carefully noted, however, that as regarding the amount of power per pound of copper produced a reduced ampere efficiency at a voltage of about 1 volt does not mean nearly as much as the same reduction with a voltage of over 2 volts. In other words, one of the very considerable practical advantages of graphite is the much lower voltage at equal current densities with lead. Regarding the ampere efficiency obtainable with graphite, it is possible that a casual reading of the paper under discussion might lead to the conclusion that a low ampere efficiency was characteristic of graphite. That this is not true was proved conclusively by Phelps, Dodge & Co. in a series of tests last summer which demonstrated that it is commercially practical to keep the ferric iron under 0.2 per cent. and under these conditions to obtain fully as good or better efficiencies than those cited in the present paper, with yields over a considerable period of 3 lb. of copper per kilowatt-hour at a current density of about 12 amp. per square foot, which it should be noted is 76 per cent. in excess of the average current density as given in the table on page 1600 of the paper under discussion. With the lower current densities as given in this paper, our pounds of copper per kilowatt-hour would have been still higher.

It is, of course, not reasonable to expect that a high SO_2 reduction efficiency will be obtained if the SO_2 is blown out of the solution before it has had time to act. In the worst case, this factor of the time needed for the reduction of ferric iron by SO_2 means a large, though not at all prohibitive, storage capacity, and it may be stated that one of the results of our work at Douglas last summer was to demonstrate that the ferric iron can be kept within our 0.2 per cent. limit by a storage of a portion only of the main solution.

It is true that the anodic efficiency of graphite is as stated on account of air agitation over 100 per cent., that of lead being 30 to 35 per cent. Expressed in another way, however, this means, with anything like a

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reasonable SO_2 reduction efficiency which can be had as we have shown, for practical purposes that the amount of acid regenerated by lead is much less than with graphite. Now if one has an acid plant already built or ordered or intends to erect one, the comparatively small amount of acid produced by the use of lead anodes is not so important, but where this is not the case this considerable extra amount of acid produced by the use of graphite is a considerable practical advantage in its favor. Where, as in this case, an ore does not require more than 3 lb. of acid for leaching per pound of copper produced, the amount of acid regenerated by the use of the graphite system will be ample for leaching purposes. A comparison, therefore, between the two systems should include the cost of an acid plant against lead as regards installation costs, and, as regards operating costs, the extra cost of acid needed together with freight if brought from a distance. It seems to me, therefore, that the statement in point 1 of the summary on page 1607 should include this factor.

Without intending criticism of the decision made in this particular case, I believe, nevertheless, that the following comparison as to the relative merits of lead and graphite will be true and more generally applicable than those given.

1. Installation costs for the two systems will be about the same, except that, where an ore requires more than 1½ lb. of acid per pound of copper produced, the additional costs of an acid plant plus railroad equipment for carrying it when produced at a distance from the leaching plant, plus storage tanks, piping, etc., at the leaching plant, must be charged against lead.

2. Careful determinations extending over a considerable period have shown that the disintegration of graphite and its consequent depreciation per pound of copper produced is negligible. The prevention of graphite anode disintegration means simply maintaining the conditions necessary for proper depolarization. When this is done graphite will not disintegrate appreciably, but if these conditions are not maintained it will go rapidly. Exactly the same thing is true for lead, but, since the anodic efficiency of lead is only one-third that of graphite, the disintegration of lead anodes under the same service may be expected to be at least as rapid as graphite.

3. Peroxidized lead detached from an anode will be lost, and the scrap value of partly peroxidized and badly corroded lead anodes, when high freight rates are considered, will not be very high. Furthermore, when the slow rate of disintegration of graphite anodes finally reduces their thickness very appreciably, they can be reassembled and the scrap loss reduced thereby to a small amount.

4. The power required for precipitation with lead is at least three times that for graphite under the same conditions.

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5. More than twice as much acid is regenerated with graphite as with lead. This, as stated above, means not only an extra installation but also an extra operating cost for this additional acid needed, when lead is used.

6 and 7. Since the anodic efficiency of graphite is very high, higher circulation and some form of agitation in the cells are required. In this connection it may be stated that, although air agitation has been used in the work at Douglas with good results in some respects, I have never been in favor of this method of agitation as compared with others, unless more than 3 lb. of acid per pound of copper are needed. The excess anode efficiency over 100 per cent. is of course mainly due to the oxidation caused by the air, and it is clearly illogical to use an oxidizing agent for mechanical purposes where reducing conditions are wanted.

8. The extra storage capacity needed for the time of reduction by SO_2 as shown above need not be more than a fraction of the total solution bulk. Furthermore, if a good SO_2 reduction is obtained, the tank-room ventilation difficulties are correspondingly reduced.

9. and 10. Since the anodic efficiency of graphite is three times that of lead, it of course necessarily follows that larger tower capacity and greater volume of solution sent to these are required. However, the saving in installation by not needing an acid plant will pay for this and the extra storage capacity needed under the preceding paragraph at least several times over.

11. The point of the necessary tank-room ventilation has already been spoken of.

12. I must disagree absolutely, so far as my experience goes, with this conclusion as to tank-house capacity being necessarily larger with graphite than with lead. While the cost of power is an important factor in this connection, we have not considered for our requirements that the low-current densities, averaging 6.8 amp. given in the paper, are economical under our conditions, and the figures we have obtained with current densities around 12 amp. per square foot, in which average ampere efficiencies at this higher density fully equal to or better than those given in the paper and obtained over considerable periods appear to throw this comparison decidedly in favor of graphite.

13. It is a little difficult to see just why operation with graphite anodes will be any less "fool-proof" than with lead. Granted that the conditions for keeping the ferric iron below our limit are known, I cannot see any reason whatever for requiring any more care or skill for maintaining this limit than for maintaining an equally vital one differing from this by only 0.2 per cent.

My opinion, not expressed as a criticism of the decision made in this particular case, but as a general one, is that the advantages and undoubted lower operating costs of graphite decidedly outweigh those

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with lead. In other words, it would only be considered advisable under very exceptional conditions to make the very large additional investment required for an acid plant over and above the acknowledged nearly equal costs of the two systems, and still with this extra investment to obtain a higher operating cost.

THE CHAIRMAN.—I would like to say that Mr. Van Arsdale should be thanked for a good many discussions and arguments, and that the conversations and arguments with him were of very great advantage to us in this work at the New Cornelia; and in a good deal of what he says I would agree. It is highly probable that we will be able some day to work out the use of graphite to make a saving in power cost, and to generally improve the process. As far as we could see from the test at the New Cornelia, this was going to be a difficult undertaking, so we decided to put it off for a few years. I think we will later come back to large-scale tests on the use of graphite. Certainly the gain to be expected when this has been worked out is a great one. I, personally, was sorry to stop the research work on the graphite.

S. J. JENNINGS, New York, N. Y.—I would like to ask for some information. I see that this paper has considered merely two kinds of insoluble anodes, lead and graphite. I understand a considerable amount of work has been done with other insoluble anodes and I would like to ask if tests have been made at New Cornelia with any other insoluble anode such as fused magnetite.

THE CHAIRMAN.—We have made none at Cornelia, except with lead and graphite, and a mixture of the two—that is, coke cast in lead.

F. S. SCHIMERKA, Clifton, Ariz.—In connection with the paper presented, I wish to ask the question why the process of precipitating the ferric iron in a neutral liquor on fresh ore was not finally employed and preference given to the introduction of sulphur dioxide to effect the elimination of trivalent iron?

In regard to the use of lead anode sheets, I would like to know whether lead-covered iron grates would be applicable, and I ask this question because I remember that I have seen such anodes employed in liberator tank practice but have not been able to collect data concerning their efficiency compared with that of solid sheets used for the same purpose.

THE CHAIRMAN.—That was the basis for starting the large plant. In the 7-ton plant and in the 40-ton plant, at first, the ferric iron was precipitated, but later for an unknown reason we got no such result, the ferric iron going up and the ampere efficiency going down.

C. G. GRABILL, Matehuala, Mex.—I would like to ask if it was possible that the difficulties arose from the depth of ore in the two tanks.

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In the first one you have a small volume of solution; and in the second, you have a very much larger body of solution. In the first condition, with the small volume, you have much more air present and oxidation taking place. If the charge is increased—rather, if the percentage of the solution is increased—that condition does not obtain in the latter part of the action. I would like to ask if you have done any work along that line—to investigate that point?

THE CHAIRMAN.—That is one question we left to find out in the 5,000-ton plant. Mr. Flynn, have you anything to say?

F. N. FLYNN, Clifton, Ariz.—The leaching that we did at Clifton, for something over 20 years, was the simple process of an acid leach, with iron precipitation, and the wasting of all liquors. The modern leaching which you are about to attempt is something which you have developed and proven to your entire satisfaction and until such time as it is working out, practically, I don't feel that I would have any criticism to make. I don't feel competent to criticize. It does seem to me that you are working in the right direction. The only question that comes to my mind is the percentage of liquors you will have to throw away. That, I understand you have worked out to your entire satisfaction.

THE CHAIRMAN.—I think that 6 months' operation of the big plant will make us all feel more confident than we do now.

STUART CROASDALE, Denver, Colo. (communication to the Secretary*).—I think the New Cornelia ores at Ajo may be considered as typical of all copper ores in the United States that are amenable to acid leaching in a raw condition, since all known ores of this kind contain a certain amount of soluble iron oxides and alumina which will be a contending factor in electrolytic precipitation of the copper.

It is interesting to observe that the results obtained by the authors have verified in almost every instance the predictions made before they started, yet their work has been so efficiently and thoroughly done that it forms a most valuable contribution to the hydrometallurgy of copper, and particularly to the section on electrolytic precipitation. They are to be congratulated on having such an opportunity to definitely advance our knowledge so much in this direction.

I regret that one thing more was not tried, at least on a laboratory scale, and that is, roasting the ore before leaching. In the Ajo ores the iron exists in all forms of oxidation from chalcopyrite to ferric oxide. The processes of nature have been so gentle that much of this iron is as easily soluble as freshly precipitated hydroxide. Aluminous compounds in the rock also exist in all stages of kaolinization and are likewise soluble in dilute acids. A slight roast would oxidize all of the iron, including the

* Received Sept. 29, 1916.

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residual pyrite, to ferric oxide and would dehydrate the aluminous compounds and salts to the oxide. Since ignited ferric and aluminum oxides are not readily soluble in dilute acid, it would seem that the lixivium obtained from roasted ore would be practically free from deleterious salts, which would obviate the necessity of depolarizing and "bleeding." Assuming this to be true and the solubility of the copper oxide not diminished, the cost of roasting would be compensated by the recovery of the copper now held in the tailings as sulphide, which is likely to be in ever increasing quantity, and as soluble copper retained by the argillaceous material; also by the expense of roasting pyrite to produce sulphur dioxide, and by the expense of precipitating part of the copper on scrap iron at each cycle of the lixiviant.

The 2,000-Ton Leaching Plant at Anaconda

Discussion of the paper of FREDERICK LAIST and HAROLD W. ALDRICH, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1281 to 1293.

F. N. FLYNN, Clifton, Ariz.—I would like to ask Mr. Mathewson what percentage of his leaching liquor is wasted at this time? It has a bearing on the question in connection with the New Cornelia, and I think it is a very important question.

E. P. MATHEWSON, Anaconda, Mont.—I am not sure what the percentage is now, but we don't discard the strong liquors. Something like 25 per cent. of the wash water taken out each day is sent over a special set of scrap iron tanks, and the copper recovered there. We had a little trouble at first. We did not proceed to use the sponge iron for the reason that the plant we originally contemplated was abandoned for flotation. Our original idea was to use a leaching process on all the tailings of the mill, but we found it would take a great deal of time to build a plant; and the cost of operating the plant we estimated to be about the same and the recovery the same. It was a question of time with us, and we decided to adopt flotation. That changed our plans. We now have the leaching plant, treating the tailings of the old dump, and the flotation applied to the current tailings. The sponge iron was figured for the large plant. We can get all the scrap iron necessary for the small plant—plenty from the scrap produced in the main plant of the smelter.

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Possibilities in the Wet Treatment of Copper Concentrates

Discussion of the paper of LAWRENCE ADDICKS, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1565 to 1573.

F. N. FLYNN, Clifton, Ariz.—As a number of my associates in Arizona know, for a great many years, I have felt that leaching was one of the coming problems. We are about to start experiments at Clifton, along the idea of leaching wet flotation concentrates direct, without roasting—differing in this respect from Mr. Addicks' suggestion. We are prompted in starting these investigations by the difficulty in transporting and handling flotation concentrates, and consideration for drying and roasting dust losses. I have had the idea for 5 or 6 years past, that those very fine concentrates could best be treated by leaching at the concentrating plant, thus eliminating our transportation difficulties to the smelter, and dust losses at the smelter. Given flotation concentrate, if it must be dried and roasted, it would, in our case, undoubtedly be cheaper to smelt the calcine, but if it can be leached raw and wet and the drying difficulties and dust loss eliminated, it would be very advantageous to leach it at the mill. I realize that it will be a difficult problem, and one which may take several years to develop, but if developed, it would pay for many years' expense of investigation, and we have been encouraged by our general manager to investigate its merits.

General Subject of Leaching

Discussion at the Arizona Meeting, September, 1916.

THE CHAIRMAN (H. W. MORSE, Los Angeles, Cal.).—I would like to open this meeting for a little while to the general subject of leaching. We ought not to hold back if we have any new schemes for the future. I know that those of us who are tangled up with this leaching work will appreciate any suggestions that will make us work harder and scheme harder to pull through these past difficulties on leaching.

F. S. SCHIMERKA, Clifton, Ariz.—Under the direction of J. W. Bennie, General Manager of the Shannon Copper Co., Clifton, Ariz., it has been my valued privilege to coöperate in the elaboration of a hydrometallurgical process aiming at the recovery of copper in mill tailing resulting from a gravity concentration of a low-grade semi-oxidized sulphide ore occurring in a porphyritic gangue.

The experimental work which has been carried on with a 25-ton plant is completed, and the company is building at present a leaching plant on a larger scale, the first unit consisting of a 150-ton roasting furnace and accessories for leaching the calcine.

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I wish to lay before you only the most essential points of our process, the outstanding feature of which is the non-application of acids or any other chemical in the leaching operation proper. After a sulphatizing roast conducted under well-defined conditions in a mechanical roasting furnace of the multiple-deck type, the calcine is treated with water only. The separation of the copper liquor will be effected by decantation, and the copper, at least for the present, precipitated by scrap iron. The character of the tailing, which is highly basic and remains so even after roasting, prohibits the application of sulphuric acid, which is consumed to the amount of more than 10 lb. for each pound of copper dissolved from the calcine. Our mill tailing, all of which passes through a 1-mm. opening, contains an average of 20 lb. of copper to the ton, 1 per cent., we may say. Fifty-five per cent. and more of the total copper is present in oxidized condition, mainly as malachite and azurite, the balance is chalcocite. Sulphur is present in the amount of 1 per cent., mostly as pyrite. The precious metals are practically absent, and that no attempt at their recovery need be made has materially simplified our working procedure.

The manner in which the roasting of the raw tailing is conducted is, of course, vital to the process, there being no other means employed to convert most of the copper into water-soluble sulphate than the agencies of heat, oxygen and sulphur during the passage of the material through the furnace. A close regulation of the temperature in the roaster by means of pyrometer control is essential. Tests have shown that the re-formation of water-insoluble basic copper sulphate and oxide takes place when the temperature rises closely to 900°F., and we aim at keeping the temperature on the hottest floor between 830 and 860°F. We have encountered no difficulty in accomplishing this. It is, however, necessary to limit the amount of sulphur in the charge to a quantity which in an empirical way has been found to produce the best results, and which excludes the possibility of local overheating, as would take place by the oxidation of a large amount of pyrite. Moreover, any excess sulphur will, by formation of sulphur dioxide, decrease considerably the oxidizing effect of the hot gases striking through the furnace. Our roaster gases contain less than 0.1 per cent. of sulphur dioxide when issuing from the stack. For these two reasons, we aim at keeping the charge as poor in sulphur as we can afford to do without injurious effect upon results. Taking the copper contents in the tailing as a basis, we adjust the sulphur in the charge to 1½ lb. for each pound of copper in the tailing. This necessitates the addition of ½ per cent. of sulphur to the charge in the form of iron pyrite.

For fuel we employ oil burned in an external firebox, which is air-jacketed to prevent loss by radiation. The fire gases which, mixed with the required volume of air admitted through openings in the firebox, at

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the entrance into the furnace, are kept at 800°F., are sent into the 10-deck roaster on the ninth floor. We do not employ muffles. The fuel consumption in the 25-ton roaster was 5 gal. of oil per ton of tailings treated, frequently less, and we are confident that we can decrease it to 3 or 3½ gal. at the most in the large roaster on account of heat-saving devices incorporated in its construction. Also, there will be another source of heat provided by the exothermic reactions that take place during the roasting process in the furnace. The additional heat derived from these reactions is quite considerable and has made itself felt to an appreciable degree in the small pilot furnace.

In the new plant, the separation of the copper liquor from the leached calcine will be effected by counter-current decantation in three Dorr thickeners. Directly from the roaster the calcine is discharged into a circular mixing tank where it receives return liquor from one of the thickeners. From the mixing tank the pulp and liquor pass into three Dorr classifiers placed in series to effect a separation between the sand and slime. The slime in the overflow from these classifiers passes successively through the three thickeners, the first of which delivers its clear overflow to the precipitation boxes. As the pulp progresses through the battery of thickeners it is washed by water and waste liquor from the precipitating launders traveling in the opposite direction.

During our test runs with the 25-ton roaster, the extractions ranged from 65 per cent. to 77 per cent. of the total copper in the tailing. For the present the copper in the liquors is precipitated on scrap iron in a system of launders, although electrolytic separation is taken into consideration by the management for the future. Concerning the electrolytic deposition of the copper and a decision to introduce it in our plant, we are looking forward to the results which will be obtained by this method at Ajo in the New Cornelia Copper Co.'s plant now under construction, where the process has been worked out by such pioneers as Dr. Morse and Mr. Tobelmann, who have presented their paper on this subject before this meeting.

B. B. GOTTSBERGER, Miami, Ariz.—For some time we have been experimenting on the recovery of oxidized copper. While trying to convert the oxidized copper to sulphide by means of calcium sulphide and sulphuric acid, we concluded that what conversion was obtained, resulted from the solution of the copper in acid and its subsequent precipitation as sulphide. Laboratory work demonstrated that a process along these lines could probably be made to work, but desiring to avoid the use of hydrogen sulphide, if possible, we concluded to look for another method before trying this scheme on a working scale. Early in the present year we did some laboratory work along the lines of the process we have just seen at Chino and at the present time are trying out this

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idea in an experimental plant handling about 50 tons per day. I am very hopeful of a successful result from this work.

H. E. WILLIAMS, Calumet, Mich.—The Calumet & Hecla Mining Co. has been experimenting for some time, under Mr. Benedict's direction, upon the problem of reclaiming copper from the old tailings banks in Torch Lake, which contain about 40,000,000 tons of recoverable sands. After many laboratory and test-plant experiments, they have finally adopted the ammonia leaching process, and have built a 2,000-ton plant, which is now in partial operation.

The plant has eight unlined steel leaching tanks 54 ft. in diameter by 12 ft. high, provided with the usual wood grating, cocoa matting and canvas filters, removable steel covers and one central and six circumferential discharge gates. After being reground and treated on tables, the tailings are pumped into the plant and deposited in the tanks by revolving distributors. The leaching solution contains about 1.5 per cent. of NH_3 and an equal percentage of CO_2, and after dissolving the copper, is pumped to the still house, where it is treated in ammonia distillation apparatus with low-pressure steam which precipitates the copper as oxide. This is sent to the smelter, while the NH_3 and CO_2 distilled off and condensed is pumped back to the leaching plant to be used again. From sand containing about 10½ lb. of copper to the ton, we recover about 8 lb. We have had to develop a roughing still, which has delayed the still-house work, otherwise the plant would now be operating at its rated capacity. I am sorry I cannot give you a description of the chemical reactions; but I am not a chemist and must leave that for Mr. Benedict in the future.

G. D. VAN ARSDALE, New York, N. Y.—I would like to ask Mr. Mathewson the reason he adopted wood tanks instead of concrete tanks.

E. P. MATHEWSON, Anaconda, Mont.—We had experimented with wooden tanks, and when we put up the 1,200-ton tanks, we thought the wooden tanks would be satisfactory. After they had been in service for about a year we noticed deterioration, and we had them lined with lead. If they had been built of concrete, they would have been more expensive. For a plant the size of ours, we think the wooden tanks, lined with lead, are preferable; but with larger ones, we think it would be preferable to build them of concrete.

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Petrography of the Mount Morgan Mine, Queensland

Discussion of the paper of W. E. GABY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1725 to 1744.

L. C. GRATON, Cambridge, Mass.—May I ask, in connection with one point in the summary, to what extent the author regards actinolite equivalent to tourmaline as an index of high-temperature and presumably magmatic conditions of deposition of the accompanying sulphides?

W. E. GABY.—I merely advanced the idea that the mineral actinolite might have some such significance. In the copper deposit at Braden in South America, the ore occurs in an andesite breccia which is cemented by a matrix of sulphides, quartz, and tourmaline. Tourmaline is a high-temperature mineral, and when we find it associated with ores, these are regarded as having come from a volcanic or deep-seated source, hence a primary hypogenetic deposit. Thus we can count on greater continuation of the ore with depth than in some other kinds of deposit, and I thought that the actinolite, being also a high-temperature mineral, might have some such bearing in the case of Mount Morgan. I did not advance this idea as a final conclusion from a study of such phenomena, but thought that it might bear some discussion in view of the fact of the peculiar occurrence of this mineral in a notable orebody. It also occurs with the copper ores at Ducktown, Tenn., and in both cases may only be due to general regional metamorphism.

Geology of the Warren Mining District

Discussion of the paper of Y. S. BONILLAS, J. B. TENNEY and LEON FEUCHÈRE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1397 to 1465.

IRA B. JORALEMON, Warren, Ariz.—This paper has covered the situation so well that I have few suggestions to offer. Two points, however, are shown more clearly in the lower mines of the Calumet & Arizona Mining Co. than in Copper Queen workings, and may be worth mentioning. The first is the usual association of high-grade primary sulphide orebodies with much larger masses of pyrite and silica, averaging less than 1 per cent. copper. The richer areas occur sometimes around the borders of the lean pyrite bodies, and sometimes entirely within them. While as yet we have not been able to find any certain reason for the location of high-grade sulphides in low-grade masses, it seems likely that the age of fracturing is one controlling feature. The first stage of mineralization, with pyrite and silica, had the effect of sealing up the rock. The later solutions, richer in copper, could pene-

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trate into the low-grade masses and bring in the rich copper sulphides only along fracture zones which were kept open by a more or less continuous motion during the time of mineralization. Fractures of the same age as the mineralization are therefore most favorable indicators of rich primary orebodies.

The second point I should like to emphasize is the very irregular pre-Cretaceous topography. This is best shown by drill holes east and south of the productive area. In the basin between Lowell and Warren, about 5,000 ft. east of the Briggs Shaft, drilling showed from 500 to nearly 1,000 ft. of Cretaceous conglomerate, filling an old canyon in the Carboniferous limestone. In the large valley south of Warren there was a far greater canyon. A drill hole about 4,000 ft. south of the Country Club, or 6,000 ft. south of the crest of the hills below Warren, was sunk to a depth of over 3,200 ft. without reaching the bottom of the Cretaceous conglomerate. This is the deepest diamond-drill hole ever put down in the Southwest. From the top of the Carboniferous limestone hills south of Warren to the bottom of this drill hole there is a difference in elevation of nearly 4,500 ft., in 6,000 ft. horizontally. Allowing for erosion of the hills and for considerable probable depth of conglomerate below the bottom of the drill hole, the total depth of the pre-Cretaceous canyon must have been 7,000 or 8,000 ft. This is pretty nearly the record canyon in history. With such a varied land surface, the pre-Cretaceous oxidation may have reached a level far lower than the present deepest mine workings in the Warren district.

F. L. RANSOME, Washington, D. C.—There are a few points in the excellent paper just read upon which I should like to make brief comments.

The authors correlate definitely the thin quartzite at the top of the Abrigo at Bisbee with what I have named the Troy quartzite in the Globe-Ray region. I have myself suggested this correlation in a recent paper but think that more work is needed before it can be concluded that they are the same.

A very important and interesting feature brought out clearly in the paper just presented is the relation of oxidation and enrichment to the former Cretaceous surface. In my report on the Bisbee district published many years ago I stated that a good deal of the enrichment had probably taken place before the deposition of the Cretaceous Glance conglomerate. The underground workings, however, were not extensive enough at that time to show how regularly the zones of oxidation and enrichment dip south in conformity with the general dip of the basal Cretaceous beds.

The statement made in discussion by Mr. Joralemon regarding the great apparent thickness of Glance conglomerate found in drilling in the plain south of the Bisbee hills should be considered in connection with

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known overthrust faulting in that general region. The conglomerate may be both faulted and tilted, so that a drill section is not necessarily a measure of original thickness.

The observations made by the authors on contact metamorphism at Bisbee agree substantially with my own results. Although the Bisbee ore deposits are generally regarded as of contact-metamorphic origin they can hardly be considered as typical examples of that class. Such metamorphic silicates as are present are generally small or microscopic.

L. C. GRATON, Cambridge, Mass.—When the summer session of the Institute was held in Butte 3 years ago, we were given a paper by Mr. Sales on the geology of that remarkable district which may, I think, properly be regarded as constituting a new milestone in the advance of the application of geology to mining, in that it represented the accumulated data and evidence and speculation that had resulted from systematic, intensive and constant study by the geologic staff of the Anaconda company for a period of some 15 years. Obviously, such advantages for observation, interpretation and verification afforded to conclusions a validity that could have been secured in no other way. Without question, Mr. Sales' paper will continue to be the standard authority on the Butte deposits for a long time to come, and even when the horizons on which its observations were based may have become exhausted and abandoned, the principles which it disclosed will undoubtedly be found in large part applicable to the deeper regions to which operations may extend.

The present paper, dealing with a district of such extensive development, such great production, such rich ores and such complex deposits as Bisbee, is certain to be of interest to all engaged in the study of mining geology. Like the paper on the Butte deposits, this one also is the product of systematic and steady effort for a number of years by the members of a company geologic staff, aided by the coöperation of their colleagues in the district.

Of the many directions in which this account of Bisbee geology merits attention, the feature that perhaps impresses one most strongly concerns the method, or at any rate, one of the methods, by which the results were achieved. Without in any way minimizing the difficulties of geological accomplishment at Butte, it may be said that, owing to the essential monotony of rock character and to the nature of the fractures in and along which the ores have been either deposited or dislocated, the outstanding problem in that district is of a structural kind. In Bisbee, the variety of rocks and the consequent variations and irregularities of the channelways that traverse them afford an ample supply of structural problems which the geologist must solve. Yet superimposed on these are problems of another sort and of great complexity; they are

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essentially chemical or mineralogical in character and relate not only to the initial deposition of the ore but to its subsequent alteration and to the enrichment or dispersal of its metal content. Problems of this kind have to be attacked by different methods from those employed for the solution of structural problems. The method which has been found most effective depends upon the microscope. No reader of the paper by Messrs. Bonillas, Tenney and Feuchère can fail to note the extent to which they have utilized the facts revealed by microscopic examination.

In this respect, therefore, it seems to me that this contribution marks still another step in the progress of scientific ore-finding. For although the method employed has been used by others, this, so far as I am aware, is the first recorded instance in which field methods and laboratory methods have been systematically coördinated on a comprehensive scale under the advantageous opportunities for constant and extended observation that are open to the geological department of a large mining company. The authors certainly deserve our congratulations and our thanks, and I think the Copper Queen company and other companies pursuing the same policy are to be commended for their far-sightedness in recognizing as valuable and permitting their geologists to follow methods of attack which some of narrower outlook might regard as too "flossy" and scientific to be of any practical service. Much is undoubtedly to be expected of such a practical, commercial application of this combination of methods which has received so gratifying a start in the present paper.

In connection with one particular detail covered by the paper and referred to by Dr. Ransome—the subject of contact-metamorphic ore deposits—I should like to express assent to the statement of the authors that no sharp dividing line can be drawn in Bisbee between the type of deposit which nearly everyone would probably agree should be called a contact-metamorphic deposit and a type which lacks many of the characteristics that one expects a contact-metamorphic deposit to possess. In other words, any boundary must be drawn arbitrarily; and perhaps no two people will draw it in exactly the same place. Dr. Ransome no doubt had this idea in mind in the discussion he gave to the subject in his report on the Bisbee deposits and also in his present remarks when he speaks of *typical* contact-metamorphic deposits. The question immediately arises, however, as to what is a typical contact-metamorphic deposit. Shall we regard as typical only those deposits in which the effects of igneous metamorphism have been most intense or extreme? Or shall we call typical those contact-metamorphic effects which are not so extreme and are met with more commonly? For in studying a number of districts in which contact-metamorphic deposits are involved, my associates and I have been impressed more and more with the fact that the situation in Bisbee is neither unique nor unusual.

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It would appear that what is now called contact-metamorphism is simply a group of effects that stands at one end of a continuous series of effects, and that agreement must be reached as to where one shall stop in that series and say: "Contact-metamorphism ends here." Indeed, as evidence accumulates, it is somewhat doubtful in my own mind whether much is to be gained in drawing such a sharp division line. Plainly, here in Bisbee, the geologists have found no reason, in fact, have found it impossible, to draw a sharp line. It is a question in my mind whether we should not either broaden and improve our definition of contact-metamorphism in connection with ore deposits, or drop it as a significant term of practical application; for present usage is likely to lead to ambiguity and confusion and to become a cloak for loose and inaccurate thinking and expression.

THE CHAIRMAN (G. F. G. SHERMAN, Bisbee, Ariz.).—As Mr. Graton has spoken of the attitude of the companies with regard to geologists, I think we should disclaim some of the credit which he gives us. While we believe that they have been of great assistance to us, we must say that geologists did not find all the ore. We had a great deal of ore before we had geologists. A great many orebodies are found by the miners and many are stumbled on by accident. But in this complicated country, we shall always need as much assistance and information as we can get from whomever we can secure it.

Coöperative Effort in Mining

Discussion of the paper of JOSEPH P. HODGSON, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 113, May, 1916, pp. 867 to 869.

JOSEPH P. HODGSON.—As to "Safety First," I did not deal with that, but I will say that in 1913, the number of men killed was nine, and that year the "safety first" program was commenced; the next year, 1914, the number was reduced to five. In 1915, I am pleased to say that the number was reduced to two in the Copper Queen's entire property, and we have none so far this year.

THE CHAIRMAN (G. F. G. SHERMAN, Bisbee, Ariz.).—I can say, further, that we have from eight to nineteen men per month incapacitated for more than 14 days. Those are classified in this State as serious accidents. The number of men injured has apparently increased, but that is due to our improved method of recording and classifying accidents, no matter how slight.

C. W. GOODALE, Butte, Mont.—The importance of coöperation among the members of the staff of an industrial organization is well brought out by Mr. Hodgson's paper.

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The Anaconda Copper Mining Co. has done something along this line, but has not gone as far as the Copper Queen Co. At the Anaconda works, on the last Tuesday of every month, the manager meets the heads and foremen of all departments, for the purpose of discussing costs and efficiency. The superintendents are asked to give the reasons for any changes in costs, and also to suggest methods by which greater efficiency may be obtained. If any member of the staff has been away visiting other plants, he gives an outline of his observations, and then a general discussion follows.

In the efforts of the company to improve working conditions and effect a reduction in the number of accidents, coöperation is of the greatest importance—not only among the foremen and shift bosses, but among the miners and smeltermen.

When the Bureau of Safety was established, about 3 years ago, our first step was to make an analysis of causes of accidents, with a view to attacking the principal causes with special vigor. A review of accidents in the Butte mines for the preceding year showed the following figures:

	Per Cent.
Fall of ground	43.21
Caught by mine cars	12.97
Handling tools	14.54
Handling material	11.87
Falling	10.00
Falling timber	3.62
Machinery	1.11
Around cages	0.74
Explosives	0.70
Miscellaneous	1.24
	100.00

In the meetings of the safety engineers and foremen these figures were considered, and the fight against accidents began. It was soon found that in order to make the miners coöperate in this movement, we should have to inflict some penalties on them for failure to look out for their own safety. For instance, when men are found working under conditions which are unsafe, and which they can remedy, they are laid off 7 days, and for the second similar offense the penalty is a layoff for 14 days. For the third offense they are discharged.

We have effected quite a satisfactory reduction in accidents in the mines of the company. During the year 1915, we had 1.42 fatal and serious accidents per 10,000 shifts, and 1.09 in the first half of 1916—a reduction of 23 per cent.—and this in the face of a mine fire which occurred in February, 1916, and which caused the loss of 21 lives.

In the reduction works our efforts have not accomplished so much. Perhaps one explanation is that the causes of accidents in reduction works

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are more varied in character than in the mines, and it has been difficult to attack any one particular cause with special emphasis. Furthermore, it should be stated that a large amount of construction work has been going on during the last 3 years, and I think it is generally admitted that there are more risks in such work than in operation—particularly when construction and operation are going on together, as has been the case in some of our departments.

At the Anaconda works the company has spent more than $60,000 in safeguards which have been suggested at meetings of the Safety Committees, and the effect will doubtless be shown in accident records as time goes on.

CHARLES A. MITKE, Bisbee, Ariz.—The practical man's lack of opportunity presents a difficulty to his satisfying his interests in technical subjects. The object to be attained in introducing this course was to show that there is no dividing line between the so-called practical and technical sides of mining; that if any ideas, devices, methods or applications are practical, then they are also technical because all practical methods have a technical basis. The converse is also true that all technical methods which have commercial value and are worth introducing are practical and have a practical basis. However, the terms practical man and technical man still exist, and it is only by coöperation of both classes that the best results can be obtained through their combined coöperative effort. Four years ago none of our shift bosses or foremen would attempt to write or read a paper at any meeting. However, after certain members of the operating staff presented papers, then they were also induced to prepare papers. These men were given all possible assistance. Their subjects were first discussed with them individually so as to bring out the important points and then were written and revised to emphasize these points and in their final form presented at the meetings. Two courses were pursued during the past year—a general course at the mine foremen's dinners and a definite course in mining methods, etc., at the Copper Queen mining conference. An example of subjects taken up in the general course was a lecture on "The Importance of Keeping up the Grade of the Ore," while an example of the subjects used at the conference was the discussion of "The Details of Doing Development Work." At mining conferences a constant effort was made to bring out the basic principles which govern the varied mining operations here as well as in other fields. After they were taken up, the application of these principles was then considered in connection with specific and local mining problems. It was reasonable to believe that this plan would be interesting and at the same time would present the subject from a broader point of view. When members of the conference were asked to prepare papers, they received every encouragement to take

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pains and make them as thorough and complete as possible. It also led to another field. This led in some cases to suggestions from the shift bosses as to the details of our operations. These suggestions from the shift bosses were then embodied in papers and presented at subsequent meetings with the guidance and assistance of the instructor. During this coming year we plan to increase the scope of the field. A course to miners will take up the principles and first course in mining methods; a course for shift bosses and engineers, will give more advanced work in mining problems and greater detail in mining methods; and a course for the foremen's dinners will include subjects of a broader nature that will apply to all concerned. The course at the foremen's dinners consists of lectures, while that at the Copper Queen mining conference for shift bosses and engineers is a combined course of lectures, problems, readings, study, etc. There are many things that a shift boss ought to know and, since they can only be learned through study and application, these courses have been instituted.

C. E. ARNOLD, Miami, Ariz.—Mr. Chairman, Mr. Hodgson made a practice of putting on some instructors. It would be interesting to hear what the results of that have been.

J. P. HODGSON.—Some months ago we took up the study of classifying ground. After the ground was classified we experimented in the different classes of ground in an attempt to standardize our drilling methods underground. It was found that in trying to standardize we bumped into old miners who, in their opinion, knew more about it than we did. So we decided to take the line of least resistance. When we found a man unwilling to learn a new method we did not fight him. We put him in a stope to work and took young men and trained them. We appointed several instructors and have them in the mine. They take young men who want to get ahead and instruct them for a period of 10 days to 2 weeks, then we supply them with a machine and equipment and put them in a drift, or raise, as the case may be. They get miners' pay and a bonus, depending upon the classification of ground upon which they are working and advance made. We have found this to be beneficial, and we have decided in the future rather than to fight our miners, to help them.

D. W. BRUNTON, Denver, Colo.—I would like to give some figures on speeds maintained in tunneling operations where persistent effort in teaching the miners who were engaged in driving tunnel headings resulted in an increase of speed from 250 ft. per month to 750 ft. per month, and in ordinary ground, after the workmen were thoroughly trained, there was little or no difficulty in maintaining speeds of from 600 to 750 ft. per month, with three 8-hr. shifts—in headings 7 ft. high by 8 ft. wide.

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THE CHAIRMAN.—This subject verges on that allied subject, efficiency engineering, with which we have been experimenting. A characteristic of that subject is that it does not take very long to find out how little you know about a great many standard operations, and that they are not done as they should be done, and are not yet as we would like to have them. I don't know whether our experience has been like that of others or not.

The Block Method of Top Slicing of the Miami Copper Co.

Discussion of the paper of E. G. DEANE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1515 to 1518.

THE CHAIRMAN (P. G. BECKETT, Globe, Ariz.).—The mining of large orebodies has in the last few years been such a big factor in the copper output of this State, and, in fact, of the whole country, I feel that we should have some valuable discussion this evening on the various mining methods described in the papers which are to be read; and I hope you will start a discussion on Mr. Deane's paper.

J. A. EDE, La Salle, Ill.—I would like to ask whether, the timbers being left in, they have any fires owing to that fact?

E. G. DEANE.—No, we do not. The timbers in the mat seem to lose life. I cannot tell you exactly what change takes place, but they do not burn readily. There is undoubtedly a chemical change in them—a change that makes them lose all strength. I spoke of the strength of the sills. When we first started, we put in 4 by 8 sills, but those sills had no more strength in 6 months than a 2-in. plank would have.

J. P. HODGSON, Bisbee, Ariz.—I would like to ask Mr. Deane if the central supply raise in the center of the block has been satisfactory; and also whether in extracting the ore they put small raises up so that wheelbarrows are not used in the stope?

E. G. DEANE.—The supply raises have been very satisfactory. We do use wheelbarrows, as our raises are spaced 50-ft. centers; it may be that later it will pay us to put up small inclined raises. But just above the 420-level, where we have been doing our slicing, it has not paid us to do this.

J. P. HODGSON.—I would like to ask another question, and that is this: From your experience in top slicing, where have you found that the overburden decreases in weight? At about how many floors downward in course of extraction?

E. G. DEANE.—We have not gotten down that far. We have taken about ten slices in some parts of the mine, and have as much weight as ever. In other parts we are slicing immediately under the capping at the present time.

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Mine Fire Methods Employed by the United Verde Copper Co.

Discussion of the paper of ROBERT E. TALLY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1545 to 1553.

CHAUNCEY L. BERRIEN, Butte, Mont. (written discussion).—Having had much actual experience with mine fires which have occurred or have been active in the mines of the Anaconda Copper Mining Co. during the past 4 years and having lately visited the fire zone of the United Verde mine with Mr. Tally, I wish to congratulate him for the success which he has had in controlling this fire and making the mine perfectly safe for his miners. By this work he is not only keeping the property in operation but is reclaiming many lost orebodies.

The ventilation system as described by him is practically the same as that used under my direction at the Mountain View mine fire in Butte in 1913 and at the Pennsylvania fire in 1916. This same system is at present being followed at the Leonard mine to extinguish a fire which has been active for 10 years although bulkheaded.

The mining system in the Butte mines at the origination of these fires was the same as that at the United Verde, namely, square setting accompanied by the use of much timber bulkheading in the stopes owing to heavy ground. The pyritic content of the ore in Butte is not as high, but on the whole conditions have been the same, with the exception of actual location of the fire in the mine.

I agree with Mr. Tally that flooding, the use of CO_2 or any gas, and the use of steam are all impracticable unless the fire zone can be absolutely sealed. His system of upcast escapement shafts in the fire zone, the use of bulkheads and fans to keep the gases away from the firemen, must be resorted to. By the method he describes actual stoping may be carried on safely if started from the bottom of the fire and if the fire zone extends to the surface. In other words, the fire zone must be isolated from the rest of the mine by bulkheads and solid ground, with no ground above which can be damaged by spread of fire or caving.

The serious fires in Butte have occurred on or between levels in the mines where it was necessary to prevent their extension upward, downward and laterally while the work of actually extinguishing them was in progress. This necessitated proper bulkheading should we lose control of the fire, the installation of ventilation similar to Mr. Tally's and the systematic drilling of holes with diamond drills for distributing water on the fire and around it.

The Mountain View fire of 1913* was extinguished by the method described by Mr. Tally and the driving of laterals parallel to the fire

*See Fire-fighting Methods at the Mountain View Mine, Butte, Mont., *Trans.* vol. 52 (1915).

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stopes. From these laterals many diamond-drill holes were drilled through which water was run onto the fire. This fire burned through stopes between the 200 and 600 levels.

The Pennsylvania mine fire of 1916 burned between the 1,000 and 1,300 levels and for a considerable lateral extent and was finally extinguished by the Plenum system and extensive diamond drilling. The diamond-drill work was in both cases the main factor. The details of this work were carried on by C. Edwin Nighman, at present Fire Superintendent of the Boston & Montana group of the Anaconda Copper Mining Co. and H. R. Tunnell, Foreman of the Pennsylvania mine, under the direction of the writer. A description of this work will appear in the *Transactions* in the near future.

As I have said before, we are at present working on the Leonard-Minnie Healy fire with the prospect of regaining much ore, if not actually extinguishing the fire. The extent of the ground bulkheaded off from the rest of the mine is 600 by 400 ft. lying between the 600 and 1,300 levels.

This territory is filled with gases and an uncertain extent of smoldering fire. Our plan is to drive lateral drifts through the fire zone, keeping them in solid ground. From these laterals holes are being drilled with diamond drills into all stopes below, so that eventually we will have water entering all the stopes at 10-ft. intervals. We have been at this work about 1 year and have entered and reclaimed practically all of the 600 and 700 levels. We have also entered the 800 halfway and up to date the orebodies regained have warranted the installation of electric haulage on the 800. In doing this work we are using the Plenum system, and so far we have not had to resort to the use of Draeger helmets. While this work was really started as a matter of safety for the future of the mine, we are regaining orebodies at no greater expense than required for development of ore on new levels.

Our experience with fires in Butte has given us some thought as to their causes and led us to install all possible safeguards against their recurrence and to assure the necessary protection from new fires and the means to extinguish them.

The possible causes of mine fires are as numerous as of surface fires and one could set down a long list of them and the ordinary precautions for prevention. All of these should be known to the superintendent of any mine, and the subject is covered very thoroughly by George J. Young in Fires in Metalliferous Mines.*

Over a period of about 10 years in 20 or more operating mines in Butte, the *actual* causes of mine fires have been: lighted candles left on or near timbers, short-circuiting and overheating of electric wires, switches, etc., due to carelessness in upkeep and operation, oily

* *Trans.*, vol. 44 (1912).

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waste, spontaneous combustion from foreign substances in gobs, and possibly incendiarism.

We realize that mine fires are possible even though the greatest precautions in the way of regulations, warnings, etc., are taken, and in the past 2 years we have installed every protection against the spread of fires, facilities for fighting them and means of rescue.

All of our main operating shafts are downcast, around the collar of each shaft we have 2-in. sprinkler lines; all water lines, air lines and electric cables are in downcast shafts, water-storage tanks of 3,000 gal. or more have been put in every 400 ft. along the shafts with pipe lines from them through main drifts, laterals and many stopes; water may be turned into all air lines also. All underground fan stations, transformer and switch rooms and electrical apparatus have been made fireproof for safe distances from them, overload and no-voltage release and auto starters are used on fans; all connections between mines are cut off by concrete bulkheads and iron doors except where stopes are continuous; a firebug (or watchman) goes through each shift boss' run in all mines after each shift; all electric switches are disconnected at shaft stations between shifts; telephones are used in all large mines; surface fans exhaust only; metal receptacles are used for oil waste and refuse and are in safe places; first-aid and Draeger helmet crews have been trained at each mine and hold positions where they can be notified of trouble immediately; signs denoting directions to other mines and main shafts are posted on all levels; an "out of the mine" danger signal is used; smoking in the mines is prohibited; metal sconces for candles are used entirely and failure to use them means discharge of man; mining methods are being adopted to eliminate as much timber as possible, and, finally, two fully equipped rescue stations on the surface are maintained.

At each of these rescue stations we have 30 Draeger oxygen helmets with accessories, lung motors, an automobile, a first-aid man on each 8-hr. shift, a smoke room for training and a man in general charge over all first-aid and rescue work. Classes are being held continuously and the number of capable firefighters is increasing. We also have a fire superintendent who acts with mine foremen in all underground fire work.

We have had six serious mine fires during the past 6 years and they have been extinguished or bulkheaded without loss of life except in the Pennsylvania mine fire of 1916. The loss of lives in this fire was due to the failure of the men in following directions given them for their safety. Work on all these fires extended over months. There have been about 12 other fires discovered and extinguished in a few minutes, or an hour at the most. This success has been due to preparedness, the good work of the helmet men, and extreme watchfulness. The expense of installation of any system or equipment for fire protection and fire fighting will be offset many times if such trouble arises.

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C. W. GOODALE, Butte, Mont.—I wish to say that I have before me a discussion of the fire that occurred in the Pennsylvania mine in Butte, by Messrs. C. E. Nighman and R. S. Foster, but it is quite long and the conclusions are very much the same as those given by Mr. Berrien in the discussion which has just been read. I will say briefly that the Pennsylvania fire started Feb. 14, 1916, about 8:45 p.m., at or very near the 1,200 air-shaft station. This fire resulted in the loss of 21 lives; however, had these men followed the instructions of those sent to warn them, a maximum of nine, and probably only seven, fatalities would have occurred. At the time the fire occurred the main working shaft was upcast, and while this condition caused no loss of life, considerable discomfort resulted from the presence of smoke and gases, and rescue work had to be abandoned until this shaft was made downcast, which was accomplished by placing a suction fan over the air shaft. It should be stated that when the fire occurred preparations were under way to make the working shaft a downcast.

It is extremely important that working shafts be downcast wherever possible, for the following reasons: (*a*) In case of fire, ordinarily men can come to the shaft and be hoisted without danger of suffocation (in some cases action of fire has been known to change the direction of air in shafts); (*b*) Where cold air is coming down a shaft, the tendency of the ground to slough is greatly lessened; also, timbers last much longer when not exposed to hot, humid air coming out of the mine, and as working shafts must bear the strain of traveling skips and cages it necessarily follows that they should be favored where natural conditions permit. Furthermore, air shafts can usually be retimbered without curtailing production.

On the other hand, during the winter months in cold climates, ice collects in downcast shafts even where there is very little moisture. It is necessary to clear the ice by chopping it out, which is a disagreeable and somewhat dangerous task. Furthermore, it is hazardous and uncomfortable to do repairing in downcast shafts during cold weather.

At the present time the working shafts of the Anaconda Copper Mining Co. in Butte are downcast wherever possible.

THE CHAIRMAN (PERCY G. BECKETT, Globe, Ariz.).—I think, as Mr. Tally said and as Mr. Goodale has brought out, mine fires are exceedingly disagreeable and dangerous things to have in a mine; and I think it is a vital subject for us all. There are several big mining districts having fires burning in those districts for a long time. It is one problem to keep the fires under control, and another problem full of importance to us all to prevent fires from starting from the numerous sources from which they can start.

GERALD SHERMAN, Bisbee, Ariz.—We have been unfortunate enough to have some fires, and we have used very much the same methods which

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have been described by Mr. Tally and Mr. Berrien. I am not sure that the members here are familiar with Mr. Berrien's paper of last year; but I think there is a distinction between what Mr: Tally is trying to do or doing and what Mr. Berrien is doing. From the shape of the orebodies in the United Verde, it is very much more difficult to put them out than in Butte. It is very difficult to get at them. The ground over the fires is soft, crushed and open; and I doubt if drill holes could be put in to get water on the fires. I think Mr. Tally's effort, although he has put out many fires, has been principally toward controlling them by the Plenum system. He recovers the ground by blowing the fire away from the working places and driving the gases to the surface through broken ground or some passage which does not interfere with other work. This does not put the fire out unless it is kept away from other timbered ground until it goes out of itself. The only way I know by which fires have been put out in this country is by putting water on them. Mr. Tally has put fires out by that method, and Mr. Berrien has also. In the cases at Butte, most of the veins were in comparatively hard rock. They are not vertical, as I remember it, but very steep. By drilling from the hanging wall, which is safe to work in, you can pour water on the fire successfully, and that is the way we did it. In one of our fires, we were lucky enough to get water on top of the fire in such a way as to drown it, and it was finally put out so we could get in the ground and prove it. Another was kept down by putting water on it until we were able to get to the fire area and by following the smoke to dig it out, but that was a small fire. In one of our fires, which has been burning for some years, and which is now isolated, we tried to drill holes over the top with diamond drills, but it was unsuccessful, as the ground was too much broken. We were never able to get near the point we were driving for. I think the precautions which Mr. Berrien has suggested are very well worth carrying out. All of our fires have been spontaneous, as far as we know, and they have all occurred, with one exception which is rather doubtful, in the centers of filled and abandoned stopes, which were inaccessible, and we were not able to reach them quickly. We had to isolate them and cut them off from active workings and keep them from spreading. I think they are undoubtedly caused by the heating of sulphides in stopes where they are loose and granular, and where there is little ventilation. I believe if raises could be kept open for free circulation of air, the heat would be carried off so that the temperature would not rise to the point of ignition, but I am not sure of it. In some of our shafts, we are laying pipes and sprays to water the shaft in case of fire, and also putting up doors to cut them off and stop circulation from the shaft, but we do not care to say very much for the present.

J. P. HODGSON, Bisbee, Ariz.—Mr. Chairman, from my experience in mining, covering a period of about 30 years, I would say that I think

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the most prolific source of mine fires is the use of candles, owing to the carelessness of the men in leaving what we call "snuffs" around their working places. I know of at least two mine fires in Michigan where I was operating that occurred in this manner. One of these we put out with water; in the other we sealed the shafts and shut down the mine for 3 weeks before we extinguished the fire. Regarding our experience in copper mining, as Mr. Sherman has said, we have had a lot of experience in fighting fires, but we are not yet sure as to which is the better method. One thing I might mention in this connection that might be of assistance to others is that we keep on hand a 15-hp. electrically driven blower mounted on a truck, together with 1,000 ft. of wire for connections and 500 ft. of galvanized 10-in. blower pipe and a roll of brattice canvas. This equipment is kept in a certain place. When a fire alarm is turned in this equipment can be sent underground and put in working order in from 30 to 45 min. With this equipment we can force back the smoke in the drifts and sometimes succeed in reaching the fire, and in putting it out before very much damage is done. We have a fire in the Lowell mine that has been burning for over 6 years and we have gradually encroached on the fire area by putting in concrete bulkheads and thus getting ore that could not be mined before on account of the fire. We are at the present time on the 800-ft. level of the Gardner mine driving a series of drifts over a fire area similar to those explained in Mr. Berrien's paper. We are putting water in these drifts so that it may percolate through the broken ground to the fire zone below. This fire is in an old sulphide area which has been stoped out, and I believe in a year or two we will succeed in extinguishing this fire.

THE CHAIRMAN.—Is there any more discussion on this paper? I would like to ask Captain Hodgson from his experience fighting fires at Bisbee what he considers the main fundamental precautionary methods that should be taken in the average mine. Is it more a question of shafts in the right place, upcasts and downcasts, or water lines laid throughout the mine, or a question of blocking the mine off into sections that can be closed quickly, or what?

J. P. HODGSON.—I believe that all of the things you have mentioned are very good, but in such mines as the Copper Queen it is almost impossible to do all of them in every case as the property is so extensive and as a usual thing the fire breaks out where you don't expect it. After careful investigation we adopted this blower and have it in reserve so that we can get quick action after an alarm is turned in. It is lowered and in operation in from 30 to 45 min. in any part of the mine, and we believe it is very good for our conditions. Also, at the Copper Queen, we have fire doors to block off the different parts of the mine so that in case of a serious fire in one part it can be cut off and the balance of operations

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proceed, but I fear I could not say which is the best way. All the things mentioned are good but the fire usually starts in the wrong place and without giving due notice. We also believe that our method of ventilation, keeping the mine under pressure, materially assists us in keeping back our fires.

C. A. MITKE, Bisbee, Ariz. (communication to the Secretary*).—I am interested in Mr. Tally's methods of fighting mine fires at the United Verde Copper Co. because he has developed a system of mine ventilation which has made possible the mining of orebodies that are adjacent to fire districts, as well as orebodies which are actually on fire. Of course, when preliminary work is being done in the way of development of these fire areas there is considerable heat to contend with; but after the ventilation has been arranged the miners are able to stope out the ore with almost the same comfort as in ordinary orebodies which are not on fire.

Gardner Fire District.—Since the pressure system of ventilation was successful at the United Verde Copper Co., it was adopted at the Copper Queen.

There are similar mine fires in the Copper Queen, as, for example, the 9-1 district of the Gardner division where a fire has been known to exist for some time. About 2 years ago a large amount of carbon dioxide came from this district on the 1,000 level, in which three men were overcome. Steps were taken to inclose it by means of doors, and to have air pressure of about 7 lb. per square foot surrounding the district to prevent its spreading. A raise was driven between the 800 and 900 levels and intermediate drifts separating the fire country from live workings. On the 800 a drift and a number of crosscuts were driven over the hot area. Water was run into these intermediate drifts and crosscuts so as to cool the country rock. An exhaust fan was used to draw the air from the top of the drifts and allow fresh air to come into the headings. At present the amount of gas and heat coming from this district has been decreased and it is believed that in a short time the district will be cool enough so that the usual ventilating system will continue to cool it without the continued use of water.

Cananea Consolidated Copper Co.—About 2 years ago there were several fires in the Veta Grande and Oversight mines in Cananea. These fires were located principally in timbered drifts and stopes. While some of them were easily extinguished, nevertheless one of them in the Veta Grande required considerable work done by men using oxygen helmets. The greatest difficulty was experienced in getting the fire under control, on account of there being no comprehensive scheme of ventilation.

Rules.—There should be a set of rules at every large mine which

* Received September, 1916.

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should prescribe first for the safety of the men and then, for combating the fire. The importance of this may be illustrated by the Holbrook mine fire which occurred 3 years ago. As two men were taken out of a raise on the 500 level when they were overcome with gas, we asked the foreman if he had a pulmotor. He said "it is here in this drift." The pulmotor had been there for 20 min. and it was used effectively on a shift boss and assistant superintendent. When helmets were required, they were available immediately. The same was true with canvas, axes, saws, etc. During the night the fire was sealed up entirely and the mine was in operation the next day. The importance, therefore, of having all necessary supplies on hand in case of fire cannot be overestimated.

Cost and Extraction in the Selection of a Mining Method

Discussion of the paper of C. E. ARNOLD, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1519 to 1522.

THE CHAIRMAN (PERCY G. BECKETT, Globe, Ariz.).—When you are at Miami tomorrow, you will see a very interesting comparison on the surface of the two mining methods that are being employed there largely. In the case of the surface over the Inspiration mining method, you won't see very much left of the surface. Everything has gone down. In the case right at the line of the Inspiration ground and the Miami, you will see that the surface over the top slicing which the Miami company has been doing in that country has been riding down evenly and uniformly; and it gives a very marked comparison of the two types of mining, and, to a certain extent, of the different degrees of waste dilution. I would ilke to hear from Mr. Maclennan on this subject.

F. W. MACLENNAN, Miami, Ariz.—A paper based on percentage extraction by shrinkage stoping is a difficult one to discuss for the reason that it is hard to arrive at extraction figures until a section has been completely mined out. In the Miami mine we are using a method of shrinkage stoping with narrow stopes and pillars. This ore has been completely broken up and we have started in at the far end to draw off the ore mass maintaining the top of the broken ore dipping at an angle of 60° away from the direction in which we are retreating. A very important point I think about drawing a large mass of ore down at an angle rather than trying to maintain the top of the ore more or less horizontal is that drawing at a steep angle has a tendency to create a horizontal movement in the ore mass as well as the vertical movement of the ore traveling down to the drawing off chutes. We know that this takes place on account of the horizontal movement of marked

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blocks which were placed in the broken ore mass at the time of stoping. This horizontal movement tends to cut off and break up pipes and funnels which are liable to open up through the ore from the drawing off chutes, allowing waste to drop down to the chutes before the broken ore surrounding the open pipe has been drawn in.

I noticed one remark made by Mr. Lehman in presenting his paper, to the effect that ore is drawn off from the finger raises until capping appears in these raises. When this occurs the drawing off is considered completed and these raises are sealed. Our experience in drawing off ore in the Miami mine would indicate that this is not always reliable. In many cases we have had chutes piped clear through to the surface, allowing capping to drop down to the chute when only 10 or 15 per cent. of the expected tonnage from this chute had been drawn. When this happens we seal the chute off temporarily and draw other chutes surrounding it for a few days and then resume drawing in the chute in which the capping appeared with the result that, usually, after drawing 15 or 20 tons of capping the ore starts to run again and may continue to run for several hundred tons of clean ore before capping again appears. For this reason I believe it is very important to establish a horizontal movement in the broken ore mass in order to break up these vertical pipes to the capping and also to help break up the thin pillars which were left between the stopes, and I also believe it is important to have the tonnage record from each individual chute so that an indication is had of whether capping appearing in a chute is only temporary or whether it marks the final stage of ore drawing. I think both of these points have a very important effect on the final extraction and dilution of the ore drawn.

Power Plant of the Burro Mountain Copper Co.

Discussion of the paper of CHARLES LEGRAND, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1531 to 1538.

THE CHAIRMAN (B. B. GOTTSBERGER, Miami, Ariz.).—I think this experience of the Burro Mountain Copper Co. with oil engines as described by Mr. Legrand comes at a very opportune time. The fuel almost universally used in the mine power plants of the Southwest is oil. Additional economy in its use for power production is much to be desired. In the case of the Miami Copper Co., we have lately found it necessary to enlarge our power plant and, based on the experience of the Burro Mountain Copper Co., we have decided to install oil engines similar to those in their plant. I think it would be very interesting to have some discussion on this subject. Mr. Langton, did you tell me you had prepared some remarks?

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JOHN LANGTON, New York, N. Y.—This is an exceedingly useful paper. The plant it describes constitutes a long step in advance, and a bold step; and it is pleasant to know that the courage needed to undertake it has been rewarded by success. We have known that Diesel engines have been gradually and carefully developed by at least three manufacturers in Europe to units of large size, and of demonstrated reliability with the fuel oils there used. We knew that in getting the fuel oil into the cylinders and igniting it, European practice had dealt with fuels more difficult than viscous California oil. But where the same viscosity is accompanied by a different composition, the parallel ends at combustion. All contracts for California fuel oil as used for boilers and furnaces in Arizona contain a clause allowing a small quantity of sand. This oil is the only form of fuel we can depend upon, both in quantity and price, on which to base reliable estimates of the fuel-oil costs of operating Diesel engines in Arizona. As compared with the known conditions of European practice, there remained the serious question whether the sand allowance, together with any other ash content of asphaltum-base California oil, might not form enough grit to cause prohibitive wear in the cylinders. The Tyrone plant has done us the very great service of definitely determining this cardinal matter.

The Tyrone plant has also settled another minor but weighty point that has heretofore been a troublesome item in estimating operating costs; that is, the quantity and cost of lubricating oils consumed. The table of costs in Mr. Legrand's paper shows that lubricating-oil cost is more than 8 per cent. of the fuel-oil cost at Tyrone; and since the average cost of fuel oil at Arizona points is only $1.45, instead of the $1.85 to $1.96 per barrel stated for Tyrone, the figures he gives for lubrication would add not less than 10 per cent. to the fuel-oil cost in an Arizona plant. Considerable as this is, it is only half of what we have heretofore felt obliged to allow in estimating the cost of lubrication. This notable reduction is in part quantity and in part price per gallon. By inadvertence the quantity of lubricating oil and its cost per gallon were omitted from Mr. Legrand's paper. He has given me these figures with permission to state them. The monthly consumption of one engine running continuously is:

175 gal. of cylinder oil at 42c. per gallon.
170 gal. of engine oil at 28c. per gallon.

At the 1,250-b.hp. sea-level rating of these engines, these figures correspond to:

$1.16 cost of lubrication per brake horsepower-year of rating and
1.36 grams of lubricating oil per brake horsepower-hour of rating.

I have stated the quantity of oil in grams in order to compare with the

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figures for estimation given by two European manufacturers. They were:

2 to 3 grams given me by one manufacturer and
2 to 2.5 grams given to Mr. Legrand by another manufacturer in another country.

The experience at Tyrone does not cover the question of what sulphur content is permissible in the fuel. I understand from Mr. Legrand that there was less than 1 per cent. sulphur in the fuel oil at Tyrone. In Europe, I found that sulphur was a great bugbear, but from such inquiries as I was able to make, and from what others tell me of their inquiries, there is a considerable difference of opinion as to what sulphur content is permissible. One maker stated explicitly that up to 1 per cent. sulphur gives no trouble in the cylinders, provided there is a suitable quality of cylinder oil, but that 2 per cent. is the practical limit of sulphur with any quality of cylinder oil. On the other hand, another maker referred to their engines running satisfactorily on Rumania oil with 3 per cent. sulphur. I was informed that the bad effect of sulphur seems to be wear of the cylinder, not by corrosion, but indirectly by destroying the lubrication. It is possible that the low cost of cylinder oil at Tyrone, both in quality and quantity, may be due to the low sulphur. That, however, does not affect the value and certainty of the Tyrone figures to us in Arizona, where the fuel oil is identical with the Tyrone oil.

Thanks to this valuable paper, we can now estimate with confidence both fuel oil and lubrication costs. Operating labor can be foretold with a close degree of accuracy, and the minor item of supplies with a sufficient degree of approximation. The interest and amortization charges are definitely fixed, as a matter of opinion or policy based on the conditions of the industry that the power plant serves. There only remains the very important item of maintenance. That is not yet determined at Tyrone. But when it is determined for that plant, I fear that it will be of but slight help to others. In plants consisting of a few large and costly units, individual good or bad fortune must always be a very large factor in repair costs. In estimating maintenance, all we can do is to try and be on the safe side and allow for a larger cost than we can imagine to be possible. Whether the results show surplus or deficit in this item is a question of good or bad fortune at the plant, and of whether the estimator is a pessimist or an optimist.

S. J. KIDDER, Mogollon, N. M.—There is one question I would like to ask as to the cost of lubrication per brake horsepower year. It was stated that it was based on 1,250 hp. as the load of the engine.

JOHN LANGTON.—The rating of the engine, I might have said, in order to estimate the carrying load, the cost of oil being comparatively small.

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A Comparative Test of the Marathon, Chilean and Hardinge Mills

Discussion of the paper of F. C. BLICKENSDERFER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1333 to 1348.

THE CHAIRMAN (B. B. GOTTSBERGER, Miami, Ariz.).—On your trip today through the Inspiration and Miami mills you have seen in actual operation the machines which represent the changes adopted in grinding methods in the Miami district from the beginning of operations in March, 1911. In one section of the Miami plant are three Chilean mills not yet discarded, representing the first machines adopted. In another section, pebble grinding is still in operation, while in the balance of the plant, pebbles have been replaced with iron balls. In all cases these machines are handling a feed which is the product of roll grinding down to about ¼ in. in size. The latest step in grinding in the district, you saw at the Inspiration plant, where in Marcy mills a feed, the larger pieces of which remain on a 2-in. opening, is being reduced in one operation to a product of which only about 3 per cent. remains on the 48-mesh screen. A detailed comparison of the methods used at the two plants would be very instructive but our own work at Miami has not yet reached a point where such a comparison is possible, the reason being that our final product is not yet fine enough. By means of larger Hardinge mills operating in closed circuit with Dorr classifiers soon to be installed in one section of the plant, we hope to obtain under properly balanced operating conditions the same end result as at Inspiration. It might interest you, however, to know some of the results we have obtained here by the substitution of balls for pebbles. With pebble grinding, using three 8-ft. by 22-in. mills, followed by one 8-ft. by 66-in. mill for regrinding the coarse product of the primary mills, we handled 636 tons per 24 hr. in one section of the plant. Substituting balls for pebbles in the first operation and using two mills in place of three we obtained a tonnage of 714 tons to the section. With the use of balls both in the primary mills and in the regrinding mills, the tonnage for one section was raised to 822 tons per day. Coincident with this increase in tonnage, we also obtained a greater efficiency. The use of pebbles in all mills showed 21.5 per cent. plus 48-mesh material in the final tailing, which figure was reduced to 13.9 per cent. for balls followed by pebbles, and to 4.6 per cent. with the use of balls both in the primary and regrinding mills.

We should be able to have an interesting discussion on this subject this afternoon if some of the mill men present will participate, particularly

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as we have before us Mr. Blickensderfer's paper, presenting what may be one step more in the evolution of fine grinding.

R. B. YERXA, Miami, Ariz.—I would like to ask if the Marathon mill can operate at such a tonnage as to produce the same product in fineness as that produced by the Hardinge and Chilean mills?

F. C. BLICKENSDERFER.—At the time this paper was published, such had not been the case. However, the Marathon mills of the larger type, now in operation at the Burro Mountain Copper Co., are grinding considerably finer than any product mentioned in this paper. Those mills at Tyrone are, roughly speaking, operating on 300 dry tons per 24 hr., and are grinding this down from 15 to 18 per cent. on plus 48. The power consumed is about 48 and the energy units corrected for tonnage are 130,000.

A. P. WATT, St. Francois, Mo.—I was much interested in reading Mr. Blickensderfer's paper comparing the efficiency of the Chilean, Hardinge and Marathon mills. The question of fine grinding is now receiving considerable attention in southeast Missouri. At present the jig middling, which is through 10-mm. on 2-mm., is generally crushed in rolls. One company, however, is using a Chilean mill and another company is using a Huntington mill for this purpose. Recently, however, the use of pebble and ball mills has been considered for solving the problem of fine grinding. Hardinge and Marcy mills are in regular operation in the district for the grinding of middling and two companies are to install Marathon mills for the same purpose.

By reason of our grinding problem I was much interested in Mr. Blickensderfer's paper, but as I read it a few questions arose in my mind. One was that in the test the Marathon was using metallic rods and was being compared against the Hardinge which was using pebbles. It is well known that the use of steel pebbles—so termed—in the Hardinge mill increases its efficiency. I wondered if that fact had been given consideration in later tests by running the Marathon against the Hardinge, using steel pebbles instead of flint pebbles in the Hardinge. I noticed that the Marathon feed contained about 63 per cent. solids while the Hardinge mill feed contained but 40 per cent. Would not an increase in the percentage of solids in the feed to the Hardinge increase the efficiency of that machine?

I would like to ask as a matter of information whether the rods in the Marathon mill still retain their cylindrical shape after continual use or if they become elliptical? Also, do the rods retain the same diameter at both ends after use, or do the rods at the feed end wear more than at the discharge end?

F. C. BLICKENSDERFER.—In answer to the first question, as to whether or not we have tried steel balls in the Hardinge mill, I will say,

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"no." The idea of using steel balls in cylindrical grinding machines was just being introduced. The second question regarding the wear of rods is one which I failed to make sufficiently clear. The rods used in the small Marathon mill of 7-ft. length, wore almost uniformly from end to end until they reached a diameter of ½ in. After that some of them would flatten out, take an elliptical shape, and all rods would wear with a slight tapering toward the feed end, due no doubt to the fact that the particles of coarse feed kept the rods at a greater distance apart in the region of the head end than at the discharge end, so that the surfaces of the rods at the head end were constantly exposed to rock. The tail end was occupied by finer particles of pulp. Lately, in our present practice, due to abnormal conditions regarding the supply of steel, we have not obtained successful results by using the present supply of rods. This, I think, is due to poor steel. We have had rods 1 in. in diameter break in two. We have had rods nearly ½ in. in diameter go into the shape of an ellipse, and then again we have had defective castings. These are mechanical defects that are well on the way to perfection, and will not seriously retard the ultimate success of the mill.

R. B. T. KILIANI, New York, N. Y.—Mr. Blickensderfer's paper is of considerable interest as it gives the first published and authoritative data on the operation and performance of the Marathon mill and to this extent it may be considered an interesting contribution to our knowledge of milling machinery. No data are given as to the metallurgical results obtained with the different machines, and to this extent, it is not conclusive as to their relative suitability for the work under consideration. The extraction and grade of concentrate obtained by the use of various types of grinding equipment are naturally of at least equal importance to their actual mechanical efficiency. In other words, the net profit per ton of ore handled is the final criterion of economic efficiency. The mills were not operated at the same time, and no attempt was made to vary the conditions of operation to obtain the maximum efficiency from each, except possibly in the case of the Marathon mill. For these reasons, it is not a "comparative test," although, as above mentioned, thanks are due the author for some interesting data.

The Chilean and Hardinge mills were operated together for a period of 63 days, but after the first 30 days, it became necessary to shut down the Hardinge for relining, as it was started with an old lining. Owing to the type of lining used, flint pebbles set in cement, a longer time should have been allowed for this work, and especially for the cement to set, than the 2.36 days stated by the author. The short life of 159 days for the lining of the Hardinge mill may be ascribed to this fact, as well as other conditions which will be taken up later. The actual percentage of time lost through all delays in the operation of Hardinge mills at other plants

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is well under 1.5 per cent., when proper care is taken that they operate under conditions which have been found to be best. The life of the same type of lining at the plant of the Arizona Copper Co., immediately adjoining the Detroit Copper Mining Co., is as high as 400 days, with an average life of over 200 days.

Two other conditions to which may be ascribed the short life of the lining are the dilution of the pulp and the speed at which the mill is operated. It is a well-known fact that for most efficient operation of a pebble mill, the solids in the pulp should be in the neighborhood of 60 to 70 per cent., and under these conditions, the wear of the lining is very much reduced. The speed of the mill of 29 r.p.m., mentioned by the author, is too high for this type of lining, since the action of the pebbles is such that they are thrown across the mill and strike the lining instead of acting upon the pulp. This, of course, causes the lining to wear out faster than would otherwise be the case. These two conditions of pulp consistency and speed of the Hardinge mill also affect the consumption of pebbles, which is very considerably higher than that at the No. 6 concentrator of the Arizona Copper Co., which was 1.334 lb. per ton ground, for the year 1914, as against the figure of 2.33 lb., as given in the paper under discussion.

More important, however, than the effect upon consumption of lining and grinding mediums of these two factors of speed and pulp dilution, are their effect upon the capacity obtained from the mill. It has been found that a dilute pulp with a high percentage of moisture, tends to very much decreased capacity and somewhat higher power. Curves and other data showing the effect of pulp dilution on capacity, fineness of grinding, efficiency, and power have been published for the past 7 or 8 years, and it would therefore be merely a repetition to give these in this discussion. Suffice it to say, however, that the efficiency of pebble mills has been increased over 50 per cent. by thickening the pulp fed to them. Had experiments been made to ascertain the proper speed and pulp density, the results with the Hardinge mill would have been much more satisfactory.

As already mentioned, no information is given as to the metallurgical results obtained with the different machines, other than the statement that the total recovery after the Marathon mill "was almost exactly the same" as with the Chilean mills. It is a well-known fact that the recovery by gravity concentration when grinding in Chilean mills is lower than when the ore is ground in pebble mills of the conical type. This does not necessarily hold true when flotation follows gravity concentration, but it is found that the grade of the concentrate is lower, and that it contains a higher percentage of insoluble matter, when grinding in Chilean mills. The Marathon-mill discharge, however, owing to the fact that it does not contain very much minus 48-mesh material, requires regrinding in some other type of machine in order to obtain the extraction and grade of con-

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centrate resulting when grinding the same ore in the Hardinge mill. By referring to the author's data, we find that a considerably greater proportion of the Hardinge-mill product is finished than is the case with that of the Marathon mill.

Mention is made in the author's paper of the fact that metal pebbles are being used in place of flint for grinding in Hardinge mills, and the question arises why these were not tried at the plant under consideration, as it is a known fact that the efficiency of the machine is increased by their use, provided the same volume is used as of flint pebbles. It has also been found that they are cheaper than flint, considering all items of expense, except in certain exceptional cases. Another point in favor of their use, is the fact that increased recovery can be obtained without the expected higher grinding cost. Another fact which might be mentioned at this point, is that operating the mill in closed circuit with a Dorr, or other similar classifier, increases the efficiency and therefore decreases the grinding cost. It might be claimed that a similar increase in efficiency would be obtained by operating the Marathon mill in a similar manner, but this, to the writer's mind, is doubtful, owing to the much greater circulating load, due to the fact that this machine is unable to grind fine, except in larger machines where the capacity drops and the power increases very rapidly.

For a true comparative test, both machines should be operated side by side, taking the same feed and delivering the same final product, and the metallurgical results obtained with each taken into consideration. The Hardinge mill should also be operated with a metal lining and should use metal pebbles, since it would then be using the same class of grinding mediums as its competitor. It would naturally be absurd to suggest the use of a flint lining and flint rods in the Marathon mill, but the other has been proven to be entirely practicable. Should such a test be made, it would be interesting to compare the results with those reported in Mr. Blickensderfer's paper.

F. C. BLICKENSDERFER.—It is mentioned in this paper that the total unit tonnage of 440 tons per 24 hr. was sent through the small Marathon mill. On the other unit we had three Chileans in operation. This test extended over 4 days, so we were reasonably safe in drawing our conclusions. There was a difference of less than 1 per cent. in the extraction of the two units, operating as I have outlined to you. It has always been our custom to assay the mesh sizes of all screen analyses made in the concentrator. I am not able to answer the question as to what condition the material on the screen sizes was in for concentration. There are no accurate methods for separating mineral from gangue in exactly the same manner as the gravity machines do. The nearest thing I know of is by means of a heavy solution of mercuric iodide and potassium iodide, with a specific gravity of about 3.20. That portion of ore which sinks is concentrates;

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that which floats is tailings. There is a very definite separatory action in this method. In incorporating this method of separating the concentratable material from the gangue in the screen sizes, much difficulty was encountered, especially in treating slimes, so that, to my knowledge at present it is an open question as to the determination of the free mineral in the screen sizes.

The Hardinge mill slimed slightly less of the total material fed to it. The Hardinge mill slimed a little more of the mineral fed to it, as compared with the Chilean and Marathon. The method of determining to what extent the mill slimes mineral was as follows: All of the material in the feed which remains on a 200-mesh screen was assayed and its copper content obtained by multiplying per cent. weight by per cent. copper. Then all of the material in the product which remains on a 200-mesh screen was assayed and its copper content similarly obtained. Subtracting the copper content thus obtained in the tails from the copper content in the heads gives the copper content of the material actually slimed in the process of grinding. Proof of Statement that Hardinge Mill Slimes More Mineral than Chilean.

Assume Weight of 100 Tons

	Tons	Per Cent. Copper	Tons Copper
Total + 200-mesh pulp in Hardinge feed.....	91.63	1.03	0.944
Total + 200-mesh pulp in Hardinge product....	68.15	1.01	0.688
Material actually crushed to −200-mesh.....	23.48	1.09	0.256

Assay of material actually crushed to −200 = 1.09 per cent. copper.

And in case of Chilean mill:

	Tons	Per Cent. Copper	Tons Copper
Total + 200-mesh pulp in Chilean feed.....	83.46	1.15	0.960
Total + 200-mesh pulp in Chilean product....	58.98	1.19	0.702
Material actually crushed to −200-mesh.....	24.48	1.05	0.258

Assay of material actually crushed to −200 = 1.05 per cent. copper.

Hence, the Hardinge mill has slimed more mineral to produce an assay of 1.09 per cent. copper than the Chilean in producing an assay of 1.05 per cent.

S. J. JENNINGS, New York, N. Y.—I would like to ask Mr. Blickensderfer a question. In his explanation of the action of the Marathon mill, he shows a diagram which does not seem to differentiate between the two possible actions that take place in the mill—one is a crushing action and the other is a grinding action. If we rotate a cylindrical mill somewhat rapidly so that the pebbles are entrained on the up side and fall, we would have a crushing action. If the rapidity of rotation is not sufficiently great to entrain the pebbles on that side but merely to raise them up sufficiently and allow them to go back, you have a continuous grinding

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action. I would like to ask Mr. Blickensderfer if the Marathon mill was run at such a speed as to get a crushing action, or a grinding action. I assume from the fact that the rods wear evenly, with the small difference he has noted, he has merely a grinding action in his mill; but I would like to know that definitely.

F. C. BLICKENSDERFER.—We have going on in the Marathon mill both a grinding and a crushing action. In the explanation referred to, I purposely inserted the words "strike" and "rolling together." Of the charge of rods, when observed inside the revolving mill about one-third is flying through the air and the other two-thirds are exercising a rolling action on one another. This is better illustrated by standing near a Marathon mill and hearing the constant striking of rods as they leave the periphery and travel across to the opposite part of the mill.

R. S. HANDY, Kellogg, Idaho.—In the Marathon mill, in the matter of keeping them parallel—when the rods wore to small diameters, was there any difficulty in the rods getting crossed and not maintaining parallel relationship?

F. C. BLICKENSDERFER.—We found that about 8 days' time was the limit that these mills could run without having the rods changed. The rods in the 8 by 12 mill catch hold of something and they twist up and form all kinds of grotesque shapes; it is time to stop the mill. The operator should keep close watch and remove the small twisted rods whenever a few appear in the top of the charge. In most cases, we straighten them out and put them back in again. The smaller ones are thrown away, and the load of rods brought back to about 7 tons, which is determined by filling the mill exactly one-half full of rods.

R. B. YERXA.—I would like to ask what the tonnage was on the operation with Chilean mills when the test was made to find out what the recovery was between the two different methods of grinding.

F. C. BLICKENSDERFER.—The tonnage was approximately the same. We had plunger-type feeders which governed the tonnage by the distance the gate was opened. We carried those gates at the same height, and the mill bins being filled at the same time, the tonnage of the two units would be approximately the same—within 5 per cent. I would say.

THE CHAIRMAN.—I think we all ought to be grateful to Mr. Blickensderfer, in the first place for this paper, and again for his very full answers to our questions. I think that at times when we get to discussing a question of this kind, we are apt to give the impression that we may be criticizing, but I hope Mr. Blickensderfer does not feel that way in this case, as I believe he has brought something forward that is going to make us think. Present-day methods of concentration for handling the so-called disseminated copper ores call for fine grinding and it will only be

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by comparison of results obtained at different plants that progress will be made.

Referring to the results in the present paper, I am impressed by the fact that taking −48-mesh product as the desired result, the Marathon mill yields a comparatively coarse product—one which in our case would not be suitable for the work we have to do. If Mr. Blickensderfer would not mind answering one more question, I would like to ask if he thinks putting the mills in closed circuit with drag classifiers would help in the fineness of the product.

F. C. BLICKENSDERFER.—In the efficiency comparison of these machines, I call your attention to the fact that the Marathon mill when running with a tonnage of 440 turned out an efficiency, corrected for tonnage, of 143,000. When it was running as a finishing mill, taking 235 tons, in 24 hr. its efficiency was 107,000. Now, I take it from those two figures that if the Marathon mill was filled up, so to speak, with coarse material from its own discharge, much better results could be obtained. I have never seen any figures on those mills grinding in closed circuit.

C. W. MERRILL, San Francisco, Calif.—I am not going to ask Mr. Blickensderfer a question. I am going to congratulate him. We men who have been studying the evolution of fine grinding in connection with gold milling for the past 20 years have always rebelled a little at the spherical contact—the point contact that we have had to contend with in all of the ball mills. We have felt that it was an extravagance of power that eventually would be eliminated; and when we prepared our paper for the International Engineering Congress on the subject of fine grinding, we ventured to predict that the knowledge we had of the small Marathon mill would result in just such work as Mr. Blickensderfer has explained to us today. I think there is a new principle involved in this machine—that of linear contact, as compared with spherical contact—and I want to express my appreciation of the work Mr. Blickensderfer has done.

F. C. BLICKENSDERFER.—I feel as though I ought to make a few remarks. I should like to thank the Institute as a whole, and the members, individually, who have been so kind as to bring up these points. They seem to have brought out a better understanding regarding this mill. In publishing this paper, I wanted to be sure that we were right in drawing our conclusions concerning the Marathon mill. I feel that everything which has been said has been corrected in so far as it was possible to correct it. It might be of passing interest to note that every sample taken from the mill, all screen analyses, and all meter readings were made by the writer. It is not for the Marathon mill alone that I stand, but it is for the principle of the grinding element which the gentleman has just mentioned; and I think, and predict with equal empha-

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sis that within another 5 years your ideas on the subject of fine-grinding machinery will have a different viewpoint.

ROBERT FRANKE, Miami, Ariz.—I would like to make a few remarks in regard to the use of equating the work performed by crushing. The tendency of this use has been rather noticeable during the last few years, and in this discussion has digressed to stating the capacities of grinding machines in terms of "energy units." I realize the need of resorting to an equating medium for efficiency comparisons, and favor its practice, but feel that we should be cautious, in view of several unknown and disputed factors, to apply this principle only within its limits of reasonably known accuracy, or under very analogous conditions. Otherwise, erroneous deductions may be the result.

The factors to be regarded are these: Rock is a heterogeneous substance, and in the porphyries generally is composed of feldspars bonded by a siliceous matrix. As crushing proceeds from a coarser to a finer state the softer components are first reduced to colloidal sizes until practically only the harder components remain for final crushing. Hence there exists an increasing unit resistance to crushing for which a determined allowance should be made in equating the production units.

Rittinger's and Kick's laws, upon which the interpretation of work performed in crushing is based, are in dispute, because of a disagreement as to the manner in which crushing forces act. Relatively, these two laws are fairly parallel for the coarser range of sizes, but become very divergent with the finer sizes.

The average size of −200 material is an unknown quantity, and in our present knowledge thereof can only be unreliably approximated. Moreover, it is also to be kept in mind that with this range of sizes there are encountered two classes of material, sands and colloids, each of which undoubtedly have different surface-volume relations.

In view of these factors, it is my opinion that the conversion of the work performed by crushing into energy units, for comparison of efficiency, should only be attempted when the fines of the feeds and products to be compared are practically equal, say within 2 per cent. of each other Further, the practice of stating the capacities of grinding machines in terms of "energy units" should not be adopted until the absolute value of these units has been determined, or misunderstandings will result. In lieu thereof, the practice of stating such capacities in terms of "Tons through a given mesh per horsepower unit" (which mostly is the primary object aimed for), for a given ore and given feed, would be more reliable and be resorted to until research along these lines will broaden the field for more exact application.

F. J. H. MERRILL, Los Angeles, Cal.—I would like to ask if the manufacturers of grinding balls have in any way made a suggestion in providing

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sections of rods or in casting balls with spherical cavities in the hope that the balls will engage the adjacent one. I think the Jeffrey company makes balls of that kind.

THEODORE B. COUNSELMAN, Duluth, Minn. (communication to the Secretary*).—I have read Mr. Blickensderfer's paper with special interest because at the time these tests were being carried on I occupied the position of Efficiency Engineer at No. 6 concentrator of the Arizona Copper Co., Ltd.

A comparison of the work of the 8-ft. by 36-in. Hardinge pebble mills, at the neighboring concentrator of the Arizona Copper Co., Ltd., immediately suggests itself. This plant contained 12 mills exactly the same as the mill used in this test. In the following table, the figures for this plant are averages for the year 1914.

	Detroit Copper Co.	Arizona Copper Co.
Power, horsepower	56.56	56.65
Tonnage	244.00	196.70
Dilution, per cent. solids	42.80	55.00
Pebble consumption per day	527.00	262.00
Pebble consumption per ton	2.160	1.334
Life of pebble lining	159.00	200.00
Tons per lining		40,000.00

The feed to the mills at No. 6 concentrator, was deslimed in drag belts. The tonnage is low because it was not necessary to crowd the mills. Ten would have been, and frequently were, sufficient for the crude ore tonnage at that time being handled. When the mills were first installed, 500 lb. pebbles per day were charged for an entire month, without seeming to create an excess. This amount was gradually cut down to 250 lb. where it remained.

The pebble lining used in all these mills was first introduced by David Cole, then in charge of the remodeling of No. 6 concentrator. It consisted of the largest-size Danish pebbles, set on end in neat cement or very rich concrete, the pebbles being rather carefully fitted together, much in the manner of building a dry-wall. The lower half of the mill was lined the first day. The mill was turned over the second day and the other half lined. It could then be put into operation, if desired, but was ordinarily allowed a day or two longer to take a final set. Such a lining required 10,000 lb. of pebbles, 50 sacks of cement, and cost installed, including cutting out the old lining, about $160. The life would vary from 150 days to as long as 400 days actual running time, with an average of about 200 days. Linings composed of silex blocks would last only about 60 days.

* Received Oct. 7, 1916.

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Lifting bars were tried in these mills. These were formed by rows of silex blocks, set on end, in the cylindrical portion of the mill. Repeated tests could detect no advantage from them.

A curious fact was brought out by several tests to determine the best dilution of the pulp. Mr. Blickensderfer mentions the "accidental oversize" in the discharge of the mill. The same thing was noticed at No. 6 concentrator, and the mill operators would correct it by "sticking a hose in the feed launder." This seemed so contrary to accepted ideas, that carefully conducted tests were made between several different pairs of mills, each pair having feed from the same source, but water being added to one, while the feed to the other was as dry as possible. In repeated tests conducted with the utmost care, it was shown that with greater dilution, there would be less accidental oversize. It was decided at that time that the most satisfactory way to operate these mills would be in closed circuit.

The Marathon mill at the West Yankie Concentrator was of great interest to all of us. The low power consumption was, of course, the direct result of the small diameter of the mill. I very much doubt if the same attractive difference in the power consumption of the Marathon and the Hardinge mills, would hold true with a larger-sized Marathon designed for finer grinding. In fact this is shown by figures from Burro Mountain.

The capacity and general character of the work of this mill seems to be explained by the fact that its action is that of a series of long-faced rolls. The wide face precludes the possibility of accidental oversize getting through, and it would be conceivably possible to pass an enormous tonnage through the machine, if no questions were asked as to the resulting grinding. The Marathon mill, judging from results given in Mr. Blickensderfer's paper, produces very little slime. This was noted all through the tests on the Marathon mill. Had this machine been invented 10 years before flotation was perfected, it would have solved many a difficult milling problem. Now, however, copper metallurgy has so changed that it is advantageous to make as much slime (minus 200 mesh) as can economically be made at each grinding. In 1913–14 the Miami Copper Co. replaced their worn-out Chilean mills with Hardinge pebble mills in order to get a more granular product. Now with flotation installed they are using balls instead of pebbles and grinding in closed circuit, in the endeavor to make more slime.

In comparing the results of the Marathon and Hardinge mills in these tests, the fact should not be overlooked that the Marathon mill was equipped with steel rods, while the Hardinge mill used flint pebbles. It would be ludicrous to suggest that the Marathon be equipped with flint rods, but to suggest that iron balls be tried in the Hardinge would be quite to the point. In one case that I know of the capacity of the

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mill was increased more than 60 per cent., and the fineness of grinding increased by substituting balls for pebbles.

The measurement of grinding efficiencies has received a good deal of attention, and two well-known theories and methods have been advanced, of one of which, and a modification of it, Mr. Blickensderfer makes use in his paper. The other would give a very comparable result. To my mind, however, both methods fall down in that ore is not a homogeneous substance. The ores of the Morenci district, for instance, are relatively easy to granulate to, say, 20 mesh, but very much more difficult to pulverize to 200 mesh. The only real comparison between grinding machines would be to equip otherwise identical sections of a mill with the two machines to be tested, and determine the net profit made by each section. In one case that I know of, balls versus pebbles in Hardinge mills, have been compared in this way, with the advantage very much in favor of the ball-mill section, despite the fact that the relative mechanical efficiency was only slightly better.

I do not think that the Hardinge mill has been really tried out in Mr. Blickensderfer's tests. It should have been operated with the steel lining and the steel balls, and working in closed circuit with a Dorr, or similar classifier. Had both mills been equipped in this way, and operated to give products of equal fineness, I think there would have been little to choose between them. I should like very much to see this comparison made.

The Marathon mill should have a field where slime is anathema, where the mineral can and should be saved at a relatively coarse size. As Mr. Blickensderfer points out, there were several mechanical defects in the first machine, which have to some extent been corrected in the second. These defects can doubtless be eliminated entirely, if the machine proves to have a wide enough application.

The decline in the popularity of Chilean mills is due more to mechanical drawbacks than to quality of work. The wear of tires and dies is not serious, but as Mr. Blickensderfer himself notes, it is the auxiliary parts, the screens, screen frames, mullers, feed spouts, ploughs, etc., that give the trouble, not only in repairs but in operation.

DAVID COLE, El Paso, Texas (communication to the Secretary*).—The paper by Mr. Blickensderfer presented at the session at Miami was very interesting to me because I witnessed the operation of the machines when they were being tested. I can vouch for the great pains that were taken by Mr. Hall and Mr. Blickensderfer in making the tests as nearly parallel as possible as to feed going to the competing machines, the power consumed by them, and the measuring of results as accurately as possible. Mr. Blickensderfer stated in his discussion that

*Received Oct. 4, 1916.

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the tables compiled by him were done with great care and "are about as accurate as they could well be," and I have pleasure in corroborating his statement from the standpoint of observation on the ground at the time. We are indebted to Mr. Blickensderfer not only for his paper, but also for the very courageous way in which he defended the position he had taken, when under the rapid fire of questions that resulted.

The errors of comparison are fundamental and the question raised by the gentleman from Missouri (I don't remember his name) discloses one of the most important ones. Except for the purpose of showing different results when using grinding media of different gravity, it is fundamentally wrong and illogical to compare the work of a grinding mill of the tube-mill family, using flints, with one using steel grinding media. It would be illogical to suggest that the Marathon mill be operated with a set of rods made of agate, and it is equally illogical to compare the work of a Hardinge mill using flints with a Marathon using steel rods. Little importance should therefore attach to the comparison.

The comparison of work done, based upon the scientific theory of Stadler, Gates, Kick, et al., is beautiful on paper, but there are a lot of us who hesitate to accept the theory as "law." We are inclined to regard a direct comparison of grinders arranged side by side, getting feed from a common source through a mechanical distributor, and making a product that affords as nearly as may be the same screen measure, and at any rate affording an equal metallurgical opportunity for the subsequent treatment, as the Supreme Court in these grinding matters. The Marcy versus Hardinge ball-mill controversy is soon to have this kind of a hearing at the Inspiration plant, and the results will be watched with great interest, and if Mr. Blickensderfer can arrange to have the Hardinge mill at Morenci blocked down to about 6 ft. in maximum diameter, have it lined with steel with lifter bars, and filled with a charge of properly assorted steel balls, and then repeat the comparisons mentioned in his paper, our knowledge of fine grinding will be much advanced.

I believe in the use of the rods. I feel that the line of contact affords a better opportunity for the power being expended to be applied to the particles to be broken or crushed. I believe that the large-ball idea will prove the better scheme for breaking down the coarse particles, say to 8-mesh size, but that the rod idea will win out for the finishing process, particularly in closed circuit with an overflow classifier. But I do not think that the difference will be so overwhelming as to put the ball type in the discard, so to speak, for the balls may be modified in shape so as to get the line of contact result. What will prove to be the best length of rod, or the proportion of length to diameter, have not been determined. Perhaps a length equal to a single diameter, but with a dual axis will prove to be good. A "grinder" of this form presents a circular side and

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end elevation and a square appearance in plan. It is balanced around its center of mass similarly to a sphere and affords lines of contact of lengths varying from nil to the largest diameter. It would, of course, tend to wear into a sphere, but it ought to do a lot of grinding before it finally surrenders to that form.

There is one point distinctly in favor of rods, and that is that the steel mills everywhere are equipped to produce them at a minimum expense. I carried on some experiments in a small way at Morenci looking to the development of some "laws" about shapes of mullers, but found nothing better than rods, and since these are easiest to get for the reason above given, the investigation was halted on that ground.

Mining and Milling Practice at Santa Gertrudis

Discussion of the paper of HUGH ROSE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1295 to 1332.

JAY A. CARPENTER, Tonopah, Nev. (communication to the Secretary*).—This description of the Santa Gertrudis mill is of great interest to the operators of similar silver mills in Nevada. At the San Francisco meeting last September there were two excellent papers presented on Nevada silver mills; one by A. H. Jones on The Tonopah Plant of the Belmont Milling Co., and the other by E. E. Carpenter on Cyaniding Practice of the Churchill Milling Co., Wonder, Nev. At the same meeting, I presented a paper on Slime Agitation and Solution Replacement Methods at the West End Mill, Tonopah, Nev., covering this particular feature of our milling practice. A comparison of the data presented in these four papers is interesting in that, while the general design and practice is much the same in all cases, there is a wide difference in the design and practice in the various departments of the mills.

It is to be regretted that Mr. Rose did not round out this excellent article by following Mr. Jones's and Mr. Carpenter's lead in giving the average gold and silver content of the ore milled and the resulting tailing, with the consumption of chemicals and the cost of milling.

The practice of the Santa Gertrudis mill in the use of cyanide and zinc offers an interesting comparison with that of the Nevada mills, and a discussion of this point may bring out the reasons for the differences that exist.

Mr. Rose describes the ore as "clean with only small quantities of base metals." The fact that the zinc dust precipitate is 85 per cent. gold and silver, and that melting with only 3 per cent. of fluxes the bullion is 945 fine, points to the fact that the bases present in the ore are quite inert to the action of the cyanide solutions. In fact, the ore ap-

* Received Sept. 21, 1916.

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pears to be very similar to the Tonopah and Wonder ores; yet the strength of the cyanide solution in the mill and the amount of cyanide used per ounce of bullion is nearly double that of the Nevada plants, while the zinc consumption is about the same.

The figures given in the accompanying tables are from articles in the *Transactions*, except those of the West End mill, which are taken from the annual report and mill data for 1915.

In the accompanying tables the gold and silver content of the Santa Gertrudis ore is given the same as Mr. Rose gives under "a representative analysis" in discussing flotation, and the ounces gold and silver precipitated are assumed, since extraction figures are not given, at a little above 90 per cent. of the ore content, which corresponds closely to Tonopah results. The total time of the ore in the different mills is roughly figured from the total ore tankage in the mills, and the figures are relatively correct.

These two tables give considerable data that should be of special interest to the students of cyanide solutions and the cyaniding of silver ores.

S. J. KIDDER, Mogollon, N. M. (communication to the Secretary*.)—Mr. Rose says that in clarifying their solutions they pass all solutions to be precipitated through sand filter tanks and that the addition of about 40 per cent. by volume of sawdust to the sand considerably improves the clarifying efficiency of the filtering medium. In this connection the writer would like to ask Mr. Rose if such use of sawdust is not liable to cause serious loss unless the sawdust is finally recovered and burned and the ashes saved? In the larger silver plants in Nevada it was the general experience that wood shavings, sawdust, cotton waste, sticks and rubbish in general which contained carbonaceous matter, if in contact with pregnant solutions for any length of time, would invariably cause considerable amounts of silver and gold to be precipitated. For this reason it was the usual practice to burn all such material and save the ashes, which were shipped from time to time to the smelters with either concentrates or slag. It would seem that the addition of 40 per cent. by volume of sawdust to the sand filter through which pregnant silver-gold solutions were to be passed would certainly result in a decided loss by precipitation unless the sawdust was saved, which Mr. Rose does not mention.

HUGH ROSE (communication to the Secretary†).—Carefully taken samples of this product after a year of service assayed about 120 grams silver per ton, this being more or less the grade of solution passing through clarifiers. These same samples washed with distilled water in a small vacuum filter, where there is no possibility of washing out anything but

* Received Sept. 5, 1916.
† Received Oct. 26, 1916.

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Tables Accompanying Discussion by Jay Carpenter.

Cyanide Data

Mill	Contents of Ore		Per Cent. Values Removed by Concentration	Strength of Solution		Consumption Sodium Cyanide		Hours of Ore		Temperature of Solutions
	Oz. Au	Oz. Ag		At Batteries	At Start of Agitator	Per Ton Ore	Per Fine Oz. Bullion Recovered	In Agitators	In Mill	
				Lb. KCN	Lb. KCN		Lb.			
Wonder	0.216	19.48	None		4.5	1.72 lb. NaCN	0.095	60	120	About 90°F.
						3.5 lb. KCN				
Belmont	0.234	22.34	About 12	5.0		or 2.71 lb. NaCN	0.153	48	96	90°F.
Santa Gertrudis	0.067	12.00	None	8.0	11.0	3.15 lb. NaCN	0.280	50	96	Presumably 90°F.
West End	0.245	23.42	None	3.4	3.7	2.60 lb. NaCN	0.120	70	110	115°F.

Zinc Data

Contents of Ore the Same as in Table Above

Mill	Method of Precipitation	Os. Au and Ag Precipitated per Ton Ore	Approximate Tons Solution Precipitated per Ton Ore	Os. Au and Ag Per Ton Solution	Os. Ag in Tails		Lb. per Ton Ore	Zinc Used per Os. Au and Ag in Bullion	Per Cent. Au and Ag Precipitate	Per Cent. Fluxes Added	Fineness of Bullion
					Partial	Complete					
Wonder	Thread	18.15	6.8	2.7	0.12	0.12	1.63	0.089	63.5	About 15	968
Belmont	Dust	17.67	6.0	2.6	0.16	About trace	0.95	0.054	80.6	14	932
Santa Gertrudis	Dust	11.00	4.5	3.0	1.00	About trace	0.69	0.063	85.0	3	945
West End	Thread	21.70	3.6	6.0	0.40	0.07	1.21	0.056	65.0	15	930

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dissolved values, reduced to 6 grams silver per ton. Were any precipitate present, it would not be soluble in water and would appear in the re-assay after washing. While there is a remote possibility that precipitation might take place, the chances are very small compared to the risk we run with the large amount of decayed timber brought into the mill every day in the ore flow, and from which up to the present time we have been unable to notice any bad effect. We have frequently had decayed wood pulp collected from various parts of the plant, principally from the foam on the top of the secondary Dorrs, where there would be every opportunity of considerable precipitation and enrichment, but with so far no sign of precipitation. The contained values run approximately what the solution assays, and a water wash will in each case remove them, proving that they are not there as metallic or precipitated values.

The Antecedent Mineral Discovery Requirement

Discussion of the paper of E. D. GARDNER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 117, September, 1916, pp. 1693 to 1707.

VICTOR G. HILLS, Denver, Colo. (communication to the Secretary*).—If anyone advocates the abolition of the antecedent mineral discovery requirement for the purpose of making it easier to secure title to public land, I hasten to say that I am not of that number. I am in hearty accord with all that Mr. Gardner has to say in regard to easy patenting as a handicap to small camps, and that in general it acts to retard the mining industry. And the following sentences I want to repeat in order to emphasize and endorse them, "None of the writers on this subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some people would appear to believe." However, I believe that the discovery prerequisite should be abolished because it fails to effect the protection for which it was designed, and I look for other provisions that will better serve the purpose.

We propose to abolish the law of apex not because the theory is objectionable but because the question of physical fact gives rise to never ending litigation. Now the design of antecedent mineral discovery is good: the court rulings are unobjectionable; but when it comes to the physical fact this requirement is second only to the apex law as a healthy litigation breeder. The only circumstance that has prevented the prediscovery lawsuits from becoming as famous and as expensive as the apex

* Received Nov. 3, 1916.

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suits is that these cases are fought prior to patenting when usually there are not the valuable orebodies proven to stimulate greater expense. The question of discovery can not be raised after patent. In the pre-patent days disputes and conflicts are settled largely by compromise and the public is cheated right and left. In the course of my observation of claim disputes, and it has been long and extensive, the only question seriously raised, as a rule, is whether the contestant's "discovery shaft" has rock in place or is only in wash, and even the question of wash has been successfully disputed and a good assay allowed to save the day. If the discovery shaft has reached rock in place it is too well known that any crack in the rock, any oxidized seam, or any pegmatite streak in the granite, will "hold" if the question comes to a jury, and the other claimants usually throw up their hands to the stake with the oldest date. If there is no conflicting claimant the location is recognized and patented without so much as a thought of the public interest. Thus the discovery requirement, by being depended upon to safeguard against the frauds of securing springs, water holes, range privileges, power and reservoir sites, rights-of-way, summer residence sites, timber land, natural curiosities, etc., becomes a lever to assist in fixing such frauds. A crevice in the rock can be found almost anywhere, and there is nothing with which to contest a mineral title. When, as occasionally happens, two parties race for discovery on the same piece of ground, it is not ore they look for but any crevice in the rock which will serve as an excuse for a "vein." Quoting from Mr. Gardner's paper (p. 1696), "The discovery requires ment is designed, and operates, as a check on the disposal of land- of the people under the mining laws for purposes foreign to the intent of the law, viz., the speedy and bona fide development of the mineral resources of the public domain." The design is all right, but the operation is a shameful failure. Indeed, in the next to the last sentence of the article we read (p. 1707), "By far the greater number of mining claims that have been desired for purposes other than mining have had no discovery of mineral." It is not the use, but the abuse of the discovery requirement which renders it desirable to discard it. We should remember the adage that a law which can not be enforced is worse than none.

Let no one rise and say that we should have some one detailed from the U. S. Geological Survey to examine every discovery, at least prior to patent application. That would make only a little more interesting sport than heretofore. The geologist would come into court and say that there was not sufficient showing to constitute a mineral discovery, and his report would be true and just; then would come the "prospector" with his assay, scraped out of a knife-blade seam (and taken in a sack which had been previously used in sampling high-grade ore); the next witness would say that there was "a sufficient showing to justify a miner in following the same with a reasonable expectation of finding pay ore"

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—that is standard court language which we all know by heart; the next witness would say that he once saw the famous ..... mine, which has produced millions, when it looked no better than the one in dispute; then would come the jury and give the claim to the "poor prospector," every time!

Now the question will naturally arise, "What substitute have you to offer for the protection of the public domain?" In reply, I would say, that in the first place the situation could scarcely be worse with the discovery antecedent removed. Next, the proposal to require recording a notice of annual assessment with vouchers and strict penalties has been so universally endorsed that we may regard it as certain to be included in any new law. This provision will entirely do away with the blackmailing scheme of resurrecting dead claims when some one else has proven the ground valuable. Also, since the only claims which can remain alive after one year are those on which the locator makes a bona fide expenditure, it will very largely do away with the fraudulent practice of holding ground for timber, springs and other surface values. There is an effective difference between an actual expenditure of $100 and an assessment which can be contracted for from $10 to $25. Further I should favor at least $10 per acre for annual assessment. This change alone will do more to protect the public domain than the pre-discovery requirement has ever done. The fact that more is charged for mineral land than for any other class (and the preliminary expense of patenting is also greater) safeguards agricultural land, summer residence sites, etc. The proposal made by one writer,[1] that, when patenting, $100 per acre be charged when there is no discovery, is worthy of consideration; but I would rather be rid of the discovery question altogether.

The one great thing which would do away with all of our troubles on the discovery question, and also a lot of other mining-law troubles, is the divorce of surface and mineral titles. I frankly acknowledge the radical nature of such a proposal, and presume that a vote among the fraternity would show me in a sad minority, but I exercise the courage of my conviction. The famous 33 questions which have been put forth by the Mining and Metallurgical Society do not bring out this most vital question, perhaps because it was regarded as practically useless. However, several members in their discussion recur to this subject, and evidently this matter forces itself upon the mind of every thorough thinker on the subject.

The use of the surface and the extraction of minerals do not, except to a limited extent, naturally belong together, and any law which persists in keeping the two inseparable must be full of injustice and trouble breeding. Why not meet the main difficulty squarely? Grant the pros-

[1] *Engineering and Mining Journal*, vol. 12, No. 16, ¶ C, p. 721 (Oct. 14, 1916).

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pector the full and exclusive right to the minerals which he claims to be searching for, without discovery, together with all of the surface which he desires to actually use, but allow all other citizens the surface right-of-way for roads, trams, pipe lines, ditches, etc., and have the Government retain control of mineral springs, reservoir sites, and the like. Then all of the abuse found in the use of mining locations for hold-up purposes will disappear, and with it most of the urgency for pre-discovery

Our mining companies, after all their ground is patented, frequently buy and sell surface without mineral rights and mineral titles without the surface. They simply meet a natural physical condition. Why should not the Government do the same? Let the patent applicant pay $5 per acre for the mineral rights and $5 per acre additional for whatever surface he requires. If there be a specially valuable tract of timber, beyond the needs for mining the property, a spring or a town site, let the Government inspector assess the value of the same and grant the patentee the choice of buying or eliminating such portions of the surface. Any such surface exclusions can be monumented at the time of the official survey and the Surveyor General will have a map of both surface and mineral titles with scarcely any additional expense.

Separating surface and mineral titles also disposes of the "Classification" question, which I do not consider a practical thing in any case.

I favor making annual assessment work at least $10 per acre and pre-patent work $50 or $100 per acre. This is no more than Colorado, with its 10-acre claims, has always paid, and such a provision would help as a safeguard in the absence of the antecedent discovery.

If the separation of surface and mineral titles can not be accomplished, why can not the Government at least retain control of certain surface privileges during the pre-patent period, and at patent application make an assessment of special surface values and require payment for the same in addition to the nominal per-acre charge?

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The Diastrophic Theory

Discussion of the paper of MARCEL R. DALY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1137 to 1157.

EUGÈNE COSTE, Calgary, Alberta (communication to the Secretary*).—This new theory to account for the accumulation of commercial deposits of oil and gas, is deliberately and admittedly based on the hypothesis that the origin of these products is organic. Mr. Daly says (page 1142), "Should the organic origin of the petroleum that is found in pools be granted, the following interpretation is offered for the mechanics of its accumulation." But suppose the origin thus accepted by definition be incorrect, then what becomes of the diastrophic theory, Mr. Daly's interpretation? As Mr. Daly remarks himself (same page), "It is obvious that the history of petroleum will have to be entirely different, dependent upon its origin, either from emanations coming from the depths, or from organic decomposition in the strata themselves." Then why should one propound a theory, through which he is endeavoring to trace the history of petroleum, by starting with a definition of its origin? The origin should be the deduction, the forced conclusion of the theory, and not its starting point. Facts alone should be considered first, in any theory worth the name, until sufficient proofs have accumulated to permit of the deduction of a complete explanation for these facts. This would lead one to the origin through a real theory founded on facts, and not based on a preconceived idea perhaps true, but possibly fanciful. Mr. Daly's way can only lead on one road, namely the organic road, and, I repeat, it may be the wrong road.

In the case of the petroleum deposits, as I have endeavored to show in several papers on the subject, to which the reader may refer,[1] the facts clearly lead to the force conclusion that their origin is not one of accumulations of hydrocarbons at first disseminated in the sedimentary strata but the reverse process of subsequent local infiltrations and impregnations of hydrocarbon emanations from the depths. This new theory, therefore, founded on an erroneous conception of origin cannot be a help in the solution of the petroleum problem, not any more than were the anticlinal theory or the hydraulic theory, both of which were also based on the same error, although Mr. Daly is inclined to think that the authors of these two theories did not accept the misconception boldly enough (page 1142).

* Received July 31, 1916.

[1] *Journal of the Canadian Mining Institute*, vol. 3, pp. 68–89 (1900), vol. 6, pp. 73–128 (1903) and vol. 12, pp. 273–302 (1909).

Trans., vol. 35, pp. 288–297 (1905) and vol. 48, pp. 504–517 (1914).

Transactions of the Institution of Mining and Metallurgy, vol. 21, pp. 91–192 (1911–1912).

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A careful reading of Mr. Daly's paper shows plainly the consequences of a wrong start as it leads him to the following fallacies:

1. That oil and gas are (first phase of Mr. Daly's process) primary decomposed products of organic matter formed in the sediments shortly after their deposition at ordinary low temperatures, and held by them for a short period of time, until squeezed up into a porous rock deposited above just at the right time to receive them. This is a very different conception from the view held by other organists (geologists believing in the organic origin of petroleums) who hold, on the contrary, that only a very long time (aeons of ages) will cause the distillations of organic matter in the strata—this immensely long time finally accomplishing the same result as heat in ordinary distillation processes. But if Mr. Daly's view be correct, deposits of that kind should be frequent in nature, and yet no one has ever been able to observe them. As a matter of fact they do not and never did exist, as organic matter decomposes into entirely different products which either escape immediately out of the forming sediments, or finally transform into carbonaceous matter or coal without any other hydrocarbon but marsh gas being produced.

2. That porous rocks shortly after their formation (second phase of Mr. Daly's process) may be supposed to be saturated with an emulsion of oil and gas in water, the oil and gas having been squeezed out of the muds below into these porous rocks, and remaining there in some mysterious way, instead of continuing their migration to the surface. Again I will observe that we do not see anything of the kind in nature—wells near the sea shore often striking pure fresh water (entirely devoid of oil or gas and even of salt water) in the recently formed sands or porous rocks beneath.

3. That the supposed gas and aqueous contents of incompressible strata or layers (page 1145, line 12) which according to Mr. Daly could not previously be compressed out to the surface only a few feet or a few hundred feet away, are now (third phase of Mr. Daly's process) compressed laterally for miles and miles to finally accumulate into far-away oil and gas pools. Apart from the fact that these porous layers cannot be both at the same time compressible and incompressible, we know that porous sand layers form in the sediments disconnected lenticular beds seldom continuous over large areas; this would certainly stop the ingenious long lateral migration of Mr. Daly, and would send the oil and gas out to the surface and so would the faulting and fissuring in the highly disturbed portions of the mountain chain considered through which the oil and gas are supposed to migrate laterally. One certainly cannot admit that the shales covering the sands which, according to Mr. Daly's first phase of the process, could give out so easily their hydrocarbon contents have now become so impervious over distances of many miles, as to make possible this long lateral migration through the sands preventing all the while

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during this long travel the vertical escape to the surface only a few hundred feet away.

Mr. Daly says that "a partial origin of the rock pressure would thus have to be traced to orogenic deformation." This is rather vague and is only based on the remark that the pressure in Ohio and in Indiana, in the Trenton Limestone wells, was much smaller than in Pennsylvania and West Virginia; but we must not forget that the depths at which this gas was found in Ohio and Indiana were also much less than in the wells where the double pressure was recorded in Pennsylvania and West Virginia. As a matter of fact, the gas pressures in all fields are principally function of the depth indicating plainly the source from the volcanic magma below.

Mr. Daly's explanation of how capillary pressure seals an oil pool is ingenious, but may I ask why it only acts to prevent the pressure from dissipating outward into the surrounding sediments and why it did not also prevent the reverse movement of the accumulation of the gas and oil inward from the surrounding sediments into the sands?

F. G. CLAPP, New York, N. Y. (communication to the Secretary.*)—

The points of excellence in Mr. Daly's paper entitled The Diastrophic Theory are so numerous, that it is unfortunate he seems to have based his paper on a misunderstanding of the structural theory, as ordinarily understood. According to Mr. Daly, "the force which is supposed to have caused the motion is the gravity of the hydrocarbons." And again "Gravity or buoyancy is to be considered the sole agency through which accumulation has been brought about and, as such, is supposed to be adequate to explain accumulation under any condition of dip." So far as I know, no petroleum geologist believes these statements; so that Mr. Daly's paper must not be taken as a criticism of existing theories, but of his conception of them.

So far as I know, no petroleum geologist supposes the *motion* of the oil, gas and water in oil fields to have been caused by the force of gravity. To quote from an early issue of *Economic Geology*:[1]

"The 'anticlinal theory' is only one of the factors in the accumulation of oil and gas pools; and a geologist, in order to locate a pool, even approximately, has to consider every other particle of evidence found in structure of the subsurface rocks, changes in intervals, texture of the 'sands,' character of the overlying beds, differences in pressure, relations to water in the rocks, shape of the surrounding pools, and character of the oil in the vicinity; and every factor must be given its due weight, before a recommendation can safely be made. After all other factors have been considered, it might seem as if the 'anticlinal theory' has been lost sight of as unimportant; yet the fact remains that structure is the essential condition which finally determines the distribution and size of the pools when all other conditions are favorable."

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* Received Aug. 28, 1916.

[1] F. G. Clapp's Discussion of paper of M. J. Munn, March, 1909, *Economic Geology*, vol. 4, No. 6, pp. 565–570 (September–October, 1909).

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Passing this technicality, however, we can wisely weigh Mr. Daly's remarks that oil fields exist only on the side of mountain ranges *away* from the source of pressure. Certainly this appears to be applicable in the case of the Appalachian fields, and might well explain several vexing questions; as, for instance, the absence of oil fields east of Pittsburgh, Pa., west of which they are so numerous. In this connection, it is important to raise several questions, as, for instance:

1. Why is natural gas found east of Pittsburgh in abundance, while oil is generally absent?

2. Why is gas found on the *inside* of the Transylvanian Basin in Hungary, while oil seems limited to the outside regions, in Roumania and Galicia?

3. How will this theory explain the pressure of salt water in large quantities, in the regions away from the source of pressure?

4. How about oil associated with faults, as in Oklahoma, California and Wyoming?

5. Can the diastrophic theory be effective in dips of ½° to 5°, in regions several hundred miles from any mountain belt or major axis?

While the theory as propounded may account for one cause of the movement of petroleum in certain directions, the question of whether this is the *main* cause of accumulation must depend somewhat on the answers to the above questions. In my opinion, they can not all be answered favorably to the theory. Of course, however, a mere theory of cause or origin can avail little in petroleum engineering. It is where the oil is found that counts. We know that it exists in definite types of structure, when these are affected by certain other modifying conditions; but that these same structures, with a different set of conditions, will hold no oil. Successful oil-location depends preëminently on *inference—i.e.*, a comparison of conditions in prospective fields with those in fields already known elsewhere in the world.

R. W. PARK, Washington, D. C. (communication to the Secretary*).—Mr. Daly's exposure of the frailties of the commonly accepted theories as to the mode of accumulation of oil is more acceptable than are most criticisms of this type, for after knocking our theories down over our eyes he does not leave us to grope blindly, but leads us to a brand new theory, both plausible and persuasive, that is to take the place of our old friends. Yet Mr. Daly's theory seems to be heir to some of the weaknesses characteristic of our old theories.

Most of the arguments that Mr. Daly raises against the anticlinal and hydraulic theories have been raised at one time or another by one writer or another, and there are probably now few geologists who will seriously contend that the force of gravity alone is adequate to accomplish

* Received Sept. 16, 1916.

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widespread lateral migration of oil and gas, and the local accumulation of those materials in commercial amounts.

As Mr. Daly sees it, the chief failure of the anticlinal and hydraulic theories is that they do not take into consideration the effect upon the oil of the diastrophic forces that have affected the region. But most of the recent discussions of the origin of petroleum have laid stress upon the importance of these very forces, not, however, their importance as transporting agents, but as transforming agents, in changing the organic débris in the sedimentary beds into oil and gas. Mr. Daly tacitly assumes either that oil exists in the loose muds of the sea bottom practically from the moment of their deposition, or that it is formed soon afterward, during the period of compression due to sedimentation. In any case he appears to believe that oil exists in the sedimentary beds before they are deformed. However, until some well-authenticated examples of the formation of oil in unconsolidated sediments is brought forward, it would appear that the assumption that oil is formed in this manner is a less reasonable one to make than that oil is formed in part at least by the forces that caused deformation and perhaps local metamorphism. Of course it is possible that these forces not only have aided in the formation of oil, but also in its movement from one place to another; but it seems very unwise not to consider the first possibility at all.

Mr. Daly's discussion of the effect produced upon a prism of solid materials by pressure applied at the ends is interesting. After describing the purely theoretical example he applies the general principles involved to a succession of sedimentary beds, but in doing so he immediately agrees to consider only the upper part of the prism, that is, the part above the "neutral plane." One is led to wonder as to the depth this "neutral plane" lies beneath the surface, and just what the conditions are below it. Are the anticlinal axes the zones of contraction and the synclinal axes the zones of extension below this plane? If so, would oil, gas, and water tend to accumulate below the neutral plane in synclines rather than in anticlines? In the southern coast ranges of California many of the folds are shallow, and the "neutral plane" would in many cases certainly not be so very many hundred feet below the surface; so the question as to the accumulation of oil in them is not quite so academic as one might imagine.

According to Mr. Daly's theories, anticlines are important, so far as the accumulation of oil is concerned, only as they mark zones where the porosity is greater than in the surrounding region. In many of the California fields the folds are of variable character. Some are sharp and closely compressed; others broad and open. Between the two extremes almost any intermediate type may be found. It would seem natural to assume that along the more sharply flexed anticlines, the beds

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would be more fractured, or at least more extended, as Mr. Daly would say, and that they would thus determine the position of the more porous zones. Yet it is very frequently the case that it is not along these sharper folds that the oil accumulation is greatest at present, but in the upper parts of the larger, broader, and more gentle folds. Moreover, in the California fields the beds are so variable—so tremendously variable—that the differences in porosity due to lithology must be far greater than differences in porosity due to position along the folds; certainly the large broad folds can not have so altered the porosity of the bed that lithology will not still be the controlling feature. Yet in spite of this, the California fields offer a most splendid example of the occurrence of oil along anticlinal folds or in parts of the region where those folds dominate the structure. It would seem that there must be some other reason than increased porosity that causes the oil so to seek these structures.

If the forces that have caused the deformation of the region are the forces that have caused the migration of the oil and built up the pressure under which the oil and gas are now confined, one would expect that this rock pressure would be fairly constant over considerable areas. The field of action of these forces has been extensive, and, if we accept Mr. Daly's theory, these forces have swept the fluids from a wide area, segregated them, and collected them in out of the way corners. It seems hardly reasonable to suppose that if the rock pressure is due to such a general cause the fluid in one little corner would have a certain rock pressure, while that in a neighboring corner would be under another pressure that is radically different. Nor does it seem probable that within one little pool the pressure would vary greatly; yet in the California fields variation in pressure in a given stratum seems to be the rule rather than the exception. This variability may of course be explained in part by variation in lithology and other natural features, and also by artificial features, such as the mode of handling a well. It is not always the later wells that show the lesser pressure, so the variation can not be explained away completely as a decrease for the pool due to release by some wells. But even considering the factors that make for local variation, it seems that the differences found between wells only a few hundred feet apart are abnormally large.

Mr. Daly's discussion of the method by which an oil pool may be sealed is interesting indeed, yet it would seem that the very factors that he calls upon to seal up an oil pool and to maintain a given rock pressure might equally well be called upon to prevent migration originally. If a given force causes the oil to move through the rock pores and collect under pressure at a given place, and if the force ceases to be active, why will not the oil move back over the path it originally traveled?

According to Mr. Daly's theory the forces causing the oil and gas to migrate also caused the water to move. Indeed, he considers that

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the water moved carrying the hydrocarbons with it. Water, then, should be under the same pressure that the oil is under, as the force causing accumulation is the same, and the factors preventing dissipation the same for water as for oil. Yet in the California fields where water sands and oil sands are interstratified, no pressure comparable with the pressure in the oil sands is recorded for the water sands which lie between these oil sands unless a distinct flow of petroleum gas is noted with the water.

Mr. Daly notes the fact that in the Eastern fields there is an increase in rock pressure with the increase in age of the strata in which the oil or gas is found. The same can hardly be said of the California fields, for the greatest pressures noted occur in the younger beds which rest unconformably upon the shales in which the oil originated. The sands that are distinctly stratified with these shales contain oil under a much lower pressure.

In discussing rock pressure Mr. Daly points out that the chief explanations heretofore offered have been (1) that it is due either to hydrostatic pressure, (2) weight of superincumbent strata, (3) gradual accumulation of the inclosed gas, (4) capillary diffusion. He shows that some of these explanations are untenable, for the forces called upon are clearly not competent to have built up the high pressure now encountered. He therefore assumes that much of this pressure is due to the forces that have caused the deformation of the region.

An explanation that in many ways seems to be more reasonable is that the rock pressure is built up within the reservoir, not by the gradual concentration of gas, but by the change of some of the oil into gas. Also, perhaps by the same, perhaps by other reactions, heavy, viscous hydrocarbons are formed that so clog the rock pores that the movement through them of even the gaseous hydrocarbons is prohibited and the reservoir becomes sealed.

Rock pressure so developed would then have no direct relation to the pressure that originally caused the migration of the oil and it might be greater or less than that original pressure. Rock pressure need not be "fossil pressure," as one would be tempted to call it if one accepted Mr. Daly's interpretation.

Just what the reactions would be that would cause the formation of the gases within the reservoir can not be said, but as to those which result in the formation of the heavy viscous material there are a few more data. It is a pretty well known fact that, in the California fields, the highly mineralized water characteristic of the oil fields has a very pronounced effect upon the oil, and that where such water has entered the oil sands the oil in the vicinity is extremely heavy and tarry. This tarry material is in some places so viscous that it can not be pumped. Such material seems to be composed, in part at least, of oxygen and

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sulphur compounds and they seem to have been formed by the interaction of the hydrocarbons and the mineral salts in the water. This tarry material is quite competent to, and probably does, fill the rock pores and prevent the passage through them of the more fluid hydrocarbons.

The Application and Earning Power of Chemistry in the Coal Mining Industry

Discussion of the paper of EDWIN M. CHANCE, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 112, April, 1916, pp. 711 to 714.

EDWARD H. COXE, Knoxville, Tenn. (communication to the Secretary*).—Mr. Chance has omitted to mention one very important use for the chemist in connection with the preparation of coal, and that is in connection with coal washing.

Efficient coal washing, and especially of the finer sizes, can be accomplished only under the supervision of a chemist, as the proper regulation of the washer can only be had by frequent sampling and analysis of the raw and washed product and the refuse, say at least semi-weekly; and only in this way can a check be kept on the quality of the product and the amount of good coal wasted in the refuse.

In large operations, frequently enough coal can be saved from going to the refuse pile to more than pay the salary of the chemist.

The Composition of the Rock Gas of the Cripple Creek Mining District, Colorado

Discussion of the paper of GEORGE A. BURRELL and ALFRED W. GAUGER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 113, May, 1916, pp. 843 to 863.

J. S. HALDANE, M. D., F. R. S., Oxford, England (communication to the Secretary†).—The valuable paper of Messrs. Burrell and Gauger is of special interest to me as, through the courtesy of the management, I had an opportunity of visiting the Portland mine, Cripple Creek, in 1911, in connection with the investigations on acclimatization to high altitudes, carried out on Pike's Peak, near Cripple Creek, by Professors Yandell, Henderson and Schneider, Dr. Douglas and myself.[1] The analyses of one or two samples of the vitiated air were entirely confirmatory of those of the authors. A fatal accident owing to two men being lowered down a shaft filled with the gas had occurred in a neighboring mine just before our visit.

* Received Apr. 10, 1916.
† Received Aug. 25, 1916.
[1] Philosophical Transactions of the Royal Society, Ser. B., vol. 203, pp. 185–318 (1913).

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The composition of the gas is so similar to that of the "black damp" met with in coal mines, metalliferous mines of different kinds, and wells, that there can be little doubt that its origin is similarly due to oxidation processes in the strata. It is very remarkable, however, that the gas is produced in such large amount in the Cripple Creek mines. The rock is apparently very porous or full of fissures, and also contains some substance which oxidizes freely. In some metalliferous mines, and to a less extent in coal mines, this substance is pyrites, the CO_2 being due to subsequent liberation of this gas from carbonates by the sulphuric acid formed; but in many cases the oxidation process is certainly a different one, and further examination would be needed in order to determine what oxidation process occurs in the rock at Cripple Creek. It follows from the authors' analyses that the black damp formed contains sometimes so little CO_2 (*e.g.*, samples 753, 754, 761, 772, 794) that it is lighter than air, as is not infrequently the case with black damp in coal mines, wells, etc.

Further observations on the relation of the issue of the gas to changes in barometric pressure would be of much interest. As I showed in a paper contributed to the *Transactions of the Institution of Mining Engineers* (Great Britain) in 1896, the special danger to well-sinkers from black damp arises from the fact that although they are well aware of the need for testing the air with a candle before descending, they do not realize that the test must be repeated at each descent. A well which, for instance, has been quite clear of gas in the morning may, in consequence of even a slight fall of barometric pressure, be overflowing with black damp in the afternoon. The enormous quantities of gas or air which may issue from a well with a fall of barometric pressure are very surprising.

It is a question of some interest whether it is better to apply ventilation by pressure or by exhaust in cases like that of the Cripple Creek mines. The objection to pressure is that in the event of the fan being temporarily stopped there may be an immediate discharge of black damp owing to the fall in the air pressure. In practice, however, there may be counterbalancing advantages.

As regards the effects of the black damp on the men, it must be borne in mind that the miners will all be acclimatized to the oxygen deficiency due to the altitude (about 10,000 ft.) of the Cripple Creek district. The results of the Pike's Peak expedition showed that there are three factors in this acclimatization:

1. The breathing is increased, so that at Cripple Creek, the lungs are 30 per cent. better ventilated than at sea level.

2. The layer of living cells which separate the blood from the air in the lungs plays an active part in forcing oxygen inward into the blood, where at sea level this layer is only passive, so that the oxygen passes in by simple diffusion.

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3. The blood is about 20 per cent. richer in hæmoglobin and red corpuscles at Cripple Creek. Taking into account the influence of acclimatization, the effects of a given percentage of black damp in the air will probably be nearly the same at Cripple Creek as at sea level.

In a paper contributed this summer to the *Transactions of the* (British) *Institution of Mining Engineers*, I pointed out that the excessive tendency to emphyrema and bronchitis among older miners is probably due largely to breathing air contaminated by too much black damp. I was mainly responsible for the provision in the British Coal Mines Act of 1911 that the percentage of CO_2 should not be allowed to exceed 1¼ per cent. This maximum standard is easily attainable, and seems well worth maintaining. A very simple method of ascertaining how much black damp is present in the air is afforded by the "tube and taper" method which I introduced about 5 years ago.[2] This method was very useful to me in searching for a sample of vitiated air at the Portland mine, where it was by no means easy to find such a sample.

The general health conditions at the Cripple Creek mines appeared to be much better than at most other metalliferous mines known to me. The absence of miners' phthisis was a very marked and extremely interesting feature in this district, and suggested new ideas as to the causation of this very formidable disease.

Diesel Engines Versus Steam Turbines for Mine Power Plants

Discussion of the paper of HERBERT HAAS, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1171 to 1183.

GEORGE W. HAWKINS, Tucson, Ariz. (communication to the Secretary*).—The paper by Mr. Haas will no doubt be followed with considerable interest, as it covers the power-plant problem in quite a comprehensive way, including not only the usual operating costs but also the fixed charges on the investment, and for load factors ranging all the way from 25 to 100 per cent. and fuel oil ranging from $1.25 to $2 per barrel. The very fact, however, that his paper does cover such a wide range of conditions, makes it the more imperative that any conclusion drawn from his analysis be very carefully considered for any specific case, for in making such analysis the type of prime mover and size must be selected for some one condition which might not be the proper selection had some other condition been assumed as the basic one, resulting in an unfavorable comparison under all conditions except that condition assumed.

From the reasons given by the author for his selection of the large turbine units, it is evident he had in mind very high load factors, ranging

[2]Fully described in my book *Methods of Air Analysis*, London, 1912.
*Received Oct. 9, 1916.

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from 75 to 100 per cent., and high fuel cost. Under these particular conditions, then, his comparison would be a fair one, but any comparison for different conditions than these, such as low load factors, or low fuel cost, would be very unfair, for in the latter case the plant should be designed along entirely different lines, resulting in different first cost and operating expense. That this criticism is a just one is evident from his tabulation of power costs, for according to that tabulation the lower the load factor the greater is the difference in operating costs in favor of the Diesel engines. For example: With oil at $1.25 per barrel, at 6,000-kw. load the difference in total operating costs between the steam plant and the Diesel plant is 0.217c. per kilowatt-hour in favor of the Diesel plant, whereas at one-quarter load this difference has increased to 0.342c. per kilowatt-hour in favor of the Diesel plant. This being contrary to the generally accepted opinion and also to his own statement that high load factor and high price of fuel are conditions which favor the Diesel engine, leads one to make a careful analysis of the basis of his figures.

DIESEL ENGINES

Selection of Units

The full-load plant capacity is stated as 6,000 kw. in three units with one unit to serve as a standby. Inasmuch as it would be necessary to operate all of the units to get 8,000 kw., leaving no spare (which is not considered safe practice with Diesel engines), the comparison of the steam plant and Diesel engine plant at 8,000 kw. must be eliminated. With the steam plant as selected it would be possible to operate continuously at 8,000 kw., but not with the Diesel plant. The comparison, therefore, should be limited to an output of 6,000 kw., 4,000 kw. and 2,000 kw.

The author has selected units rated at 2,000 kw. each, these being of the vertical, single-acting, two-cycle type. The writer has carefully followed the development of Diesel engines in this country and does not know of a single installation having this size unit. Two of the largest Diesel engine builders take the position that, in the present state of the art, they would not attempt to put out units over 1,000 kw. Up to date the largest units in the United States are those at Tyrone, installed by Nordberg Mfg. Co., which are rated at 1,250 b.hp., or 800 kw. It is possible, of course, to build larger engines than the above by simply multiplying cylinders, maintaining the same horsepower per cylinder. This, however, does not reduce cost, except for generators, nor does it reduce operating expense. From the author's fourth reason (given on page 1175) for the selection of this size unit, it is evident he figures on a unit having greater horsepower per cylinder. While it is true that in Europe units in excess of 2,000 hp. have been built in the two-cycle type, they are not considered, even by the manufacturers, to be out of the experimental

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stage; nevertheless, they have gained some experience in the manufacture and operation of these large units, which experience would not be of much value to American builders, as the possession of suitable patterns and instructions is not sufficient to insure successful construction of Diesel engines, practical shop knowledge of construction being just as important in turning out successful engines as proper design. This size unit is therefore entirely out of the question for adoption in mining plants or any plants where reliability is a prime necessity. Therefore, engines must be selected not larger than 1,000-kw. capacity. There would then be six 1,000-kw. units for the load, with two spares, giving a total installed capacity of 8,000 kw.

Cost of Plant

Based on the use of large units, the author has assumed a total plant cost of $90 per kilowatt. As smaller units are now being considered, the plant cost will have to be considerably revised.

The writer has seen estimates of costs of various plants ranging from $110 to $140 per kilowatt, depending upon size of unit, location, etc. The author, in a previous paper appearing in the *Engineering and Mining Journal* of Apr. 26, 1913, gives the cost for a plant of 1,400 kw. as $135 per kilowatt, this being based on 450-kw. units. Owing to the fact that the units under consideration here are larger and the plant itself of larger capacity, the writer believes that $110 to $120 per kilowatt would be considered a fair estimate of the cost of this plant, this cost including not only the engines and generators, but the piping, foundations, oil tanks, crane, building, jacket water, cooling system, etc. The installed cost of this plant would then be between $880,000 and $960,000 as against the author's first cost of $720,000.

Fixed Charges

Fixed charge of 12 per cent., consisting of interest at 6 per cent., amortization at 6 per cent., has been assumed by Mr. Haas. The amortization, or sinking-fund figure is based upon 15 years' life of plant, sinking fund bearing 4½ per cent. interest. The life of the Diesel engine is an unknown factor, the present engines having been developed within comparatively recent years. The proper amount to figure for sinking fund is therefore merely an intelligent guess. Some Diesel engine builders figure that there should be an allowance made to cover the entire replacement of the Diesel engines every 9 or 10 years.

As against this, the life of the steam plant is well known. Steam plants are in good operating condition 20 to 25 years after installation, and the steam turbine has been perfected to such a degree that it is not

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likely that there will be any radical departure from present types for a good many years to come. Whatever figure is used for the assumed life of the Diesel engine plant, the life of the steam plant should be figured at least 50 per cent. longer. If 15 years is assumed for the Diesel engine, resulting in a sinking fund of 6 per cent. with money reinvested at 4½ per cent., 25 years should be figured for the steam plant, giving a sinking fund of 2.25 per cent., or 3.75 per cent. greater fixed charges for the Diesel plant than for the steam. Considering that the first cost is about 33⅓ per cent. higher than assumed by the author, the increase in this item of fixed charges is a very material one.

Fuel

Fuel cost is, of course, the largest item of expense at high load factors, and any wrong assumption in this item would affect results very materially. The author has used a figure of 0.64 lb. of oil per kilowatt hour in arriving at his full-load fuel cost, and in fact, his fractional-load fuel cost. This figure is about what builders usually guarantee for four-cycle engines, and it may be obtained on acceptance tests, but there is of necessity always a falling down in actual operation from the test results, due to the method of operation, the way engines are kept up, etc. The fuel consumption, therefore, under normal operation, even when the plant is new, would be at least 5 per cent. higher. Owing to the excessively high pressure used in Diesel engines, any slight wear of cylinder would reduce economy very materially due to the enormous leakage possible, and hence, the average economy over the life of plant is likely to be considerably poorer than for the first few months of operation.

Owing to the very short time the Diesel type of plant has been in operation, it is impossible to tell just what this falling off in economy would be from operation the first few months, but that there will be such a falling off is certain. However, this is not being considered in this analysis. Engines of this large size, however, would probably be of the two-cycle type, which is not as economical as the four-cycle. The author makes the statement in the former paper referred to that the two-cycle engines have fully 10 per cent. higher fuel consumption than the four-cycle engines, making the actual operating economy 0.736 lb. per kilowatt-hour. His fuel consumption, therefore, should be 15 per cent. higher than he has figured.

Further, he has assumed that plant would operate on ordinary fuel oil running 320 lb. per barrel. Diesel engine manufacturers are willing to guarantee operation with heavy-gravity fuel oils, but except in a few instances, even where such guarantees have been made, the owners have after a short trial gone to the lighter-gravity oils, owing to the heavy

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cost and trouble of keeping engine in running condition. To be perfectly safe, therefore, in deciding type of prime mover, one should not base operating costs on the use of heavy-gravity oils, and as the lighter oils run from 15 to 20c. per barrel more than the heavy oils in common use in steam stations, and run considerably lighter per barrel, as low as 285 or 300 lb., there is the possibility of further increase in the cost of fuel oil over that assumed by the author. The additional fuel cost at full load would therefore be at least $20,000 greater than given by the author, and may run as much as $46,000 greater if the lighter-gravity oils should be adopted for the Diesel plant. In this analysis, however, it is assumed that fuel oil can be used.

The above applies to full-load operation. At fractional loads there will be a falling off in economy of the Diesel engine in the same way that there is for steam turbines. Theoretically, it would be possible to get full-load economy at any of the fractional plant loads due to the number of small units, enabling the units in service to be operated at practically rating. This is not practical working condition, however, as Diesel engines have a fixed overload, and on a swinging load it would not be possible to have in service just the proper number of units theoretically required. This would mean that the units in service would be operated at fractional loads, in which case the economy would drop off quite materially, the same as with any type of prime mover.

Maintenance and Lubrication

The next item of importance is maintenance. The author has assumed maintenance as 1 per cent. of the plant cost. This cannot be based on actual operating costs in this country for any lengthened period, as no plants of any size have been in operation long enough to give any reliable records. It would be like assuming the average maintenance cost of an automobile from the records of the first few months of its operation. On the other hand, it may be this is based on operation in European countries. However, considering the difference in price of labor, price of material, class of labor, etc., it is practically impossible to make proper adjustment to arrive at a figure for operation in the United States. The safer plan would be to investigate maintenance costs of actual plants in the United States, increasing these items to cover average maintenance over the period of the life of plant, and then make any adjustment necessary for more favorable conditions in this plant.

In the author's article in the *Engineering and Mining Journal* he assumes a figure which is 3 per cent. of the first cost of the plant for maintenance, making the statement that the figure is based on prolonged cost records of Diesel engine power plants. The writer has in mind two plants of fairly good size where the first 2 years' operation was even higher than

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this, and it is safe to assume that the maintenance cost would increase as the plant gets older.

The cost of lubrication is one that varies widely, depending upon the design of engine, care and operation, etc., varying all the way from one-fourth to one-eighth of the fuel bill, so that the figure given by the author can be considered as a general average, although it might mount up to a considerably higher figure.

Labor

The item of labor, considering the greater number of units, will be higher than assumed by Mr. Haas. In addition to the engineer and his assistants and electrical operators, there should be one oiler for two engines, which would make nine oilers per 24-hr. period. This adds a considerable amount to the labor rate.

Steam Plant

Selection of Units

Mr. Haas states that his reason for selecting 6,000-kw. turbines is the superior economy of the 6,000-kw. turbine over, say, a 3,000-kw. turbine, which justified the very much heavier first cost. This is the proper way to select size of unit, but it can apply only to one particular assumed condition. If the plant is to run at 100 per cent. load factor the superior economy of the 6,000-kw. unit will offset the fixed charges on the greater investment, but if the plant is to operate at one-half, one-third or one-quarter load, the selection of a 6,000-kw. unit may be a very bad one, for the total operating cost might figure out much lower with a smaller unit because of superior economy at the load considered and lower first cost due to smaller installed capacity.

Cost of Plant

Assuming that his selection of 6,000-kw. turbines is the proper one, his plant cost is about right, but if plant were to operate at, say, one-half load or one-third load, an investigation of the total operating cost would probably lead to the selection of two 3,000-kw. units to carry the load with one spare, or a total installed capacity of 9,000 kw. against an installed capacity of 12,000 kw., resulting in the plant costing probably 25 per cent. less than assumed by Mr. Haas.

Fixed Charges

As stated above, if life of the Diesel plant is considered as 15 years, the life of the steam plant ought to be, for fair comparison, assumed at

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25 years, resulting in 3.75 per cent. lower fixed charge for the steam plant than for the Diesel plant.

Fuel

While the assumed steam-plant economy checks with average practice, it is not by a good large percentage as high as is actually being obtained in some recent plants under daily operating conditions with units of approximately the same size as assumed here, this increase in economy being obtained by advance in boiler design—allowing increased steam pressure and higher superheat—the use of steel-encased boilers, the use of automatic oil firing—giving better furnace regulation—the use of properly designed cooling system, and the advance in condenser design allowing higher vacuums, etc. These are not merely possible improvements in steam-plant design, but they have actually been incorporated in some existing plants, resulting in the high economy stated. Even these economies can be exceeded where plant is installed at seaboard where colder condenser water is available, giving higher vacuum, and consequently better economy of prime mover.

Further, it is just as easy to burn a lower-gravity oil as to burn lighter oils, the low-gravity oils running greater weight per barrel and costing the same, or slightly less. On the other hand, it would not be wise to use this low-gravity oil for Diesel engines.

The fuel item, then, could easily be as much as 20 per cent. lower than assumed by the author, which would mean practically $52,000 smaller yearly fuel bill at rated load. When operating at the lower loads, his fuel bill is still further off because, as explained above, for a plant operating under these conditions one would not think of installing a one-unit plant, and hence, although the smaller unit would have a lower economy, it would be operating at full load and therefore at better economy than the large unit at fractional load. For this reason, the fuel items at fractional loads are far too high, resulting in an unfair comparison.

Maintenance, Etc.

The maintenance item of a steam turbine plant is one of the smaller items of cost, as it simply means the expense of cleaning boilers, small miscellaneous apparatus, and an occasional repair, so that the figure Mr. Haas has used is exceedingly ample.

The water bill, of course, is much higher than for Diesel plants, but it is not such an item as to have very much weight in the decision of the type of plant to be installed; that is, as far as the cost of it is concerned. The question of sufficient supply is an entirely different matter.

The labor cost of operating a steam plant is less than for a Diesel,

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as a steam plant requires very little attention compared with a Diesel engine; and with an oil-fired plant and automatic system of oil firing, the boiler room labor is cut down to a minimum.

Diesel and Turbine Plant Diagrams

In addition to the tabulated analysis, Mr. Haas submits diagrams, Fig. 3, showing diagrammatically the relation between the operating costs of steam turbine plants and Diesel engine plants at various efficiencies, fuel costs and load factors. These diagrams are quite misleading in that they do not include all the operating costs. The reason given by the author for omitting these is that the remaining operating costs are either of small moment or practically the same for both types of plants. This is not true, as both maintenance and lubrication of the Diesel plant are excessively high and have no parallel in the steam plant. He states that lubrication alone varies from $2 to $4 per kilowatt-hour. The maintenance item will also run about the same rate, so that if these items were included it would make quite a difference in the comparison.

Further, the quarter-load comparison is unfavorable to the steam plant on account of the efficiency assumed. Checking back from some of the figures, it is found that the economy at quarter load is reduced about 50 per cent. over what it is at full load. This would probably be true if a one-unit plant were used, but as explained above, under such operating conditions a two- or three-unit plant would be selected, and the economy would then not be more than 15 per cent. to 20 per cent. poorer than the plant operating at full load.

Conclusion

The writer has made some figures corrected along the above lines which show that the operating costs of both types of plant are materially different from those given by the author. At 6,000-kw. output, which is considered full rating of the plant, the total operating cost per year for the Diesel engine plant is $345,000 and for the steam plant $305,000, making 0.657c. per kilowatt-hour for the Diesel plant and 0.58c. per kilowatt-hour for the steam plant. One hundred per cent. load factor, however, is only a theoretical condition and is seldom, if ever, obtained in practice, if units are properly selected when installing plant. Three-quarters load, or 4,500-kw. output, represents more nearly an operating condition, but under this condition it is still more favorable to the steam plant, the cost per kilowatt-hour for the Diesel plant being 0.76c. and for the steam plant 0.65c.

The above is based upon using the same grade of fuel oil for both the steam plant and the Diesel. If lighter-gravity oils were used for the

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Diesel, the operating cost of the Diesel plant would be still greater, and this is always a possibility as this point is not yet beyond question.

Summing up the above arguments, the author's paper is shown to give an untrue comparison of the relative operating costs of the two types of prime mover, on account of:

1. Selecting Diesel engines of far larger rating than have ever been put into successful use.

2. First cost of plant too low, resulting from this selection of units.

3. Assuming actual operating economy equivalent to builders' guarantees and not reducing this to cover results obtained in actual operation.

4. Maintenance too low, being merely an assumed figure and not checking with published records.

5. Economy of the steam plant has been assumed far lower than is actually being obtained in the latest turbine plants.

6. Fixed charges on steam plant being too high in comparison with fixed charges assumed for Diesel plant on account of same life being assumed for both steam and Diesel plants.

7. Fractional-load economy on turbine plant is erroneous on account of selecting units on the basis of 100 per cent. load factor and then operating them at fractional loads, instead of, in the case of fractional load, assuming different size units which would result in better economy and lower first cost.

While the figures used by the writer for Diesel engine operating costs are very much higher than the author's, they cannot be considered extreme for they have been compared with published records of actual operating costs, and in each case lower figures have been assumed on account of the units being larger and representing the latest development in Diesel engine design. It must further be borne in mind that in a comparison of this kind one is comparing the well-known reliability and well-known cost of operation of a steam turbine plant with a type of plant which is entirely untried for plants of this large capacity; consequently, the operating costs are entirely a matter of conjecture and in using figures somewhat better than published records of such plants, one is going as far as it is safe to go.

Aside from the question of operating costs there are other considerations which are of vital importance in the selection of prime mover for a mine plant, where any shutdown of mine due to power-plant troubles would be very serious, hence the question of absolute reliability of operation, the ease of getting sufficient supply of skilled labor familiar with the operation of the type of prime mover selected, the ease or difficulty of making repairs or getting repair parts, the suitability of the plant for the use of other fuels beside oil, are of prime importance. All of these considerations will tend toward the adoption of the steam turbine plant

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rather than the Diesel plant, even if the operating costs as estimated did show up favorably to the Diesel engine. All such questions are already answered in the case of a steam turbine plant, where they are entirely problematical in Diesel engines.

Considering all these vital points, the Diesel engine must show a tremendous margin on paper in operating costs over the steam plant before the risk is justified in mine power plants.

In conclusion, the writer would refer to the statement made by H. J. Freyn, the well-known authority on Internal Combustion Engines, appearing in the *Transactions of the American Society of Mechanical Engineers* in 1911, as follows:[1]

"Since the Diesel engine owes its existence primarily, in fact almost exclusively, to its unsurpassed fuel economy, it is not surprising that its advent and development have had a singular economic significance in Germany and France and on the European continent in general, where fuel is very expensive and the standard of manufacture high, whereas in England and more particularly in America, where excellent cheap fuels abound, while skilled labor is scarce and dear, there is manifestly not the same inducement for the introduction of very costly although thermally highly economic machinery."

Also—

"Broadly speaking the outlook for the oil engine of medium size in this country seems good, perhaps not so much in the immediate future as later on after more knowledge of the advantages of the oil engine is disseminated and its good features are demonstrated; but over-enthusiasm should be carefully guarded against, since, as mentioned before, economic conditions in this country are materially different from those abroad, where the oil engine seems to have entered upon a triumphal career."

Manganese Ores of Russia, India, Brazil and Chile

Discussion of the paper of E. C. HARDER, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 113, May, 1916, pp. 761 to 798.

HERBERT K. SCOTT, London, England (communication to the Secretary*).—I am acquainted with the manganese-ore deposits of Russia, India and Brazil, and agree generally with the statements contained in the paper regarding those deposits. The information given by Mr. Harder regarding the Chilean deposits is very welcome.

When examining the Caucasian deposits, I endeavored to determine the quantity of ore originally contained in the deposit, and by deduction, the amount remaining for extraction. These calculations were rendered possible by the great extent of the outcrop and the extensive work that had been carried on in the deposit.

In making the first estimate, the quantity of manganese ore and associated sterile material in the bed was measured in a large number of

[1] Vol. 33, pp. 921 to 923.

* Received Aug. 16, 1916.

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places, and the total quantity of ore originally contained in the deposit was calculated equal to 57,000,000 tons.

With regard to the second figure, it was found that only about 15 per cent. of the mineral contained in the deposit was exported owing to (*a*) the crude pillar and stall method of mining, (*b*) the large quantity of gangue associated with the ore, (*c*) the friable character of the mineral, (*d*) its low market value, and the exigencies of the purchaser.

In recent years, beginning in 1900, and more particularly since 1905, the introduction of longwall working and the stowing in the goaf of the associated sandstone, as well as the construction of numerous washing plants, has resulted in a larger proportion of the orebody finding its way to consumers. Already in 1906, workings were being reopened and pillars and poor ore withdrawn which had been hitherto abandoned, so that in all probability a large part of the ore in the old workings will be eventually recovered, although the measure in which this is done will depend upon the market value of the mineral at the time.

With longwall working 90 per cent. of the ore can be obtained, but for the purpose of calculation 75 per cent. may be estimated as likely to be realized by reason of the amount of crushed ground. Of the ore available, about 50 per cent. will be washed, with a loss of 33 per cent., so that the total quantity of ore likely to be extracted from these deposits will be as follows:

Mineral originally in deposit	57,000,000 tons.	
25 per cent. loss in working	14,250,000 tons.	
	42,750,000 tons.	
Lump ore 50 per cent. of above		21,375,000 tons.
Washable ore 50 per cent. of above	21,375,000 tons.	
Less loss of 33 per cent. in treatment	7,125,000 tons.	14,250,000 tons.
		35,625,000 tons.
Quantity excavated to end of 1914		13,500,000 tons.
Quantity of ore available for extraction		22,125,000 tons.

This figure differs appreciably from that of 110,000,000 tons on page 768 of the paper.

Assuming that the value of the ore permits 1,000,000 tons per annum to be marketed after peace is declared, the deposits should be able to furnish that quantity of ore for over 20 years.

In addition to the minerals, pyrolusite, psilomelane and wad mentioned by the author, an appreciable quantity of a dull reddish mineral was mined, which although originally rejected by reason of not being black, was subsequently found to be of better quality than the general run of ore. A complete analysis of some of the first-quality stuff indicated that it was probably manganite ($Mn_2O_3.H_2O$), and as the complete analysis may be of interest, it is given herewith:

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	Per Cent.
Silica	3.02
Manganese peroxide	48.02
Manganese monoxide	36.00
Ferric oxide	0.64
Alumina	0.91
Baryta	0.84
Lime	1.05
Magnesia	0.22
Phosphoric acid	0.337
Arsenic acid	nil
Sulphuric acid	0.542
Oxide of copper	0.03
Oxide of lead	nil
Oxide of zinc	trace
Combined water	8.40
	100.009
Manganese (metal)	58.25
Iron (metal)	0.45
Phosphorus	0.147
Sulphur	0.217

I visited the deposits of the Nicopol district in 1907, and found that manganese ore had been proved to exist over a large area. No calculation even reasonably near the truth can, however, be made regarding the quantity of mineral likely to be contained in the area underlain by manganese ore, inasmuch as the outcrops of the ore are limited in extent, and the details of the thickness and composition of the bed over the area have not, so far as I know, been obtained on a sufficiently comprehensive scale. I am convinced, however, that the quantity of mineral available is very large.

The totals given by Mr. Harder regarding the area of the deposit and the mineral available, apparently taken from the Bryschlag-Krusch-Vogt treatise, represent, in my opinion, but a fraction of the correct figures, and further do not appear to be in correct relation with each other, for an area of 20 sq. km. underlain by a manganese-ore bed 1 to 1.5 m. in thickness would not give a total quantity of mineral available, of 7,500,000 tons. Dr. N. Sokolow,[1] who studied these deposits in 1901, shows the manganese-ore area as exceeding 200 sq. km. in extent, and in one deposit seen by me, with an area of over 50 sq. km. in which many test pits and bore holes had been made, the quantity of ore available, estimated by a Russian engineer of standing, was many times greater than the total given by Mr. Harder for the whole zone.

Generally, the mineral excavated consists of 20 per cent. lumpy and

[1] Dr. N. Sokolow: Die Manganerzlager des Gouvernements Jekaterinoslaw, *Russia, Memoires du Comité Géologique*, vol. 18, No. 2 (1901).

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80 per cent. small ore. The establishment of washing plants is general and approximately 50 per cent. is lost in treatment.

At one property of which "the run of mine" contained 34 per cent. Mn, the whole of the output was washed, giving the following results:

	Per Cent.	
First grade.....	50.00	Manganese (metal).
	8.00	Silica
	10.00	Moisture
	0.16	Phosphorus
Second grade.....	40.00	Manganese (metal)
	28.00	Silica
	10.00	Moisture
Tailing.....	20.00	Manganese (metal)
	38.00	Silica

Regarding the Indian deposits, I consider Dr. Fermor's comprehensive treatise from which the author quotes as the best statement regarding manganese ores hitherto published.

The quantity of ore likely to be contained in these deposits is great, particularly in those of the Central Provinces which are so large and numerous, with in many cases enormous outcrops. If Dr. Fermor's theory regarding the genesis of the deposits of the Gondite series be correct, and the ore continues in depth as at surface, the quantity of mineral available will be enormous.

The Sandur and Mysore State deposits of India classed as lateritoid by Dr. Fermor have much in common with those of Rodriga Silva and Ouro Preto in Brazil, mentioned by the author on page 788.

This class of deposit has often resulted, both in Brazil and India, in financial loss, and so possesses interest mostly of a negative character. While the outcrops of these deposits are generally striking in appearance and seem to contain large quantities of merchantable ore, exploration generally proves them to be superficial in character, the manganese ore giving place at a shallow depth to iron ore, and subsequently to the original rock.

The precise origin of the manganese and iron in these deposits, as in all those of a lateritic character, is obscure, but it is generally agreed that a large part of the material is derived from the underlying rock by the action of ground waters, although Dr. Fermor, in order to explain the formation of the larger deposits, suggests that some of the manganese and iron has come from surrounding rocks. Bryschlag[2] points out that in oxidation precipitation the iron is thrown down before the manganese, which explains the presence of the manganese ore on the surface with iron beneath.

[2] Bryschlag, Vogt and Krusch: *Deposits of the Useful Minerals and Rocks*, translated by S. J. Truscott, vol. 2, p. 852 (1916).

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Analyses of Brazilian and Indian types of this class of ore, in the dry, are given below:

	India		Brazil	
	Sandur State, Per Cent.	Mysore State, Per Cent.	Botafogo, Per Cent.	Tres Cruses, Per Cent.
Manganese (metal)	40.92	38.24	41.23	37.25
Iron (metal)	14.95	16.85	16.23	19.42
Silica	1.50	10.40	3.11	5.40
Phosphorus	0.002	0.184	0.193	0.102

With regard to the Miguel Burnier deposit, the late A. O. Derby held that the manganese was derived from a limestone and suggested that the associated earthy ores, as he termed them, were residual from limestone. Certainly the limestone which did not show at grass and was some 15 m. from the deposit about 50 m. below surface; at a depth of 150 m. was in contact with the manganese ore bed, which was much thinner, and a similar condition was observed at Rodeio.

The extraordinary development of the Morro da Mina mine is an example, somewhat rare, of a property proving to be much more valuable than suggested by surface indications. This area remained for some years undeveloped after manganese mining was initiated in Brazil, principally because the lateritic covering of the hill consisted in great part of a partly altered spessartite rock, a siliceous manganese ore which was also found in an exploratory crosscut some 50 m. in length, driven into the hill near its base, at some remote period. In 1900, 10 years after the deposit was discovered, crosscuts were made on the hill in several places at a depth of some meters, below the lateritic cap, and the huge lenses, since developed, were discovered. Derby suggested that they were the oxidized portions of a manganese carbonate similar to that which produced the deposit of Piquery, and which differed from the more siliceous and resistant mineral-containing spessartite.

Had the explanatory crosscut at the base of the hill been continued for several meters, it would have cut one of the lenses at present being so profitably worked.

I saw the Bahia deposits some years ago, and at that time the Onha property was the principal producer. Indeed, more mineral has been quarried from it than from the Pedras Pretas property mentioned by Mr. Harder. Up to the end of 1908, I estimated that about 70,000 tons of ore had been got from these deposits and I have not heard that this total has since been appreciably increased. The Onha deposit consisted of a vertical lense, 150 m. in length, and a maximum width of 25 m. It was composed of a mixture of merchantable ore and a partly altered man-

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ganese garnet rock. In depth this latter appeared to be increasing in quantity and the better ore was more difficult to obtain.

The war prevented me from contributing to the discussion on Mr. Harder's paper on The Iron Industry of Brazil, read in October, 1914, but I would ask now to be permitted to express my appreciation of the excellent work of Mr. Harder and other American engineers in the study of these deposits.

I do not, however, agree with the author when he asserts that the massive iron ores were laid down as found today and practically ignores the possibility of the massive ore having been formed by alteration from the general average of the iron formation.

No one visiting the iron-ore district can fail to notice that great changes must have taken place in the character and composition of the rock, and which are indeed still going on, although in an infinitesimal degree compared with former times.

This is recognized to some extent by the author, and his statement that the laminated ore is altered to a greater depth by surface agencies, elimination of silica, hydration, etc., than the hard ore, suggests the possibility of the massive ore having been formed in the same way, and afterward suffering dehydration.

So extensive and numerous, however, are these deposits that it is obvious to anyone seeing them that they contain enormous quantities of ore. Hence the question of their origin has little economic interest for the present generation.

Notwithstanding the evidence of a large number of analyses of samples, almost all of which are probably from outcrops, I am of opinion that these ores will be found to contain, when exported in quantities, less iron owing to hydration, and more phosphorus than is generally supposed, and in this resemble the Chilean ores, of which Mr. Harder says that a large proportion will be found to be of non-Bessemer grade.

With regard to the transport of the ore, I believe it could be carried on the Central Railway, if the line were improved and an endeavor made to handle it.

I do not consider that the Brazilian Government will assist the iron-ore export business (and it will require sympathetic treatment) to the exclusion of the establishment of steel manufacture in the country, even though as yet no coal suitable for coke manufacture has been found in Brazil.

The old rule that ore should be taken to fuel rather than the reverse is not now strictly applied anywhere—for have not iron works been constructed at Duluth to utilize empty Lake ore boats as fuel-carriers, and, further, has not the Gary plant been constructed between the ore and fuel, and following the industrial centers moving west?

Contemporaneously with the initiation of the export of iron ore from

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Brazil, a commencement should be made with steel manufacture in the country. It would be easy to introduce the fuel necessary as return freight in iron-ore boats. (The success of steel works in Japan, Canada, British India and Australia will act as an incentive to Brazilians insisting that some steel should be manufactured in the country when the iron ore exports commence.)

Principles of Natural Gas Leasehold Valuation

Discussion of the paper of SAMUEL S. WYER, presented at the Arizona Meeting September, 1916, and printed in *Bulletin* No. 112, April, 1916, pp. 147 to 160.

F. G. CLAPP, New York, N. Y. (communication to the Secretary*).—I assume that where this valuable paper states, near its end, that "it is not possible to establish a market price," the author means that no particular price which will fit all conditions and fields can be given. I am not in agreement with Mr. Wyer, however, that the actual value of natural gas leaseholds is "largely a matter of opinion." I would even be willing to express myself very strongly as believing that if we, as experts, cannot give anything closer than a mere opinion, our testimony or reports are not of great value to our clients. I believe, moreover, that by detailed geological examinations and with our latest knowledge of geological structure, principles of natural gas occurrence, etc., we can approach just as near a certainty in natural gas valuations as we can in oil valuations, and much more so than in the valuation of many metal prospects, in which only the upper few feet of the vein has been opened. In other words, we are getting nearer and nearer every year to bringing geology as applied to oil and gas development into the class of an exact science.

Mr. Wyer's paper gives an excellent outline for methods of leasehold valuation. Being presumably intended as an outline only, it can be expanded indefinitely by a detailed discussion of the methods of valuation for the four different classes of leaseholds which he mentions: (*A*) "Producing;" (*B*) "Protective;" (*C*) "Reserve;" and (*D*) "Prospective." This classification is probably as good as can be made in practice.

In the producing leaseholds in particular it is easy to establish a market price at which these leaseholds should be bought or sold. The question would arise in any case whether or not the leases are connected with a pipe line or the distance from the pipe line. Assuming that a pipe line already taps the property and that gas is already being sold for a certain figure, this price must be multiplied by the number of thousand cubic feet still "in sight" in the ground, from which is to be deducted (*a*) bonuses, (*b*) rentals, (*c*) royalties, (*d*) cost of drilling, (*e*) taxes and

* Received Sept. 18, 1916.

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insurance, (*f*) overhead charges, etc., taking into account, of course, the number of years during which the property has produced and will probably produce, and paying particular attention to the decline curve for the particular wells and the property as a whole. It is also quite necessary to consider, in producing properties, whether these have been thoroughly drilled, or whether there is room for additional wells. The distance apart of the wells and the question of whether or not they have tapped the deepest known sands is important; also the pressure of the gas, its rate of decline, and whether or not it must be pumped. All these and many more questions must be considered in valuing a lease; and when these facts are known, it is not a difficult matter for a natural gas geological engineer to place a definite figure on the property.

The real "protective leases" might be grouped with the "reserve leases," since if they contain natural gas they constitute part of the reserve supply; but if they do not contain gas, they are not necessary as a protection, and are a liability rather than an asset. Consequently, for practical purposes of valuation, they would probably be considered on nearly equal terms with the reserve leases. The value of these two groups of leases is to be figured on the basis of the number of successful wells which can be assumed for them; determined, *first*, from the positions of the reserve leases relative to the producing property, and *secondly*, on the basis of the continuity of the geological structure which has caused the accumulation of the gas.

Another factor which must be taken into account in valuing the protective and reserve leases is the degree of competition in drilling, which, if keen, will cause a serious decline of the pressure and volume of the gas. This is particularly true in such fields as the Hogshooter field in Oklahoma, the Mexia field in Texas, and certain other Oklahoma fields where a number of different gas companies were in the field, all of which picked up leases, many of them of small size, drilling them rapidly and sometimes inefficiently, exhausting the gas as fast as possible, causing a decline in pressure, with resulting influx of salt water into the sand along the lower level of the gas, and finally flooding out the entire field. These fields could have been operated for years, and their value would have been great, if possible to do so without competition; but conditions were such that an engineer on the spot could say definitely that none of the properties were of any great value, but that, on the other hand, they constituted a losing proposition. Such a field, however, if considered as a whole, would have had great value, since all natural gas engineers and geologists will unite in the conviction that monopoly is the proper thing in the natural gas business, and only by monopoly can large properties or fields be operated with financial success and economically for the community.

We therefore see, in *producing*, *protective* and *reserve* leases, that it is possible for an engineer to calculate very definitely the proper value

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which should be placed upon them in order to render their purchase or sale profitable. In the case of the leases which Mr. Wyer classes as *prospective* leaseholds, the difficulty is a little greater, perhaps, since these properties are frequently situated many miles from known gas production; and to a person unfamiliar with natural gas geology, there might be no basis on which to judge their prospective success and consequent value. This is where the geological engineer comes in, and no doubt exists that our science has now reached the standpoint where we can predict the chances in such territory with a reasonable degree of success. We cannot, of course, say in any particular field, as yet untested, whether it will *certainly* contain gas; because sands may be absent, even though the correct geological structure be present; but we can say, having a group of fields or leases, about what proportion of these will be successful and what value should be placed upon them; this being fully as definite and safe a proposition as to place a value on a particular security of an industrial corporation before the latter has started its machinery. The values in such cases are based on the actual net profits "in sight," with the addition of a certain percentage for prospective future business. This additional percentage is an instance parallel to the prospective outlying territory as yet undeveloped by a natural gas company.

We see, therefore, that natural gas valuation, if not already an exact science, is approaching this limit, and that there is no difficulty, at least in the case of a particular property or company, in fixing a valuation, if the properties be studied in the necessary detail. The sellers or purchasers may not always agree on this valuation, any more than they will in a gold mine, war stock or any other commercial proposition; but, nevertheless, it is possible for each side to fix the figure which the property is worth for the purpose desired.

One of the most important questions in valuing a natural gas company, and one to which very little attention has been paid, is that of salt water in the sands. This is sometimes a serious matter, especially in fields where drilling has been close and where the sands are rapidly depleted owing to the heavy draught of gas. A field where the sand is saturated with water below the natural gas level is quite a different proposition commercially, and one not nearly so attractive, as is a field where the sands are dry. In this respect, the natural gas business differs from the oil business.

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Tungsten-Molybdenum Equilibrium Diagram and System of Crystallization

Discussion of the paper of ZAY JEFFRIES, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1225 to 1236.

A. G. WORTHING, Nela Park, Cleveland, Ohio (communication to the Secretary*).—The paper by Mr. Jeffries is very interesting. He is to be commended for his pioneer work in the study of equilibrium systems at the very high temperatures which one obtains with tungsten and molybdenum.

There is one statement, however, to which I must take exception. On page 1227, a statement is made which would seem to indicate that the Nela Research Laboratory joined Dr. Langmuir in the recommendation of 3,300°C. as the most probable melting point of tungsten. This is not the case. In the paper referred to by Mr. Jeffries, I have given 3,630°K. or 3,357°C. as the melting-point temperature for tungsten. Confirming observations by other workers in our laboratory justify the belief that this latter value is correct to within a small part of the discrepancy between the two values quoted.

Mr. Jeffries recognizes the possible introduction of errors in his calibration diagram in which he has plotted temperature as a function of the per cent. of tungsten fusion wattage. A different method of procedure which has merit would be the actual determining of points for the curve between the melting points of molybdenum and tungsten with the aid of a tungsten wire operated at intervening temperatures. It is perfectly possible by means of optical pyrometry, when once a temperature calibration has been established, to determine within a few degrees at what temperature such a wire is being operated. There are two or three such calibrations to select from. Of these the writer has considerable confidence in his own which is reported in abstract in the paper already referred to. Whichever one Mr. Jeffries might select, the procedure would seem to be better than that actually used. It would be interesting to note in this connection whether or not, with the use of such a calibration, the melting-point determination for molybdenum would consistently fall on the curve obtained with the tungsten wire.

In spite of our criticism and suggestion, we wish to express our high appreciation of this work.

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* Received Sept. 14, 1916.

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An Explanation of the Flotation Process

Discussion of the paper of A. F. TAGGART and F. E. BEACH, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 116, August, 1916, pp. 1373 to 1386.

OLIVER C. RALSTON, Salt Lake City, Utah (communication to the Secretary*).—This paper has appealed to me as being one of the most lucid, well-connected and complete papers on this subject which has been published. To be sure, the concepts called upon in an explanation of the flotation process by these authors are mostly old ones which have been more or less discussed in previous literature, but they have nevertheless been assembled in a most constructive and suggestive manner. We find a new contribution to flotation theory in the effect of fine particles increasing the viscosity of the interfacial film between a liquid and a gas. This phenomenon has been variously hinted at but never definitely stated in accepted scientific terms.

The discussion of "contact angles" is very clear and will doubtless be welcomed by those who have not had the opportunity to investigate the literature on this subject, but it is unfortunate that the "hysteresis of the contact angle," observed by Sulman and reported some years ago, is not mentioned or explained.

A few other points need discussion in order to call attention to the fact that, while this paper is the best discussion of the subject which has yet appeared, all of the truth has not yet been told.

On page 1385, in discussing the reasons why heating a pulp often allows the recovery of a higher-grade froth concentrate, four different factors are mentioned. One of these factors deemed probable by the authors is that there is an "increased number of air bubbles formed by the air released from solution." While it is commonly known that heating a solution expels the dissolved air or other gases, to one who has seen the great volumes of air that enter the pulp in a flotation machine it is hard to believe that the pitifully small number of additional air bubbles released by heating the solution can have such a good effect. I would call attention to the following more probable explanation of this fact: the viscosity of water at 80°C. is less than one-fifth of the viscosity at 0°C. In other words, on heating an ore pulp the fluidity of the water in the pulp is very greatly increased and much less water containing entrained gangue will be carried along with the froth during flotation. In other words, a cleaner concentrate is obtained. The surface tension and other physical properties of water are not altered in anywhere near the same degree by change of temperature, and I would

* Received September, 1916.

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suggest that the viscoscity effect is probably responsible for the major portion of the improvement.

I notice that in their discussion of the Potter-Delprat process the authors claim to have observed microscopically the condition of the solid particles in the froth, and find them completely "within the films and at no point in contact with gas." The manager of one of the large Australian companies using flotation told me that two physical chemists in his employ consumed about 6 months' time in an attempt to definitely find out whether the mineral particles were completely submerged in the film or whether they were at some point in contact with gas. In view of their failure to reach a decision, I am inclined to ask the present authors if they are certain of this observation.

Further, we notice that electrical forces are not called into question in this discussion of flotation, although the authors acknowledge that the potential surface energy in the contact surfaces between dissimilar substances might well include electrical forces which might be called on in explanation of certain aspects of flotation. However, they feel that "*Once in contact* the preferential adhesion . . . to a sulphide surface in the presence of water is sufficient to account for the persistent attachment of the sulphides to the bubble films." I feel that they are justified in this opinion, but having seen particles migrate under the microscope when in an electrical field, and knowing that the various kinds of particles and bubbles carry electrical charges, may I not suggest that the electrical charges assist the various finely divided phases to *get in contact?* The authors consider that mechanical agitation, etc., is sufficient to get the various phases into contact, but the acceptance of these electrical phenomena in no way damages the theory which they have presented and merely adds another contributing factor to the theory.

Finally, the authors state that they have not appealed to colloidal phenomena in this discussion. This statement, in the face of the fact that the largest single section of the paper is headed "adsorption," and that most of the forces called upon are those of "*capillarity*," is hard to understand. While I agree with the authors almost entirely, in so far as they go, I find that their paper is filled with something very closely akin to colloid chemistry. The study of capillary forces comes under the head of physics and the study of the effect of chemicals on these capillary forces comes under chemical physics or physical chemistry. Now the physical chemistry dealing with capillary phenomena is known as colloid chemistry, and Freundlich even went so far as to entitle his text on colloids, *Kapillar Chemie.* So it can be seen that while I am delighted with the work these authors have done in formulating their theory, I am disappointed in the way they have attempted to name it. I would say that it can be accepted as part of a *colloidal theory* of flotation.

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The Flotation of Minerals

Discussion of the paper of ROBERT J. ANDERSON, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1119 to 1136.

OLIVER C. RALSTON, Salt Lake City, Utah (communication to the Secretary*).—The literature on the theory of flotation has been enriched, of late, by the views of a number of excellent mining engineers who unfortunately were tyros in physical chemistry and physics. Hence the obscurity and mystery with which the process is supposed to be surrounded.

The present paper goes far toward explaining some of the ideas which have seemed obscure or which have been poorly expounded by the men who advanced them, but there are a number of places where the author's explanations are unsatisfactory, or where he has fallen into the same mistakes made by the original propounders of the theories involved.

While recognizing the value of constructive compilation, it is nevertheless disappointing to find a paper reviewing the recent theories of flotation which presents nothing new—not even a new viewpoint.

At the bottom of page 1121, there are stated to be present the following "phases" in flotation: "solid-liquid (ore-water), solid-liquid (ore-oil), solid-gas (ore-air)," etc. The writer could not possibly have meant to apply the word "phases" to the ideas involved. Each of the *pairs of phases* mentioned has an interface in which exists the interfacial tension that he is talking about.

On page 1127, in discussing electrified bubbles, it is stated that "the bubbles in flotation are simply air spaces contained by a mantle of oil or of water and there is, therefore, nothing within to bear the charge. In case it would carry a charge, which would only be possible by the presence of contained ionized gases or water vapor, the charge would be speedily dissipated by contact with the interfacial boundary. Then in order for a bubble to carry a charge it must be protected by a dielectric film." It can be seen from this series of arguments that the author has fallen into the same error that has confronted a number of exponents of an "electrostatic theory" of flotation. The theory of Bains,[1] which is objected to by Fahrenwald,[2] holds that the friction of the liquid and solid and gaseous masses on each other generates static electric charges and that the function of the oils used in flotation is to insulate these charged particles. Fahrenwald's objections to this theory are sound. However, all these writers have failed to consider the mechanism of attachment of electric charges to

* Received September, 1916.

[1] T. M. Bains, Jr.: Electrical Theory of Flotation, *Mining and Scientific Press*, vol. 111, pp. 824 and 883 (Nov.-Dec., 1915).

[2] F. A. Fahrenwald: Electro-statics of Flotation, *Op. cit.*, vol. 112, p. 375 (March, 1916).

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colloidal particles. They apparently have not understood how suspended particles, including air bubbles, can carry electric charges when suspended in a conductive solution. I have already called attention to the fact[3] that the *film* of liquid in contact with some other phase (gas, liquid or solid) has different properties from those of *bulk* water (or liquid). Mr. Anderson has been kind enough to quote some of the properties of this film, such as its thickness, density and tension, but has failed to remember that I also gave the difference in potential between its outer face and the bulk liquid, even when that liquid is highly conductive. Powis[4] has very recently expanded on the conception of charged colloidal particles and the theories of colloid chemists as to how such particles can hold electrical charges. Briefly, the charges are due to absorbed ions in the interfacial film. Any electrolyte in the solution is more or less ionized, and in case one of the ions is absorbed in the film more than the other, an apparent electrostatic charge is the result. In fact, the electric charge could not exist if the solution in question were not ionized, and hence conductive. It is a strange fact that the conductivity of the solution places serious obstacles in the way of the theory of Bains while conductivity (or ionization) is necessary to explain electric charges on the particles as viewed from the colloid-chemical or "capillary-chemical" standpoint.

On account of such serious misunderstandings as those above mentioned, it would seem that there is still room for the writing of a rigid critique on the theory of flotation.

As Mr. Anderson says, "in the line of development of flotation theory there has been steady progession of thought and a remarkable increase of knowledge in the past few years." That is just the trouble—the progression has been mainly in thought and too little in laboratory investigation. Too many have been willing to contribute *thoughts* to the technical press and too few have busied themselves with experimental measurements to prove the theories proposed.

[3] O. C. Ralston: Why Do Minerals Float? *Op. cit.*, vol. 111, p. 623 (October, 1915).

[4] F. Powis: Transference of Electricity by Colloidal Particles, *Transactions of the Faraday Society*, vol. 11, p. 160 (April, 1916).

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Erosion of Guns—The Hardening of the Surface

BY HENRY FAY,* PH. D., D. SC., CAMBRIDGE, MASS.

(New York Meeting, February, 1917)

THE erosion of guns is a complex problem which can be solved only by a detailed study of all the factors involved. In the present paper it is proposed to submit the results of observations and experiments which have extended over several years. Certain facts have been established and preliminary reports have been published.[1] Conclusions have been reached since then which, it is believed, are worth presenting for the purpose of promoting discussion, and with the hope that ultimately the whole truth will be known. The particular phase of the problem which will be presented here is the hardening of the inner surface of the gun tube.

It has been known for a long time that after firing a large caliber gun for some time, the surface of the metal becomes hard and brittle, cracks, and wears away.[2] The character of this phenomenon may be learned by an examination of some results which are typical, obtained from a detailed study of a 12-in. gun. The gun was trepanned and rings representing various sections of the gun were cut out for examination. The first ring represented the metal at the muzzle end, and the other rings were removed at approximately 120, 240, 325 and 337 in., respectively, toward the breech end. The appearance of the surface of the metal in the different sections is shown in Fig. 1. The greatest amount of wear and maximum amount of hard surface layer was found in section marked *E*, which is nearest the powder chamber and at the beginning of the rifling; the maximum amount of cracking in *D*; the maximum amount of copper deposition in *C*. Section *B* showed heat cracks on both lands and grooves, and section *A* showed heat cracks *on the driving edge of the lands only*. There was much flow of metal in both sections *E* and *D*, and there was progressively less effect noticeable toward the muzzle end. The polished cross-section of these same pieces is shown in Fig. 2. The

* Professor of Analytical Chemistry, Massachusetts Institute of Technology.

[1] Tests of Metals, Watertown Arsenal, 1913 and 1914.

[2] This action is not confined to large caliber guns exclusively but will occur also in smaller guns, although, as will be shown, the action is much accelerated with the increase in the weight of the projectile.

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extent of the erosion is greatest nearest the powder chamber, and diminishes toward the muzzle end.

The depth of heat crack varies irregularly in each section, and somewhat regularly in sections proceeding from *A* to *E*, increasing in depth toward the powder chamber. The number of cracks does not, however, vary in the same way. In section *E*, where the greatest erosion has taken place, the smallest number of cracks appear.

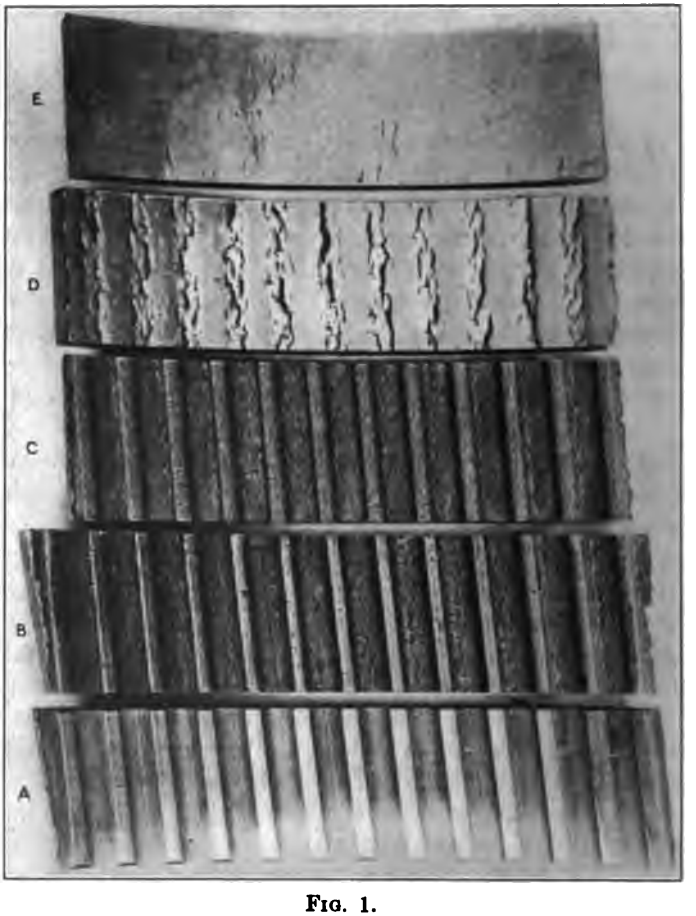

FIG. 1.

The greatest depth of crack is found in section *D*, with an average depth of 0.038 in. The width of crack in this section varies between 0.001 and 0.015 in. In section *E* the depth varies between 0.024 and 0.039 in. In section *C* the average depth of crack is only a trifle less than in *D*, but the width is not nearly so great. The cracks diminish progressively in depth and width in sections *B* and *A*, the latter section showing practically no cracks on the face normal to the axis of the gun. On a

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face parallel to the axis of the gun and cut along the driving edge, section *A* shows some clean, sharp cracks about 0.01 in. in depth.

Sections *B*, *C*, *D* and *E* show a hard layer of metal on the face cut normal to the axis. All five sections show the hard layer on the face cut parallel to and along the driving edge. The depth of the layer of hard metal on the face normal to the axis varies as follows: *A*, 0.0000 in.; *B*, 0.0004 in.; *C*, 0.0008 in.; *D*, 0.0013 in.; *E*, 0.0015 in.

The cracks are undoubtedly due to the unequal expansion and contraction between the hard and soft layers. When subjected to strain

FIG. 2.

the hard, brittle surface develops cracks, and the cracks penetrate into the soft metal. The section showing the greatest thickness of hard layer ought to show the greatest depth of crack, but other influences counteract this effect.

The microscopic appearance of a surface showing cracks is shown in Fig. 3, and in Fig. 4, the latter a characteristic structure always found at the junction of a land and groove. The interpretation of these photographs will be considered in detail after discussing the cause of the formation of the hard layer. In addition to the hardening which takes place in the gun tube proper, hardening and cracking of the surface takes

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FIG. 3.—× 50

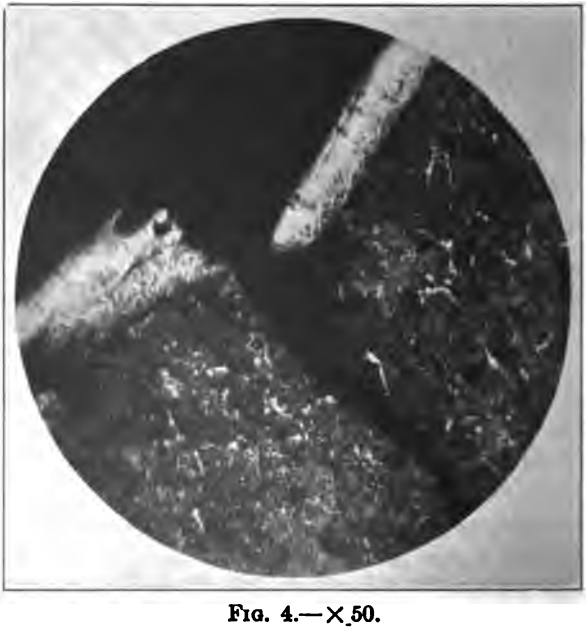

FIG. 4.—× 50.

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place on the pressure plugs which are inserted in guns for the purpose of measuring maximum powder pressures. The pressure plug shows hardness around the edge of the central hole, around the edge of the plug itself, and shows the effect of hardening in the form of cracks all over the surface, but particularly in the vicinity of letters which have been stamped on the surface. Any theory which may be offered to explain the hardening of the gun tube proper must also explain the hardening and cracking of the pressure plug.

While these results are in general typical of what takes place, there are variations in individual cases. For instance, the metal on the driving edge may not have been affected as it was in the 12-in. gun referred to, but it is merely a matter of time until the conditions as described are obtained. The action is a progressive one and is dependent upon the number of times the gun is fired.

The Cause of Hardening

Hardening may be produced by three well-defined methods: (1) cementation; (2) heat treatment; (3) mechanical deformation or cold work.

Cementation.—The principal products of combustion of the powder are carbon dioxide, carbon monoxide, water, and nitrogen. The equilibrium phenomena are very complex and will vary with the composition of the powder, the temperature and pressure. It is conceivable, however, that large amounts of carbon monoxide may be formed, and under the conditions of high temperature and pressure cementation could take place. Giolitti[3] has shown carbon monoxide to be an ideal carburizing compound, and on repeated firings small increases of carbon to the surface might take place. This would produce a hard, brittle layer on the surface richer in carbon than the original metal.

Heat Treatment.—When a piece of steel is heated above its critical temperature, Ac_1, and then suddenly quenched, it is hardened, the depth of hardening being dependent upon the temperature of heating and the rate of cooling. The temperature of combustion of the powder is sufficiently high to heat a skin of the metal above the Ac_1 point, and hardness would result if the surface were cooled with sufficient rapidity. The relatively low temperature of the large mass of steel would produce a rapid loss of heat.

Mechanical Deformation or Cold Work.—It is a well-known fact that when any ductile metal is subjected to mechanical deformation it becomes hard and brittle, and if the work is carried too far, cracking will result. Even the most ductile metals, such as gold and platinum, lose their plasticity and become hard and brittle when subjected to drawing, hammering, etc. In wire drawing it becomes necessary to anneal the metal

[3] *Journal of the Iron and Steel Institute*, vol. 84, p. 307 (No. 2, 1911).

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frequently in order to restore plasticity and prevent it from cracking. This phenomenon is common to both ductile and brittle metals.

The mechanism of this action has been studied in detail by Beilby[4] and his conclusions have been confirmed and accepted by many other investigators. According to him, a surface skin may be built up by mechanical movement which gives unmistakable evidence that the surface must have passed through a state in which it possessed the perfect mobility of a liquid. This surface possesses distinctive properties which differentiate it from the surface beneath it. Hardening thus results from the formation at all the internal surfaces of slip or shear of mobile layers similar to those produced on the surface by mechanical movement. These layers retain their mobility for a brief period only, and then solidify in a vitreous amorphous state cementing together all of the surfaces of slip or shear throughout the mass.

Such a surface produced by polishing, burnishing, drawing, hammering, or cold work of any kind, possesses the property of hardness and brittleness. Metal in this condition may be restored to its natural crystalline, ductile state by annealing.

Having considered the manner in which hardness may be produced, it becomes necessary to examine the different methods in detail, and to see by which method the facts can best be explained. It would seem at first sight to be rather difficult to prove whether or not cementation had taken place, as the maximum thickness of layer which had been hardened amounted to only 0.0015 in. It was obviously impossible to cut off this layer for analysis, not only on account of the thinness, but also on account of the hardness.

Microscopically the appearance of this surface suggested cementite, as it remained bright after etching, as shown in Figs. 3 and 4, and, furthermore, in no case was there any evidence, even under high power, of martensite, which would also remain brighter than the original structure. Etching with boiling sodium picrate, which darkens cementite, gave negative results. The hardness of the layer and its appearance would indicate free cementite, but this test would, therefore, appear to be conclusive that the surface cannot contain any free or excess cementite. The test was carried out always with a control piece which contained about 1.25 per cent. of carbon, and therefore known to contain free cementite. Under all conditions the control gave positive results. There is some doubt as to whether or not a small amount of cementation takes place, and experiments are in progress which, it is hoped, will prove this point definitely.

While it seems fairly clear that the hard surface layer is not free cementite, judging from the sodium picrate test as evidence, it would have been difficult on this basis to explain the fact that the hard layer

[4] *Journal of the Institute of Metals*, vol. 6, p. 5 (No. 2, 1911).

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occurs only on the driving edges in the muzzle end of the gun. Selective formation would not seem probable.

That hardness might be produced by raising the temperature above Ac_1 and suddenly cooling seems entirely possible, and most of the facts can be explained on this basis. The temperature within the gun at the time of the explosion is far above Ac_1, in fact may be above the melting point of the metal. A thin skin of metal is heated for a brief period during the explosion, and before the heat can penetrate any distance, it is cooled by the mass of metal back of this layer. Such a hardening ought to produce the microscopic constituent martensite, but this structure in its characteristic needle-like form is never found, although troostite

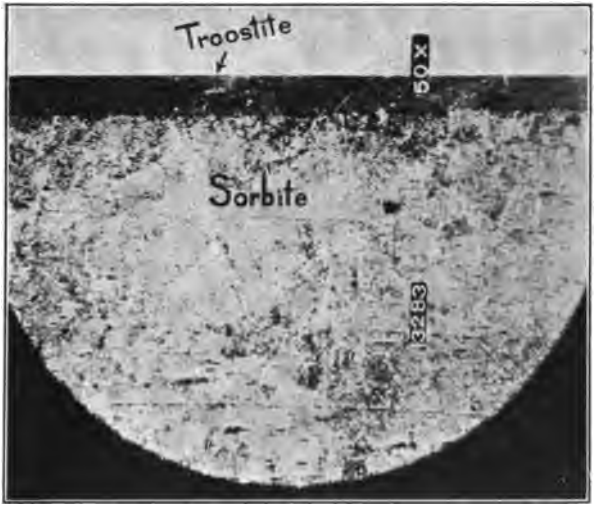

FIG. 5.—× 50.

is frequently found. Rosenhain[5] and Belaiew[5] have reported martensite in the hardened surface of the rifling of a gun, but so far in the examination of many gun fragments, and in trepanned guns, the characteristic martensitic structure has been missing. In some experimental steels which had been subjected to erosion tests by allowing the gas of an ignited charge of powder to escape through a tube of the metal, there was found not only well-characterized martensite but also troostite adjacent to it. The essential difference in this case is that there was no projectile to produce mechanical deformation on the surface.

That the hard surface is actually composed of martensitic material is proved by the fact that on tempering it passes over to characteristic troostite. Samples were cut out of the ring from section *E*, and heated

[5] Rosenhain, Belaiew: International Association for Testing Materials, New York, 1912, *Proceedings*, vol. 2, section A, p. 127.

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for 15 min. to 300°, 400°, 500°, 600° and 700°. The specimen heated to 300° showed troostite development most markedly and unmistakably, as shown in Fig. 5. The 400° and 500° samples also showed troostite, but less well defined, the transformation having progressed further, and there was almost complete reversion in the specimens heated to 600° and 700°. This experiment seemed to prove conclusively that the hard surface was martensite, but threw no light upon the fact that in the muzzle end of the gun the surface was hardened *on the driving edge only*.

It was thought for a time that mechanical deformation or cold work produced by the rotating projectile would account for all of the phenom-

FIG. 6.—× 75.

ena, but this idea was negatived by (1) the martensite→troostite conversion; (2) by the fact that the powder chamber itself, untouched by the rotating projectile, showed hardness; (3) and by the fact that the pressure plug showed hardness.

That mechanical deformation has much to do with the production of hardness is shown by the selective hardening on the driving edge of the land on the muzzle section, and by some experiments which were conducted on pressure plugs.

Knowing that pressure plugs harden in the same way as the surface of

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the gun tube itself, it was decided to experiment with pressure plugs, and to follow the development of the hard surface layer in its various phases. For this purpose, a plug having approximately the same composition as gun steel was prepared for service. The metal was carefully analyzed, it was heat treated, and photographs were made of the fine sorbitic structure. The surface as a whole was uniform. The prescribed lettering was stamped on the surface. It was then placed in service, and after having been fired until it showed signs of heat cracking, was returned for microscopical examination. The face was polished and etched, and the

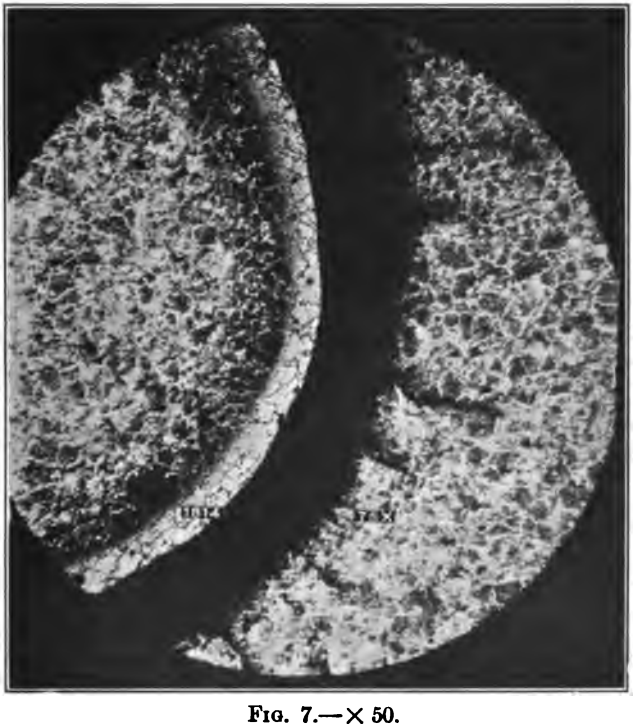

FIG. 7.—× 50.

structure seemed to be normal, but it was seen on detailed examination that the hard surface had begun to appear on the edge of the center hole and on some of the letters. Examination showed that this development had taken place at those points where the *maximum amount of cold work had been applied in stamping the letters*. This is shown in Figs. 6, 7 and 8, representing the letters A, O, and a period respectively. In the letter A and in the period the die was apparently not held in a truly vertical position and one side of the letter was cold-worked more than the other. Selective conversion of soft to hard metal has taken place at these points. This is shown in the lighter area of Fig. 6, and also of Figs. 7 and 8,

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the light area in each case being the hard area. Observations of a similar character were made on almost every letter on the face of the pressure

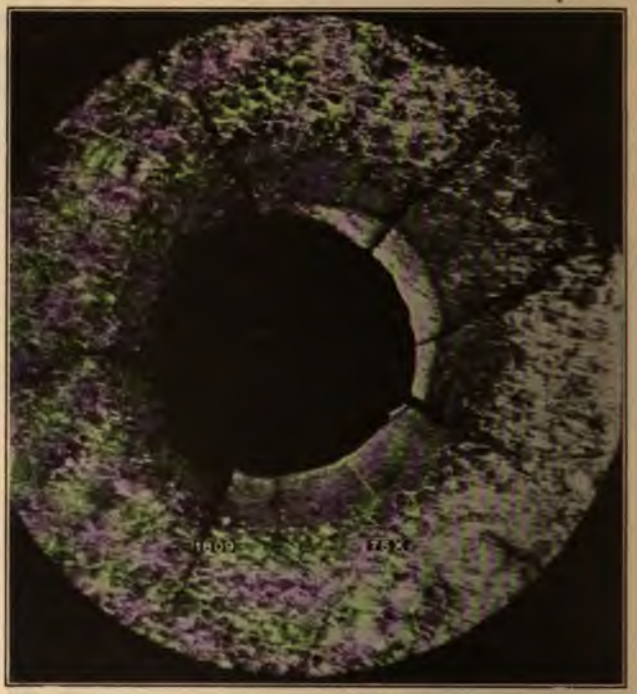

FIG. 8.—× 50.

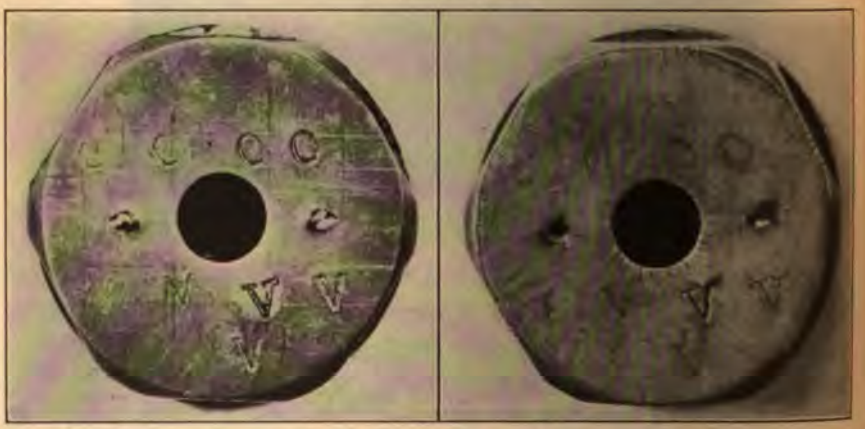

FIG. 9.—PRESSURE PLUG BEFORE HAVING BEEN PLACED IN SERVICE AND THE SAME PLUG AFTER HAVING BEEN IN A GUN WHICH HAD BEEN FIRED 100 ROUNDS.

The surface of the metal (right) has been worn away by some volatilization of the metal and by the polishing necessary to produce the structure.

plug, each showing partial conversion from soft to hard structure at the point of maximum cold work.

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This evidence seemed to favor the view that cold work was an important factor in the martensitization[6] of the metal.

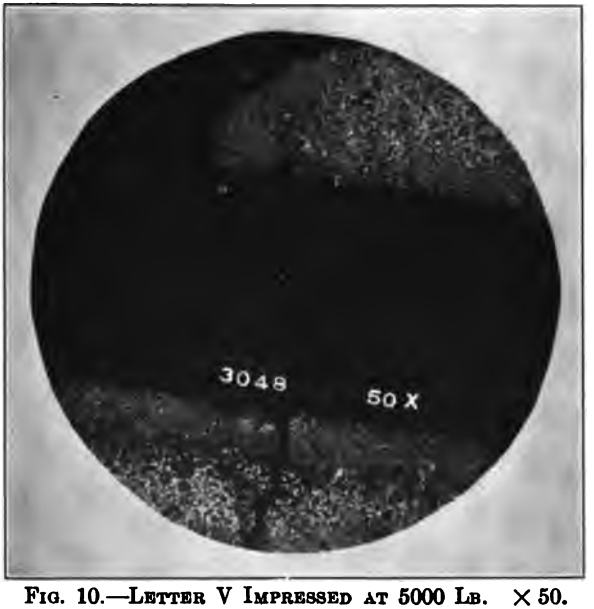

FIG. 10.—LETTER V IMPRESSED AT 5000 LB. × 50.

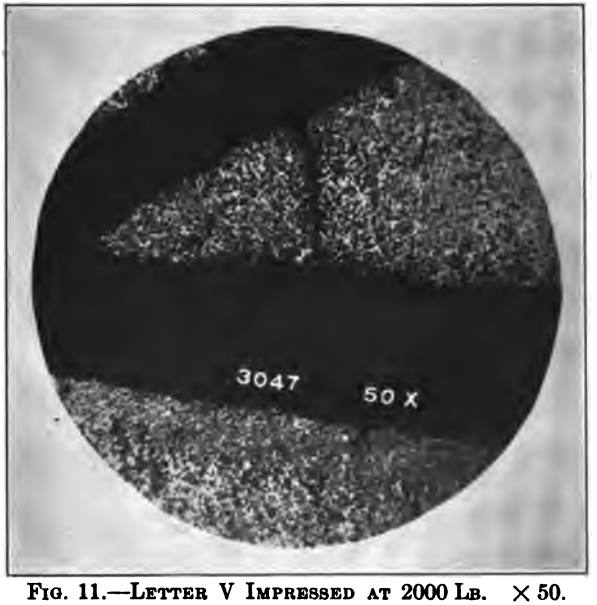

FIG. 11.—LETTER V IMPRESSED AT 2000 LB. × 50.

[6] The process of producing hardness by converting the free iron carbide into solid solution of iron carbide in iron. This is commonly done by quenching the steel from above Ac_1.

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In order to test this view systematically, a second pressure plug was prepared and the letters V and O were stamped on the face of the plug

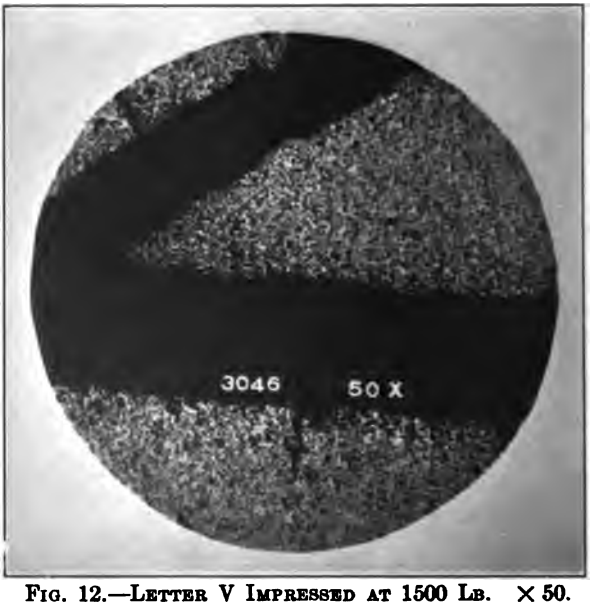

FIG. 12.—LETTER V IMPRESSED AT 1500 LB. × 50.

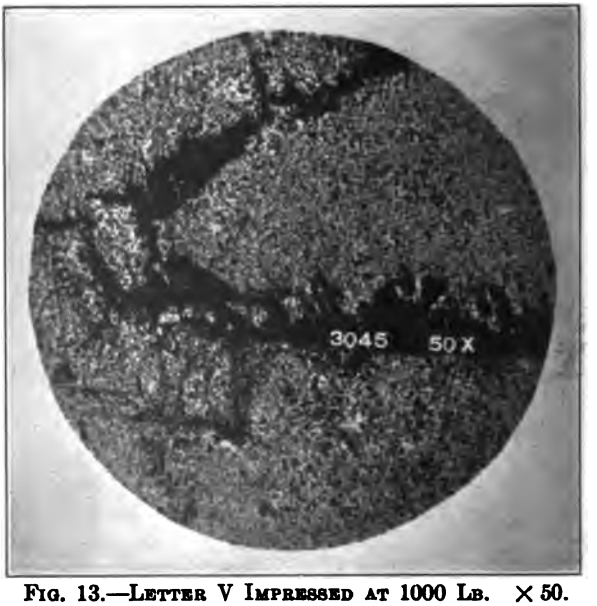

FIG. 13.—LETTER V IMPRESSED AT 1000 LB. × 50.

at varying pressures. This was done by holding the stamping die in one jaw of a tensile testing machine and the plug in the other jaw. The

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machine was now put into compression and the letters stamped at varying pressures, from 500 to 2,000 lb., as shown in Fig. 9. By mistake, one of the letters V was stamped at 5,000 lb. instead of 2,000 as originally intended.

The pressure plug was then put in service and the gun was fired 100 rounds, when the plug was returned for examination. The original sorbitic condition of the metal was found to have changed to the troostitic condition at those points of maximum cold work and to have remained practically unchanged at other points. Thus the first step in the change: sorbite→troostite→martensite had been accomplished. Figs. 10, 11, 12 and 13, show the letter V at 5,000, 2,000, 1,500 and 1,000 lb., respectively.

FIG. 14.—LETTER V SAME AS SHOWN IN FIG. 10, BUT AFTER FIRING 200 ROUNDS. × 50.

In Fig. 10, the troostitic character of the metal is clearly shown on the edge and apex of the letter V. The letter impressed at 2,000 lb. also shows the change, but not advanced so far as in the letter of greater pressure. At 1,500 lb. the change has begun but the advance has not been far, and at 1,000 lb. pressure the change is not noticeable.

In the same plug similar changes were noticed on the edge of the central hole, on the outer edge of the plug, and on two periods stamped at an angle of 45°. The period was strongly cold-worked on one side and the change was well-marked.

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The plug was again placed in service for 100 rounds and then returned for examination. The result is shown in Fig. 14, which is the letter V impressed at 5,000 lb. It will be noticed that the transformation has proceeded to its final stage, viz., the light etching structure which had previously been shown to be martensite.[7]

These experiments seem to prove that the hardness of the surface of a gun tube as well as the pressure plug is due to a combination of mechanical deformation and a process of martensitization. This process requires time, and the development takes place in stages. Even in the early stages there was evidence of heat cracking before the change of structure had taken place. When the structure begins to change it first shows as troostite, and then develops into martensite, but the latter is amorphous[8] in form. Temperature and work are also important factors. The transformation takes place with the greatest rapidity where the work has been most severe.

These factors counterbalance each other in the gun to a certain extent. In the powder chamber there is the maximum temperature, but the minimum amount of work. The only work which has been done upon the metal here is the original machining of the gun. In section *E* where the rifling begins, the maximum amount of cold work is done. The rotating band is undeformed and the whole surface bears on the rifling. This is also true of section *D*, but in each section toward the muzzle there is a decreasing amount of work due to the grooving of the rotating band until at the muzzle end the driving edge alone receives work.

The reason that a cold-worked metal is more susceptible to the sorbite→troostite→martensite change when exposed to firing is not clear. There may, however, be two factors at work: 1. Beilby has shown that mechanical deformation of any kind produces an amorphous state in the metal, and it is entirely possible that the amorphous metal may be more sensitive to the reaction, *i.e.*, it may have greater solvent power for iron carbide. He has shown, and many others have confirmed the observation, that such metal has a greater solution pressure and it is possible that it may have greater dissolving power for iron carbide.

2. Spring has shown that certain reactions, which ordinarily take place when under the influence of heat, can be made to take place at the ordinary temperature when under the influence of pressure. Thus he caused dry sodium carbonate to react with dry barium sulphate with the formation of sodium sulphate and barium carbonate. He also caused

[7] There seems to be a difference in the degree of magnification of Figs. 10 and 14, but this is due to a wearing away of the metal by volatilization and polishing, thus exposing the deeper and narrower part of the groove of the letter.

[8] Capable of being easily converted into troostite, but showing none of the characteristic needle structure.

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powdered zinc to alloy with finely divided copper to form brass. In each case precautions were taken to eliminate any heat effects.

In a gun the original metal is sorbitic and the iron carbide is therefore mostly in the free form, only a minimum amount being held in solid solution. When subjected to the pressure of cold work, or the pressure of the explosion at a high temperature, solution may take place, each round of firing adding to the amount which goes into solution. This would lead naturally to the formation, first, of troostite, and when the solution is complete, of martensite. These steps are actually observed.

Either hypothesis—solution in amorphous metal, or solution by pressure—may explain the phenomena observed, but the latter seems the more likely. This would more adequately explain the fact that in erosion tests of steels of various composition, those steels high in nickel or manganese show the greatest amount of martensite formation. In these steels the Ar_1 point is lowered slightly, thus increasing the tendency for the carbide carbon to be held in solid solution form. Thus, in one case, a 3.50 per cent. nickel steel showed a martensite surface 0.05 in. in depth, while under exactly similar conditions a plain carbon steel showed only 0.004 in. Nickel and manganese lower the Ac_1 point, and hence make it easier for the sorbite→troostite→martensite transformation.

Experiments are now in progress with alloy-steel pressure plugs to see whether or not the hard surface develops more rapidly with them than with plain carbon steels. It is highly probable that those steels having the highest Ac_1 point will show hardening the least rapidly. A convenient means of testing this is to make up a series of steels into the form of small screws, stamp some letter at 5,000 lb. pressure, place them in the mushroom head of the breech block, and make observations after firing.

The preceding work was carried on at the Watertown Arsenal and the writer wishes to express his appreciation of Colonel Wheeler's interest and his courtesy in allowing the data to be published.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Problems Connected with the Recovery of Petroleum from Unconsolidated Sands

BY WILLIAM H. KOBBÉ*, ST. LOUIS, MO.

(New York Meeting, February, 1917)

I. INTRODUCTION

THE word recovery as used in this paper is applied in its broader sense and not limited to wells producing from horizons of unconsolidated sands. Certain problems connected with the winning of petroleum from such horizons present themselves while drilling is in progress and should be considered before drilling is commenced wherever unconsolidated sands are expected. Maximum recovery depends not alone upon efficient pumping methods but also upon the selection of a proper drilling system and the completion of a well especially adapted to the extraction of these oil-bearing sands.

II. DRILLING

DETERMINING FACTORS IN SELECTION OF METHOD

The selection of a drilling method for the development of a region in which the oil reservoir consists of unconsolidated sands is based on the following considerations: (1) Character of the formations overlying the reservoir; (2) thickness of the oil-bearing stratum or strata; (3) thickness and character of water or gas sands; (4) gas pressure to be encountered or expected; (5) total depth of well.

1. *Character of the Formations Overlying the Reservoir*

The strata overlying the reservoir may consist of beds of shales, clays, sands and other soft materials or they may be slates, limestones and sandstones consolidated to all degrees of hardness. Very often the overlying strata are predominantly of soft materials but interbedded with harder formations. Likewise, soft strata are encountered at different horizons in an overburden consisting almost entirely of the harder sedimentaries. The physical character of this overburden is frequently

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complicated by the occurrence of quicksands, gypsum, rock salt or iron pyrites.

In deciding upon the surface equipment best adapted for the penetration of these various formations composing the overburden, the governing factor is based upon evidence or expectations that the strata are: (*a*) predominantly soft, (*b*) predominantly hard, or (*c*) about equally divided.

If it is known that the overburden consists of predominantly soft and cavey material such as sands, clays, soft shales, quicksands, beds of gypsum, etc., a rotary drilling rig of a size and type commensurate with the depth would be the most efficient method of overcoming such conditions.

On the other hand, if it is known that the overburden consists of predominantly hard strata such as limestones, sandstones, hard shales and slates, cable-tool equipment would be employed and a standard rig, with or without calf wheels, erected.

In many parts of the world it is impossible to classify the overburden either as predominantly soft or as predominantly hard. It is made up of many strata of each class. Such conditions are typical of the deep and difficult drilling in certain parts of California, for example at Fullerton and in portions of the Midway district. The only efficient method of drilling wells through 3,000 or 4,000 ft. of these mixed strata is the combination system employing both cable and rotary on the same rig. A heavy combination equipment, with a 24 by 112-ft. derrick, iron crown block, latest type of draw works and large rotary, frequently costs as much as a completed well located in the comparatively shallow portions of the Mid-continent or Eastern fields.

2. *Thickness of the Oil-bearing Stratum or Strata*

The thickness of the unconsolidated pay sand that must be penetrated is an important factor in the selection of the drilling system. One hundred feet of loose oil sand carrying even a moderate gas pressure often necessitates the use of rotary equipment notwithstanding the fact that a decision based solely upon the physical character of the overburden would select cable tools for the work. For example, in the extreme northern portion of the Midway field in the San Joaquin Valley, California, there is 700 ft. of predominantly hard cable-tool overburden which must be penetrated in order to reach an unconsolidated oil sand 280 to 300 ft. thick. Experience has shown that the rotary drilling method completes a well in that district in half the time and at less cost than cable tools. This for the reason that the former system averages more hole per day from top to bottom than the latter although more time is required by the rotary to reach the sand. The cable tools have no difficulty "making hole" through the 700 ft. of predominantly hard

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overburden but encounter the greatest difficulty in making headway in the oil sand. In fact, the heaving nature of these sands causes innumerable fishing jobs and frozen strings of casing when an attempt is made to overcome them with cable equipment. After fighting them for weeks, if the standard driller has 200 ft. of pay to his credit he is doing well, whereas the same driller on a rotary rig can wash through the 300 ft. of sand in one tour of 12 hr.

These conditions indicate the important bearing the thickness of the unconsolidated pay sand has upon the problem of selecting a drilling system.

A few miles south of the area just described the oil sand is only 10 ft. thick and underlying similar formations of hard strata. Here the cable tools are far superior to the rotary because they can readily overcome the thin bed of pay sand and are better adapted to drilling through the overburden.

Therefore, it is seen that the mere thickness of unconsolidated oil sand may be the controlling factor in classifying the physical nature of the territory to be drilled.

3. *Thickness and Character of Water or Gas Sands*

Cable tools are superior to the rotary for prospecting new territory and for determining the exact depth of water- or gas-bearing horizons. It sometimes happens that an area or pool has been inefficiently developed and exploited; that available well records are unreliable, or that the lenticular nature of the sands causes great uncertainty regarding the exact depth of water strata. Under such conditions it is advisable to employ cable tools until all necessary data have been obtained, when the drilling method may be advantageously changed to the rotary. Conditions such as these at one time existed in portions of the Burkburnett field in northern Texas and although it was known that the territory was particularly adapted to rotary drilling it was necessary to "feel out" from proven areas with cable tools on account of the "spotted" nature of the oil and water sands.

4. *Gas Pressure to be Encountered or Expected*

Gas pressure is a factor of the utmost importance in the selection of a drilling system. Other things being equal, the greater the pressure the greater its importance; not only from the mechanical standpoint but to serve the ends of conservation. This pressure may occur above the oil reservoir, within the reservoir, or both. When it is known to occur above the oil sands it may be cased off with cable tools and conserved between strings of casing by the use of a packing spider, a special

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device which has met with great success in the deep and high-pressured territory of southern California.

Where high gas pressures exist in formations difficult for a rotary to overcome it frequently becomes necessary to utilize special methods in conjunction with the cable system to properly cope with the situation. For example, the mud-laden fluid method and circulator systems may be called to the aid of the cable tools in passing through strata under high gas pressure.

The rotary system, however, is the ideal method of overcoming gas sands and should be employed wherever it is possible to drill by this method. In portions of the north Midway field the pressure within the unconsolidated and very loose oil pay is sufficient to heave the sands 200 to 300 ft. up in the casing of cable-tool wells unless special safeguards are introduced, whereas the rotary with its column of mud slip penetrates these sands with the greatest ease and dispatch.

5. *Total Depth of Well*

This is naturally an important factor in the selection of a drilling system, whether the oil sands to be developed are unconsolidated and loose or hard and compact. No attempt will be made in this paper to describe the many different types of rigs and methods of drilling. As has already been pointed out, the character and thickness of the oil sand often modifies the selection of any particular drilling method but the same sand at 1,000 ft. requires lighter equipment and possibly a different system than if it occurred at 4,000 ft. The former may be easily reached with cable tools and a 20 by 84-ft. standard rig while the latter may require the heaviest rotary or combination equipment with a 24 by 112-ft. or even 120-ft. rig, 7½-in. Ideal rig irons, calf wheel with sprocket drive, etc.

Wildcat Territory

In the foregoing paragraphs it was assumed that developments of unconsolidated oil sands was to be undertaken in proven territory–that is, in a region already known and where accurate well logs were available for study and correlation. This is a very frequently not the case. Oftentimes little or nothing is known concerning underlying strata and unless geological evidence enables the construction of a fairly accurate columnar section of the region the selection of a drilling system is practically limited to cable tools. As has already been stated, this is the only method of pracedure in unknown or wildcat districts because the cable system is peculiary adapted to prospecting and the accurate determination of oil-, water- and gas-bearing strata.

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Johnson and Huntley[1] make a good comparison of the advantages and disadvantages of the cable and rotary systems which indicates this superiority of the former for work in wildcat districts.

Comparison of Drilling Systems

Cable System.—Advantages.—

1. Less first cost of tools and rig.
2. Lower labor cost per day.
3. Less water necessary.
4. Can drill in the hardest rock.
5. More drillers available in some fields, although this is becoming less true.
6. Gives more information as to the formations passed through, and is thus better for prospecting.
7. Less cost per foot for relatively shallow wells.

Disadvantages.—

1. Longer drilling time.
2. Much slower when under-reaming is necessary.
3. Danger of delays and fishing troubles in soft strata.
4. When many water sands, hard to carry large hole to deep pay.
5. Greater cost per foot for moderately deep wells.
6. More casing necessary to handle caves and water sands.
7. Liability of getting crooked hole in soft formations.
8. Harder to control heavy pressures and more likelihood of "blow-outs."

Rotary System.—Advantages.—

Faster drilling in soft strata.

2. Less trouble from caving and water sands.
3. Less casing used in soft formations with water and gas sands.
4. Straighter hole in deep drilling in soft formations.
5. Can handle alternate hard and soft formations, with less danger of accidents than with cable tools. This is made possible by the new bits and heavier rotary machines.
6. Can carry a large hole deeper.
7. When "drilling in," easier to control high gas pressure and prevent blowouts.

Disadvantages.—

1. Very slow in hard strata.
2. Greater daily labor cost.

[1] R. H. Johnson and L. G. Huntley: *Principles of Oil and Gas Production*, pp. 118–119. New York, Wiley (1916).

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3. Limited trained labor supply in some fields.
4. Greater cost per foot for shallow wells.
5. Does not show up smaller oil and gas pays, and important reservoirs may be passed through in prospecting.
6. More water necessary, a drawback in arid regions.

Cable System Generally Used in Wildcatting

Good practice demands that cable tools be used on any test well in an unknown or wildcat district. If the approximate depth of the stratum which is thought to be oil-bearing is known, or if the depth is decided by contract with the landowners, a standard or calf-wheel rig is erected of a size and type commensurate with that depth and drilling commenced with cable tools. If the territory proves to be suitable for cable-tool work and stands up, the well progresses satisfactorily and rapidly as a rule but if caving and unconsolidated sands are encountered, complications surely follow. These unforeseen contingencies will be treated briefly.

A test well passes through 1,500 ft. of hard formations and enters a stratum of unconsolidated oil-bearing sand under pressure. The 10-in. casing is landed at 1,450 ft. as a water string to shut off several water-bearing strata encountered at higher levels. The 8¼-in. casing put in after landing the 10-in. becomes "logy" and finally "freezes" when 40 ft. of the sand has been drilled.

The foregoing case is typical and one taken from actual practice. It is essential to keep the casing "free" when carrying hole through unconsolidated sands, whether pay sands or not. Sometimes 10 ft. of sand will freeze casing and in other cases with careful management in experienced hands 150 to 250 ft. of heaving oil sand may be overcome with cable tools.

In the example just mentioned, the 8¼-in. casing is frozen. The first thing to be done is to attempt freeing it. Alternate pulling and driving may accomplish this. The power of a calf wheel applied through seven lines to the heavy casing block is tremendous and an expert driller knows just the amount of stretch the casing will withstand before parting. If it comes an inch or two the drive clamps are adjusted on the square of the stem and the casing driven back to its former position, when it is again pulled—possibly yielding an additional inch. The principal thing is to cause it to move even slightly, as it is sure ultimately to become free by this method. It is well to apply the casing tongs after each pull or two and set up the entire string a trifle. This tightens any threads that may have become loose.

It frequently happens that the alternate pulling and driving method proves futile. The casing refuses to move an inch. In this event other methods are at the command of the engineer. One that is often success-

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ful and is not commonly known is to relieve the static pressure within the casing, it being presupposed that the hole is full or nearly full of fluid, and allow the sand to heave within the drilling string. It is seldom necessary to bail more than 300 or 400 ft. of fluid from the well in order to so disturb the balance of pressures that a sudden upheaval of the oil sand takes place. Following this upheaval the driller tries the casing and if it is free, which is nearly always the case, the hole is cleaned out to bottom and drilling resumed.

It sometimes happens on account of insufficient pressure in the sand, or from other causes, that the sand fails to heave when the static pressure is reduced. In that event a casing spear, preferably of the trip type, is run to bottom on the tools and engaged near the shoe, with long stroke jars and sinker bar or stem giving necessary impact to the jars. A strong pull is then taken with hydraulic jacks applied to the casing by means of a spider, while jarring is commenced on the tools. This method will free exceedingly tight casing. A nail may often be driven out when it cannot be pulled. The jarring of casing is exactly similar.

If this method fails to free the casing, a spear may be run on a string of the next smaller size casing and a hold secured near the bottom of the frozen pipe. A heavy pull is then taken on both strings of casing while jarring is commenced with the tools which carry long stroke jars and a casing spear as described in the previous method. In fact, the two methods differ only in that a much stronger pull can be taken on the two strings of casing than on the one; in other words, the pull is doubled.

In rare cases it happens that the unconsolidated sand holds the casing in such a grip that even this tremendous pulling force and the heavy jarring, which may be likened to the blows of a steam hammer, fail to free it. As a last resort the casing may be split in several places to allow the binding sand to run into the well. To accomplish this a casing ripper is run in on the tools and several long gashes cut in the casing at a point where it is thought the maximum "friction" exists. If this results in the entry of sand it will nearly always free the string.

In the event that all these attempts fail and it is impossible to free the casing, two methods of overcoming the difficulty remain. A four-wheel casing cutter may be run in on 3-in tubing and the casing cut at a point just above where it is frozen. The upper portion of the string is then removed from the well, a new shoe adjusted and the joints imbedded in the sand "side-tracked" or drilled past. This is slow and difficult work and not always successful. Some operators shoot the casing instead of cutting it. A small charge of dynamite is exploded at the point where the casing is frozen, but this results in jagged ends at the place of rupture, making the casing more difficult to side-track. It is a quick and inexpensive method but cannot be considered good practice in many cases.

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If it is not desired to cut or shoot the frozen string, the only thing to be done is to abandon it and case with the next smaller size pipe. This is permissible providing the frozen string is of such a size, say 8¼ or 6⅝ in., that the well may be completed to the required depth before "pointing out."

Loose oil sand under pressure often heaves in a cable-tool drilling well when the bit drills through the cover rock, and unless the driller is warned in time the tools are buried for many feet by the sand. In certain of the California fields the oil sand has often buried a string of tools 300 ft. or more and frequently led to a bad fishing job. Whenever such sands are anticipated the experienced driller "carries" the hole full of water or thin mud slip in order to counterbalance the rock pressure. The jars and sinker bar are also used on the tools so that in case of a sand heave they may be jarred free. It is sometimes necessary to jar the tools through 100 ft. or more of oil sand, which requires possibly 12 to 24 hr. If the wire drilling line parts a bad fishing job is apt to result and it may be necessary to pull the casing (if it is free) in order to recover the tools. Or if it is frozen a "fishing string" is run in, after cleaning out to the top of the lost tools, and an attempt made to recover them with a socket or other special fishing tool. This is simply an example of one of the many fishing jobs of endless variety that may occur from the heaving of unconsolidated oil sand. These same remarks apply with equal force to conditions arising from the penetration of any unconsolidated sand or other material of a cavey nature which may freeze casing or bury cable tools.

The foregoing problems connected with the drilling of wells with cable tools in unconsolidated sands are, of course, not met with when the rotary system is employed, but it must be remembered that wildcat territory, which demands the use of cable tools, is being considered. The difficulties encountered were unforeseen and unexpected. After the cable tools have proved the district, future exploitation may be carried on by the rotary system.

Rules Governing the Drilling of Deep Test Wells

A few general rules and cautions governing the drilling of deep test wells in wildcat territory where unconsolidated sands are expected, follow:

1. Allow a wide margin above the calculated cost of the well.
2. Commence drilling with a sufficiently large hole—20 to 22 in. for a deep test.
3. Do not expect a light rig for shallow territory to drill a deep test.
4. In remote districts have all the casing on the ground and an adequate supply of small tools, pipe and fittings.
5. Have a rope grab, combination socket and other frequently needed fishing tools available for instant use.

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6. After selecting the site for the rig, have a cellar dug about 8 by 10 ft. by at least 15 or 20 ft. deep. The depth is important.

7. Provide an adequate supply of the best obtainable water for steaming purposes.

8. Anticipate heavy oil or gas pressure and have a control casing head on hand if occasion arises for its use to cap the well.

9. Locate a deposit of fine clay for use in mixing mud slip to shut off pressures or to use while drilling in unconsolidated sands that refuse to bail readily.

10. If the well is to be drilled by a contractor at so much per foot, do not expect sands tested, water shut off, and logs kept as carefully and efficiently as would otherwise be done. Many important features of the work are slighted and the time required for their proper accomplishment sacrificed in the effort to make the maximum amount of hole in the minimum time.

11. Carry every string of casing to the maximum depth possible and never abandon the effort until every means at the command of the operator has been exhausted in the attempt.

12. The management of the work, selection of methods, etc., should not be left to the drillers but placed in the hands of a competent superintendent.

13. Keep a most careful and accurate log of the well, preferably in several copies in order that if one is lost the record may be preserved.

Taking up some of these points in more detail: In allowing for the cost of a test well in wildcat territory a safe rule to follow is to figure the actual cost as accurately as possible and multiply this figure by two. How many wells are reported as "abandoned for lack of funds?" It is safe to say that unconsolidated sands or other cavey materials are responsible for a large percentage of the failures and for most of the difficulties, discouragements and delays in the drilling of test wells. Many wells that "ought not to cost $15,000" actually require the expenditure of $25,000 or $30,000 for their completion. Much larger amounts are not at all unusual and it is well to keep in mind that figuring the cost of a test well in a wildcat district is a good deal like provisioning an exploring party to penetrate unknown wilds—emergencies must be anticipated in both cases.

The importance of commencing the well or "spudding in" with a sufficiently large hole cannot be over-estimated. In order to reach the desired depth, whether 2,500 or 4,000 ft., it is imperative to make the size of the hole at the surface commensurate with the depth. In the soft Tertiary formations of California and in the Baku fields of Russia it is found advantageous to use large-diameter stovepipe casing for the first few hundred feet, especially where beds of boulders are encountered. This type of casing is made by riveting sheet iron or steel of 14 to 10 U. S.

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gage and may be purchased in lengths of 2 ft. or multiples of 2 up to 20 ft., the shorter lengths being used when progress with the string becomes slow. In the Russian fields, wells are commenced with holes of extremely large diameter, not on account of any extraordinary depth but in order to complete a well in the pay sand of sufficient diameter to permit the use of large-sized bailers, the method used in that country to bring the oil to the surface, because the excessive amount of unconsolidated sand precludes the use of pumps. The usual sizes of the California stovepipe casing are 16 in., 18 in. and 20 in. and if 500 to 700 ft. of the 18-in. size can be put in a well the operator has a good beginning for a deep test in unconsolidated formations. A great advantage possessed by this type of casing is the absence of collars with a consequent reduction in friction, making it especially suited for driving through strata of unconsolidated sands. Its use will probably extend to Oklahoma and the Mid-continent fields as development extends westward into deeper territory possessing all the problems arising from the presence of unconsolidated sands or other strata of loose materials. Many test wells have been and are being drilled in the Permian Red Beds of western Oklahoma and the occurrence of deep deposits of unconsolidated materials in this region is gradually bringing about improved drilling methods and the introduction of more efficient mechanical devices. For the same reason, operators accustomed to the hard strata of the Eastern fields where casing problems are of small importance are realizing the necessity of large-diameter holes and the use of heavy-weight pipe with improved methods of handling it. Commencing a test well with too small a diameter means that it "points out" before it "tests" anything. In Pottawatomie County, Oklahoma, a typical wildcat section, are many abandoned "test" wells that pointed out before a depth of 2,500 ft. was reached, although it was known that if oil occurs it would be found at 3,000 ft. or more. These failures were all due to the fact that the holes were commenced with diameters too small and that the presence of unconsolidated sands was not taken into account. These sands necessitated the use of a greater number of strings of casing which in turn caused the well to point out sooner than would have been the case had it been commenced with a hole of larger diameter. "It may be said in conclusion that in drilling in wildcat territory where the number of water and gas sands is unknown and the depth of the oil sand uncertain, and therefore the ultimate depth of the hole cannot be known, one must start with a larger size of casing than that which will probably be used for later wells when the field is developed. Pioneer wells have been drilled in some pools, which failed to discover oil because the hole was so small it could not be carried deep enough to penetrate the sand which was later discovered to be the main oil pay, perhaps only a few feet beyond where the early well stopped drilling. Wells in the Calgary dis-

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trict in Alberta are started with 18-in. casing. In California even larger casing has been used."[2]

Probably the best type of rig for handling heavy strings of casing in unconsolidated sands is the California or calf-wheel with sprocket drive and extra heavy rig irons. A light standard rig such as is commonly used in the Eastern fields and parts of Oklahoma is sufficiently strong to handle light-weight strings of casing through consolidated formations but is totally inadequate for the heavy pulls required to free long strings of California pipe in contact with several hundred feet of unconsolidated sands. For the heaviest work, iron or oak-calf and bull-wheel shafts are required, and concrete foundations for the derrick. The usual practice of placing a few pieces of waste lumber under each derrick leg as foundations should not be countenanced, especially where the casing may have to be carried through cavey material with the consequent heavy strains which inevitably result in pulling the derrick out of plumb.

A feature connected with cable-tool drilling in unconsolidated sands which many operators, accustomed to hard formations, overlook, is the importance and advantage of an adequate cellar. When a string of casing is being carried through sands or other cavey material it has to be "worked" almost continually and sometimes freezes while one "screw" is being run. It is difficult or impossible to make any hole ahead of the shoe and frequently the casing is allowed to "follow;" that is, it is released from the spider slips and follows the drilling bit of its own weight. If it refuses to follow it is worked up and down with the casing block and calf wheel and sometimes spudded with a jerk-line or driven with the drive clamps and tools. In all of these operations a deep cellar is of the greatest advantage because it allows the driller to work the casing below the derrick floor until there is sufficient clearance for another joint to be screwed on. Casing averages about 20 ft. to the joint, hence a cellar of approximately that depth is most desirable, although 15 or 18 ft. is a great improvement on the inadequate and shallow pits commonly placed under derricks in regions where the problem of unconsolidated sands is unknown or under-estimated.

Finishing a Well in Unconsolidated Sands

The process of finishing a well in an unconsolidated pay sand is quite different from the "bringing in" of a well tapping a hard sand. It sometimes requires weeks to penetrate a thick stratum of loose pay sand with cable tools, whereas the "drilling in" and "shooting" of a hard sand is a much simpler process. Where the sand, whether loose or hard, is under great pressure and the well "comes in" as a gusher or flowing well

[2] R. H. Johnson and L. G. Huntley; *Principles of Oil and Gas Production*, p. 125. New York, Wiley (1916).

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as soon as the sand is tapped, the results are exactly similar except that the well in the unconsolidated pay is apt to throw out great quantities of sand.

The ideal method of finishing a well through a thick stratum of loose oil sand is the rotary system. As mentioned in the first part of this paper, it is usual for a rotary to penetrate 300 ft. of unconsolidated oil pay in 12 hr. or less, and by thickening the circulating mud the pressures may be balanced to such a nicety that upon reaching the stratum underlying the sand the entire string of drill pipe may be withdrawn without the slightest danger of caving. In other words, a column of mud-laden fluid extends through the sand and is substituted in the physical balance of forces for the loose sand which previously occupied the same space. Although all the purposes of an open hole have been served, nature's balance is left undisturbed. This mud may be allowed to stand for several hours but the best practice is to prepare the "liner" or perforated casing in advance and have everything ready for "casing" immediately upon withdrawal of the drill pipe.

The liner may consist either of "shop-perforated" casing or one of the several types of screen pipe which have come into extensive use because of their greater efficiency in handling sand with a consequent increase in the amount of oil recovered.

It will be assumed that screen pipe is to be set and that the water string consists of 8¼-in casing landed in a hard stratum a few feet above the oil sand. Three hundred feet of 6⅝-in. screen pipe is carefully tallied and two blank joints of the same size made ready. A collar is screwed on the bottom of the last joint of screen pipe and a back-pressure valve and wooden wash plug placed therein. The 300 ft. of screen pipe with the two blank joints on top is then run into the hole to bottom on 3-in. tubing, this tubing resting on the wooden wash plug and being attached to the 6⅝-in. casing with a right- and left-thread nipple. The principal thing is to have the tubing extend through the screen pipe in order that the water may pass out of the bottom of the liner and not through the meshes of the screen. When the liner has reached bottom it is raised an inch or two and the well thoroughly washed by pumping clear water through the tubing or wash pipe for 24 hr., or until the "returns" cease to show any trace of mud. The wash pipe is then backed off by turning to the right and withdrawn. This leaves the liner clean, with no mud between the meshes and the oil sand, and the two joints of blank 6⅝ in. extending up into the 8¼ in. forming a 40-ft. lap. An adapter from 6⅝ in. to 8¼ in. is then lowered on the bailer or dropped through the water in the casing until it rests on top of the liner, making a smooth connection between it and the 8¼ in. The well is then tubed and when the water pressure is relieved begins to produce oil.

The proper setting of screen pipe with a rotary requires experience

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and knowledge, especially as quick work is desirable and any accident may prove costly. Instead of using tubing, a heavy liner, whether of screen pipe or perforated casing, may be set with the rotary drill pipe, as the same method is used with both.

The finishing of a well with cable tools in a very thick stratum of unconsolidated oil pay is much more of a problem than that just described where a rotary is used. The ordinary procedure, especially in districts where screen pipe has never been tried or is unknown, is to carry the casing, of the size the hole is to be finished with, through the oil sand and then perforate it with a perforating machine. It frequently happens that the string freezes after penetrating the sand for 100 ft. or less, in which event it is usual to place an iron or wooden heaving plug in the bottom of the casing and then perforate. If the operator is fortunate enough to carry the casing through the pay it can be landed in the stratum underlying the sand, which obviates the use of a heaving plug, the only function of which is to prevent the sand heaving inside the casing and overwhelming the pump. Irrespective of the depth of sand attained, the use of the perforator is necessary with this method and is its greatest drawback, mainly on account of the uncertainty of the results therefrom. For mechanical reasons, variation in the thickness or toughness of the casing, and other factors beyond the control of the operator, the use of a perforating machine is frequently disappointing. If the well comes in as a small producer there is always the suspicion that the perforator failed to punch a sufficient number of holes, whereas if the casing "sands up" and the well requires constant "pulling" the reverse is true. Arnold and Garfias[3] mention these points in describing methods of oil recovery in California:

"The indiscriminate use of perforating machines has been the source of much trouble. In the Coalinga field one well about 1,300 ft. deep was perforated three times, and when the casing was removed it was found that, except for a few holes, the machine had only indented the casing. In other wells, owing to local brittleness of the casing, the perforator has removed large pieces of it; and in some deep wells, owing to the great weight of the column of tubing to which the perforator is attached, the perforator has cut into or strained the collars so as to cause collapse of the casing."

In the light of these facts and of past experience in drilling many wells in unconsolidated oil sands, it may be said *that every effort should be made to avoid bringing in the well by perforating the casing after it has been placed in the hole.* As a substitute for such perforating the operator may use screen pipe or shop-perforated casing. In finishing a cable-tool well with either of these two devices the problems involved are largely or entirely dependent upon the thickness of the unconsolidated sand. If this sand is a thin bed 50 ft. or less in thickness there is usually no difficulty in

[3] R. Arnold and V. R. Garfias: Methods of Oil Recovery in California, *U. S. Bureau of Mines, Technical Paper* 70, pp. 9–10 (1914).

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setting any type of screen or of shop-perforated casing, but where the sand is 250 to 300 ft. thick many obstacles may be encountered.

The ideal method of finishing a cable-tool well in a very thick stratum of loose pay sand is to carry, say, 8¼-in. casing *through* the pay and then set a 6⅝-in. liner, either screen or shop-perforated, preferably the former, inside this 8¼ in. and then remove the 8¼-in. casing, leaving the liner in contact with the oil sands. This is very difficult to accomplish, however, because it necessitates keeping the 8¼ in. free at all

After Layne & Bowler Corp.

FIG. 1.—SECTION OF WIRE-WOUND SCREEN CASING.

After McEvoy Wireless Well Strainer Co

FIG. 2.—SECTION OF WIRELESS SCREEN CASING.

times, and this is seldom possible. In fact, it may freeze beyond all power of freeing while the liner is being set.

It well may be asked, why not carry 8¼-in. screen or shop-perforated pipe through the sand in the first place and thus save the extra labor of setting a liner and gain the advantage of completing a well of larger diameter. The answer is that it is much more difficult to carry perforated than blank casing through unconsolidated sands and that all types of screen pipe possess the same objection. In addition, certain screens are not made to withstand such work and are easily damaged, necessitating the use of a conductor casing when setting.

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Screen casing may be divided into two general classes: (*a*) the screening device on the outside of the casing, see Fig. 1; and (*b*) the screen forming a part of the casing wall, see Fig. 2. In the first type the screen generally consists of wire wound around perforated casing, while in the second type the screen is in the form of numerous buttons or slotted brass plugs carried in the walls of the casing and flush with the outside surface, or in slits cut directly in the casing wall itself. This latter type, sometimes known as "wireless" screen, has the advantage of withstanding hard usage, and many operators have carried it for considerable distances through unconsolidated sands. But in very thick strata of loose pay it is as difficult to carry as shop-perforated casing and entirely impossible where blank pipe encounters difficulties. The wire-wound screen casings are entirely unadapted for carrying through thick strata of unconsolidated sands and the attempts to finish a well in that way should never be made. The wire screen, however, possesses many advantages over the wireless, as will appear in later paragraphs, but in finishing a well with this type of screen it is necessary to set it either inside a conductor string or by means of a rotary, as otherwise the wires may become detached from the casing and cause endless complications and trouble.

In concluding the subject of the drilling and finishing of wells in unconsolidated sands, it is desired to emphasize the fact that most of the problems connected therewith arise from two general causes: (*a*) the development of unknown districts; and (*b*) the existence of such a combination of physical conditions in the strata that it is impossible to employ one system capable of overcoming them. For example, if a thick body of unconsolidated oil sand has an overburden of 1,200 ft. of hard compact formations the cable system demanded by the latter is confronted, upon entering the oil sand, by many of the problems already described. These conditions would not justify the installation of an expensive combination system and could best be overcome only by the exercise of good judgment and foresight backed with skill and experience. Even though possessing these qualifications in high degree, the seeker for oil must recognize the element of "luck" in many of his operations and be duly appreciative when it is in his favor and stoically patient when it is otherwise.

III. EXTRACTING THE OIL AND SAND

The process of obtaining oil from the earth when that oil is contained in a loose unconsolidated sand is very different from the method employed to extract the recoverable portion of the oil content of a hard stratum. *With an unconsolidated sand the extracting of its oil content may take place above ground whereas with a hard oil-bearing stratum this is impossible.* This distinction has an important bearing upon all questions relating to

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oil production. It is the factor deserving the greatest consideration in determining the efficiency of methods employed for the maximum recovery of oil from unconsolidated sands and is the basic principle of the statement that: *Other things being equal, the maximum recovery of oil from an unconsolidated sand is directly dependent upon the maximum recovery of the sand itself.*

The truth of this law is based upon two important considerations: (1) The removal of sand by a well causes a larger volume of sand, and therefore of oil, to move toward that well than would be the case if oil only were removed. (2) The efficient separation of sand and oil, and therefore a nearer approach to obtaining the recoverable oil content, is made when this separation takes place above ground.

The Movement of Sand and Oil Toward a Well

The higher the viscosity and the lower the gas pressure within the oil reservoir the greater becomes the importance of creating and maintaining a movement of sand toward a producing well. This has been proved beyond all question in extensive operations in the north Midway district of California where the oil is of 14.5° Bé. gravity with a very low gas pressure.

"In the Balakhany oil district, where the oil is heavy, and the gas has nearly all escaped from innumerable wells, the concentration of the petroleum toward old producing areas is very marked, and whilst old bailing properties, which have for many years yielded a moderate payable production show little or no fall-off, new lands interspersed between these are almost valueless and give practically no output when new wells are sunk. This is additional testimony of the movement of oil towards spots where immense quantities of sand have been removed by bailing, creating areas of low density into which the petroleum, no longer assisted by a gas pressure, percolates by gravitation. In one case, a new well bored in the Balakhany district was a distinct failure, for stratum after stratum failed to yield even a mean production, but an old well which had been bailed for many years before abandonment, gave, when cleaned out, the extraordinary production of 2,500 poods a day for more than a month, and then only fell off by degrees to 400 poods daily."[4]

The fact that large producers come in without "making sand" evidences the existence of high gas pressure and insufficient means of entry for the sand to pass into the casing, but it is not proof that the large production is due to the absence of sand or that the output would not be increased if sand were expelled with the oil.

With proper mechanical arrangements a well tapping a thick stratum of unconsolidated pay produces a tremendous amount of sand. That it continues to do this for long periods is proof that a constant replenishment takes place near the casing, necessitating a movement of sand and

[4] A. B. Thompson: *The Oil Fields of Russia*, p. 51. London, Crosby Lockwood and Son (1908).

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oil toward the well. This movement is frequently of such volume, or occurs so suddenly, that the casing penetrating the reservoir is damaged or completely severed from the upper and more rigid portion of the string. Several instances of such damage may be mentioned: In one case 250 ft. of 36-lb. 8¼-in. casing was twisted somewhat in the shape of the letter S and was removed from the well only with the greatest difficulty. In another instance several hundred feet of 6¼-in., 20-lb. casing was broken just below a collar and displaced to such an extent that no part of it was touched by the tools while repairing the damage. Several cases may be cited where casing has been so deflected or bent by sand movements that it was impossible to tube the well beyond the point of flexure. No type of casing is proof against this force of moving sand, and both screen and perforated pipe suffer, although it is believed the former less so than the latter. Likewise it appears to make but little difference whether the casing passes entirely through the sand and is anchored in an underlying stratum or penetrates for only half the distance.

Methods of Extraction

In order to bring about a movement of sand with its oil content toward the well, it is necessary to provide means for its efficient removal with special casing devised to facilitate this process. If the sand cannot enter the casing, not only is the movement of oil retarded but the sand surrounding the casing is partially drained of its oil content, resulting in a constantly increasing body of "dead" sand opposed to the richer portions of the reservoir. Instances are known where a well, because of faulty perforations or inadequate screen openings, produced small quantities of clean oil free from sand for several years and when an attempt was made to substitute screen with proper-size mesh the sand had become so "dead" in the vicinity of the well that it lacked the "life" to enter the casing and continued to obstruct the path of better "pay." The same conditions apparently explain similar failures when a well was reperforated, but in such a case there is always the element of doubt that arises whenever a perforating machine is used. These cases occurred with a very loose oil sand containing an asphaltic crude of 14.5° Bé. gravity and were undoubtedly influenced by nearby wells having established strong drainage channels. In any event, the fact remains that they produced no sand and very little oil, and that these conditions could not be improved. Furthermore, there was no local variation in the reservoir or other controlling factors to account for this phenomenon.

On the other hand, when a well is properly equipped and produces great quantities of sand with the oil a large area of lower density is gradually established around the casing and, unless encroaching water is present in quantity, this removal of sand must inevitably lead to the for-

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mation of a cavity. This cavity probably assumes the form of an inverted cone as sand continues to be removed while new sand with its oil flows by gravity down the slopes of this subterranean funnel and supplies the well with fresh material. The shape of the cavity is undoubtedly influenced by the dip of the strata, when marked, and by the disturbing effect of nearby wells. That a cavity forms, however, cannot be disputed. Sand slips, damaged casing, and collapse of the cover rock evidence such a cavity in addition to the impossibility of otherwise explaining the effect of removing the tremendous volume of sand that wells have been known to produce.

It would be interesting to know when the slopes of these cavities reach an angle of repose and experiments with saturated oil sand on the surface should prove most instructive. The fact that wells producing from unconsolidated pay sands gradually diminish in their output of sand may indicate the approaching stability of these underground slopes, and evidence presented in later paragraphs strengthens such belief.

Having attempted to show the necessity, not only of removing sand in order to recover the maximum quantity of oil, but the favorable results of such removal in cavity formation with its accompanying low pressures and "live slopes," the mechanical handling of this sand will now be described.

Function of Screen Casing

Screen casing of proper mesh serves its best usefulness in permitting the *regulated entry of the maximum amount of sand and oil* while excluding gravel and small stones which would "sand up" the pump by sticking the valves, or otherwise prevent its proper action. It probably serves another use in that it results in a more even distribution of the oil and sand channels radiating from the well than is the case with ordinary perforations. This even drainage of sand and oil equalizes the pressure in all directions around the casing, whereas the use of a perforating machine may result in the sand being drained from one side more than the other, thus creating a pressure against the casing. This unequal drainage due to faulty perforation is an argument for the use of screen pipe and a possible explanation of deflection and damage to oil strings.

A section of wire-wound screen casing is shown in Fig. 1. This type of screen utilizes "keystone wires" whose function is to prevent clogging of the mesh—the space between the wires increasing radially and thus permitting any substance that enters to pass through unhindered. An exceptionally large screening surface is exposed to the sand with this type in that the wires between the perforations are utilized as fully as those directly over the openings.

Fig. 2 shows a section of wireless or button screen which is especially

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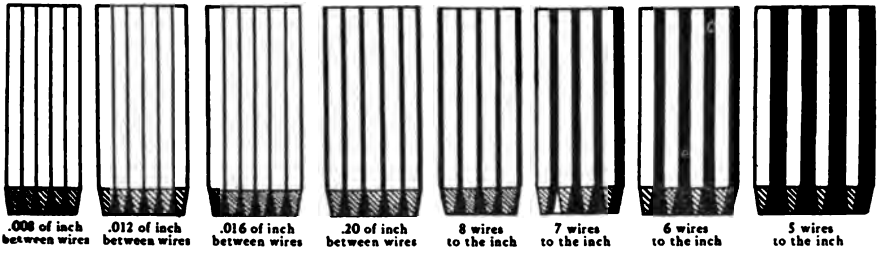
adapted for work with cable tools, as it will withstand hard usage and the driving and pulling often necessary with that system in drilling through unconsolidated sands.

The selection of the proper-sized mesh depends upon the physical character of the sand, viscosity of the oil and the gas pressure. The mesh should be of sufficient size to allow the maximum entry of sand and oil while excluding pebbles and small stones. This is best determined by practical experiment above ground. Other things being equal, a high-viscosity oil demands a coarser mesh than one of low viscosity. A screen made up of five wires to the inch (see Fig. 3) proved most efficient with a moderately fine sand carrying oil of 14° to 15° Bé. gravity. The erroneous belief that the function of screen casing is to prevent the entry of sand in oil wells, with the consequent avoidance of pumping troubles,



After Layne & Bowler Corp.

FIG. 3.—SHOWING DIFFERENT MESHES OF A WIRE-WOUND SCREEN.

leads to the selection of too fine a mesh in many instances. *The mesh should be too coarse rather than too fine.* Screen pipe came into use and was devised in connection with water wells and it was natural that the same rules were thought applicable when it was introduced in the oil fields.

PUMPING THE SAND AND OIL

Assuming that a well has been properly drilled, thoroughly washed, screen casing set, and that sand and oil are passing freely through the mesh, the most efficient device to bring the sand and oil to the surface is the plunger pump (Fig. 4). Although many wells flow naturally for a time or may be agitated by periodically moving an "agitator string," and others containing much water may be pumped with an air lift, the standard equipment for handling sand is the plunger pump. This device consists essentially of an outside cylinder or "working barrel" and a hollow plunger with upper and lower valves (Fig. 5). Although simple in principle, these pumps are very carefully made of the best material and ground to an exact fit, and their proper installation and operation require experience and knowledge in order to obtain the best results in

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FIG. 4.

FIG. 5.

FIG. 6.

Sectional View

FIG. 7.

For description, see foot of next page.

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the removal of sand. Fig. 4 shows a pump completely assembled and ready to be attached to the tubing. The working barrels are made in lengths from 42 to 60 in. and from 2 to 4 in. diameter. A common size is 3 by 60 in. for the "renewable" or liner pump with extension and top collars at both ends (Fig. 6). This is one of the best types of pumps for handling sand, as the upper portion of the plunger does not leave the barrel and therefore is not in contact with sand on its downward stroke. The standing valve barrel is carried on a 3 by 24-in. nipple which allows the plunger to pass below the lower end of the barrel on each stroke, thus retarding the entry of sand between plunger and barrel.

The Parker pump shown in detail in Fig. 7 is said to be particularly efficient in handling sand on account of the large area of the valve openings and the positive action of the valves.

Pump Operation

A properly installed and operated plunger pump will handle a surprising quantity of sand; in fact, it may be said that it is capable of pumping sand containing oil rather than oil containing sand. It is not at all unusual for one of these pumps to handle a mixture containing 50 per cent. of sand by volume. A careful record should be kept of the performance of the pump on each well: Depth at which it is pumping, date when new valve seats or balls were substituted for the old, when the barrel was renewed and on what date the tubing was lowered. The lasting qualities of a pump are directly dependent upon the quantity of sand produced and as this varies with each well the importance of keeping a record of pump performance is apparent. A barrel may require renewal every 6 weeks on one well while on another it shows but slight wear after 3 months' use.

The common practice of allowing a well to pump until the barrel becomes so worn that the production dwindles to a small stream or until some accident or breakage necessitates pulling should be condemned. Each pumping well should be studied and every effort made to maintain the production not only of oil but of sand.

The well should be tubed to such a depth that the maximum quantity of sand which the pump is capable of handling reaches the barrel at all times. This point can be determined only by experiment and gradual lowering of the tubing as sand production diminishes. Continued pumping of sand may save expense and labor of cleaning out "dead" sand with tools and bailer.

FIGS. 4 TO 7.—PLUNGER PUMP AND PARTS FOR RAISING SAND AND OIL.
FIG. 4.—PLUNGER AND VALVES IN WORKING BARREL.
FIG. 5.—PLUNGER WITH UPPER AND LOWER VALVES.
FIG. 6.—WORKING BARREL WITH LINERS AND EXTENSION NIPPLE TO STANDING VALVE BARREL.
FIG. 7.—PARKER PUMP.

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In order to determine the exact condition of the barrel and plunger it is essential that both be removed from the well, washed with distillate and then tested for fit and the absence of lateral play between the plunger and the barrel.

The common practice of removing only the rods and plunger for examination is to be condemned.

The pumping of sand is troublesome and means much work around the wells. For this reason there is a tendency to avoid it.

A long stroke is advisable for pumping wells producing sand and the wristpin should be carried in the "second hole."

If a valve becomes clogged with small gravel or sand it may often be freed by "shaking up the well" or running the pump at high speed for a few strokes. Twenty-five strokes to the minute is the usual pumping rate when the rods drop freely and prevents the sand from settling.

Hot Oiling

Methods of Overcoming a Decline in Production

After a well has produced sand for a time varying from months to years it gradually diminishes in output even though the well is tubed to bottom. In such cases several methods are used to "liven up" the well and increase production. Steam has been used in such wells and "swabbing" is commonly practiced, but the best method is probably the introduction of hot crude oil. Where this method fails it is undoubtedly caused by faulty casing, either screen or perforated, or the hot oil is improperly applied.

The best practice is to heat about 100 bbl. of crude almost to the boiling point and allow it to flow by gravity or pump it down the well casing. It is usually treated in a nearby tank with steam coils and although some operators introduce it through the tubing this practice has the disadvantage that the necessary stoppage of the pump may cause it to sand up.

The effect of the hot oil is to wash the screen or perforations and to lower the viscosity of the underground supply, causing both the oil and sand to flow more readily toward the well.

The hot oil need not remain in the well for more than ½ to ¾ hr. and is then pumped out together with large quantities of sand and "live" oil, oftentimes accompanied by considerable gas. It frequently happens that so much sand enters the well following this treatment that the pump is unable to handle it, in which case two or three joints of tubing are removed, the barrel afterward being gradually lowered as the sand becomes exhausted. After a few weeks the pump again reaches bottom, when another hot oiling is given.

There is no fixed rule as to the frequency of this treatment. This

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can be determined only by experiment and is largely dependent upon the age of the wells. In some cases it may be applied once a week with beneficial results while in others once a month is sufficient.

IV. QUANTITY OF SAND PRODUCED

This is a most interesting subject but, unfortunately, exact figures on the actual quantity of sand produced by flowing or pumping wells are unavailable.

In speaking of the sand produced by wells in Russia, Mr. Thompson[5] says: "The oil from fountains is commonly accompanied by an equal bulk of sand, large numbers of stones, and the liberation of millions of cubic feet of gas which becomes disengaged from the oil on its exit from the tube. A recent Bibi-Eibat spouter on plat No. 29 gave as much as 10,000 tons of oil and 10,000 tons of sand in a day, and in a few weeks yielded, in addition to several million poods of oil, no less than 1,700,000 cu. ft. (85,000 tons) of sand; an amount which will be better appreciated when it is realized that this quantity of material would raise the natural level of the ground on 1 dessiatine nearly 14 ft., or cover an acre of land to a depth of 40 ft."

The famous gushers of California and other loose-sand fields have expelled tremendous quantities of sand which in some cases have completely buried the engine house, belt house, and the lower panels of the derrick.

"In the old Sunset field wells that have a strong gas pressure produce more sand than any others in the State, it being estimated that sometimes as much as two-thirds of the gross yield of the wells is sand. One well alone produced over 110,000 cu. ft. of sand in about 4 years, and another has yielded almost as much in 2 years. The yield of sand gradually decreases with the age of the well, but at no time entirely ceases. These peculiar conditions—soft sand in great quantities accompanying the flow, heavy oil, and strong gas pressure—make the problem of well operation in this field difficult."[6]

Some pumping wells in the north Midway field produce sand at the rate of over 200,000 cu. ft. a year and careful measurements would probably show that this amount is actually produced in a year or even exceeded.

V. OIL SEPARATION AND SAND DISPOSAL

The separation of oil and sand, although more fully accomplished above ground, is one of the difficult problems connected with production from unconsolidated sands. With low-gravity, viscous oils and great

[5] A. B. Thompson: *The Oil Fields of Russia*, pp. 52–53. London, Crosby Lockwood and Son (1908).

[6] R. Arnold and V. R. Garfias: Methods of Oil Recovery in California, *U. S. Bureau of Mines, Technical Paper* 70, p. 22 (1914).

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volumes of sand, nothing has been found to take the place of the open-air "sump," which is simply an excavation made near the well with teams and scrapers. The sand collects in great piles in these sumps while the oil separates by gravity and is pumped from the lowest point in the depression. With heavy oil the loss through evaporation and percolation is slight, as it has been determined that it does not penetrate the ground for more than a few inches. Light oils, however, should not be allowed to flow into sumps, as the loss is undoubtedly great; but when accompanied by much sand other disposition is difficult. As the piles of sand are gradually freed of their oil content they become fairly dry and a crust is formed on their surface which will bear the weight of a man.

Many tanks, settling boxes and other devices have been designed for the separation of oil and sand but with little success. Most of these devices are entirely lacking in scientific principle and are carelessly constructed affairs designed by the workers in the fields and their failure is in large means due to these facts. An efficient means of separating light oils and sand can undoubtedly be devised and would prove of great benefit to the producer. Heavy oils being much more difficult to separate, of less market value, and of high viscosity, may be most economically handled in properly constructed sumps.

The great disadvantage of such excavations is the space they occupy, their unsightly appearance, and the huge deposits of sand which gradually surround the well. The removal of this sand finally becomes imperative when all available space for new sumps has been utilized, but its disposal is a problem.

The only use to which it could be put in the north Midway was road surfacing and although this is possibly a very good use it is a more or less expensive one. The method used is simple.

When a sump had been drained of its oil and the sand had become sufficiently dry, it was loaded upon wagons and distributed on the many dirt roads traversing the property. The roads were sometimes dragged to prepare them for the sand but this was frequently dispensed with. A layer of sand 2 in. thick was sufficient to surface the ordinary road, which was immediately opened for traffic in order that the sand might become packed. The roads in that section of the country were in very poor condition, being formed from use and never constructed or built. They were deep with dust and ruts and the beneficial results attained through the use of oil sand were truly remarkable. This sand as it comes from the sumps contains sufficient oil to stain the hands and serves as a binding material for the road surface. It soon packs into a durable layer of asphaltum-like hardness. It withstands the traffic of heavy automobile trucks and is easily maintained by occasional dressings of fresh sand in the summer when the heat softens the mixture.

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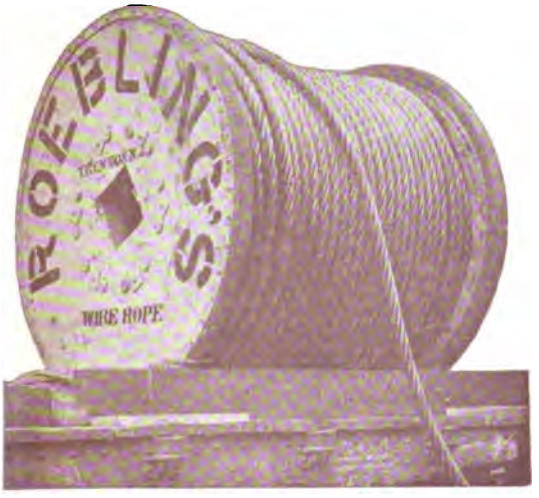

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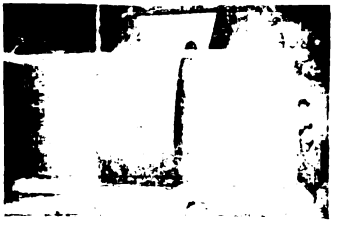

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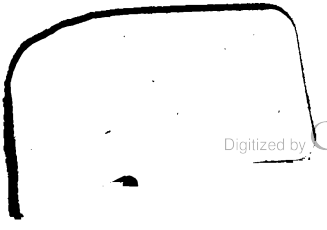
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